UNIV. OF
TORONTO
LIBRARY

DEPARTMENT OF MINING ENGINEERING
Library Number 1142
M
2

~~Section - 52~~
~~No - 554~~

Digitized by the Internet Archive
in 2011 with funding from
University of Toronto

http://www.archive.org/details/journalsouthafri10sout

Tech C

THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA.

VOLUME X.

JULY, 1909 — JUNE, 1910.

118608
21/9/11

JOHANNESBURG:
Published by the Society: 9, South African School of Mines Building, Eloff Street

TN
1
S66
v.10

CONTENTS OF VOLUME X.

PAGE

Papers and Notes Read Before the Society ... ... v.

List of Illustrations ... ... vii.

Officers and Council of the Society ... ... ix.

List of Members ... ... x-lvii

Journal No. 1.—July, 1909 1

„ „ 2.—August „ 41

„ „ 3.—September „ 81

„ „ 4.—October „ 121

„ „ 5.—November „ 161

„ „ 6.—December „ 197

„ „ 7.—January 1910 233

„ „ 8.—February „ 273

„ „ 9.—March „ 309

„ „ 10.—April „ 341

„ „ 11.—May „ 381

„ „ 12.—June „ 421

Index to *Journal* 477

PAPERS AND NOTES READ BEFORE THE SOCIETY.

PAGE
Inaugural Address, by A. McA. JOHNSTON (President) 3
Acid Washes, Experiments on the Assay of, Resulting from the Cyanide "Clean Up" by the Use of Bisulphate, by L. J. WILMOTH 136
Amalgam, Silver, Making of, by EDWARD H. CROGHAN 12
Barberton Goldfield, The, by ALEX. RICHARDSON 122
Bisulphate Solutions, Specific Gravity of, by THOS. DONALDSON 86
Black Sands, The Treatment of, by FRASER ALEXANDER 171
Corrosion, Notes on, with Special Reference to the Corrosion of Steel Winding Ropes, by M. THORNTON MURRAY 51
Cyanide, The Destruction of, by Dr. J. MOIR and JAMES GRAY 433
Cyanide Plant, A Sketch of the Small, as Erected and Worked in Rhodesia, by F. J. THOMAS ... 82
Cyanide Solutions and Slime Residue, The Assay of, Carrying Dissolved Gold, by A. WHITBY 131
Extractor, Rotary, for Precious Metals from Solution, by W. D'ARCY LLOYD and E. T. RAND ... 201
Grading Analyses and Their Application, by H. STADLER... 382
Iron Ore, Titaniferous, Some Experiments on Smelting, by Prof. G. H. STANLEY Part I. (Crucible Experiments) 162 Part II. (Furnace Experiments) . 345
Iron and Steel, Crystallization of, by A. McA. JOHNSTON (Inaugural Address) 3
Mining at the Redjang-Lebong Gold and Silver Mine, Sumatra, by H. PHILP 315
Nitro-Glycerine Explosives, Analysis of Gases from Burning, by W. CULLEN and D. W. GREIG. 90
Ore Slime, Treatment of, by ANDREW F. CROSSE 172
Precipitating Effects of Substances Containing Various Forms of Carbon and Cellulose on Cyanide Solutions Containing Gold and Silver, by A. J. CLARK and Dr. W. J. SHARWOOD 234
Precipitation of Gold from Cyanide Solutions by Carbon in Lime, Experiments on the, by E. H. CROGHAN 391
Prussic Acid, Test for, by Dr. J. MOIR 342
Pumps, Air Lift, for Underground Work, by B. C. TRAVERS SOLLY 2
Rand Mining, Further Notes on, by TOM JOHNSON 276
Rope Haulage, Endless, by H. G. KAY 198
Sand for Cyaniding, Continuous Collection of, by Dr. W. A. CALDECOTT 43
Sand Filling on the Witwatersrand, by EDGAR PAM 129
Slime, Accumulated, The Treatment of, and the Use of Filter Presses for Clarifying Slime Solution and By-Products, by J. D. O'HARA 342
Stamps, Heavy Gravitation, The Development of, by Dr. W. A. CALDECOTT 108
Tin Ores, The Assay of, by JAMES GRAY 312
Tube Mill Circuit and Classification, The, by G. O. SMART 282
Zinc, A Method for the Recovery of, from Solutions of Sulphate, by W. CULLEN and G. F. AYERS 87

LIST OF ILLUSTRATIONS.

PAGE

Plates illustrating the Inaugural Address on "The Crystallization of Iron and Steel," by A. McA. JOHNSTON ... ... to face p. 40

Plates and Diagrams illustrating "The Continuous Collection of Sand for Cyaniding, by Dr. W. A. CALDECOTT 44, 45, 46, 47, 48, 49, 50, 51, 52, 53

Plates illustrating "Notes on Corrosion, with Special Reference to the Corrosion of Steel Winding Ropes," by M. THORNTON MURRAY 57, 58, 59

Diagram illustrating A. A. Coaton's discussion on "Stope Measurements" 64

Diagrams illustrating reply to discussion on "Grading Assays and Grinding Efficiencies," by A. YATES 101, 102, 103

Diagrams illustrating "The Development of Heavy Gravitation Stamps," by Dr. W. A. CALDECOTT 111, 113

Diagrams illustrating "The Barberton Goldfield," by A. RICHARDSON 124, 125, 126, 127, 129, 130

Diagrams illustrating K. Tonnesen's discussion on "Stope Measurements" 141

Diagrams illustrating "Theory of the Settlement of Slime," by H. E. ASHLEY ... 149, 150

Plates illustrating "Some Experiments on Smelting Titaniferous Iron Ore," by Prof. G. H. STANLEY 164, 165, 169 170

Diagram illustrating "Treatment of Ore Slime," by ANDREW F. CROSSE 173

Diagram illustrating "The Treatment of Black Sands," by FRASER ALEXANDER 175

Diagrams illustrating "Endless Rope Haulage," by H. G. KAY 199, 200, 201

Diagram illustrating "A Rotary Extractor for Precious Metals from Solutions," by W. D'ARCY LLOYD and E. T. RAND 202

Diagrams illustrating C. O. SCHMITT's discussion on "The Development of Heavy Gravitation Stamps" 243, 246, 247, 248, 249, 250, 251, 252

Diagrams illustrating "Further Notes on Rand Mining," by TOM JOHNSON 277, 278, 279, 280, 282

Diagrams illustrating "The Tube Mill Circuit and Classification," by G. O. SMART 283, 284, 286

Diagram illustrating G. A. WEBB's discussion on "Endless Rope Haulage" 291

Diagram illustrating "Tube Mill Liners," by F. C. BROWN 297

Diagrams and Plate illustrating "Some Experiments on Smelting Titaniferous Iron Ore," by Prof. G. H. STANLEY 349

Plates and Diagrams illustrating reply to Discussion on "The Development of Heavy Gravitation Stamps," by Dr. W. A. CALDECOTT 362, 363, 364, 365, 368

Diagrams illustrating "Grading Analyses and Their Application," by H. STADLER 384, 388

Diagrams illustrating R. GASCOYNE's discussion on "Further Notes on Rand Mining" ... 395

Diagrams illustrating C. O. SCHMITT's discussion on "The Tube Mill Circuit and Classification 398, 399

Diagram illustrating J. H. JOHNSON's discussion on "The Treatment of Accumulated Slime, and the Use of Filter Presses for Clarifying Slime Solution and By-Products ... 404

Diagrams and Plate illustrating reply to discussion on "Stope Measurements," by O. TONNESEN 405, 406

Diagram illustrating "Smelting Gold and Silver Slime," by A. L. BLOMFIELD 406

Diagram illustrating "Treatment of Slime," by H. G. NICHOLS 410

Diagram illustrating "A Simple Safe Fuse Lighter" 413

Diagrams illustrating "Sand Filling on the Witwatersrand," by EDGAR PAM 430, 431

Diagrams illustrating "The Destruction of Cyanide," by Dr. J. MOIR and J. GRAY 434, 435, 436, 437

Diagrams illustrating J. E. THOMAS' discussion on "The Tube Mill Circuit and Classification" 453, 454

Plates illustrating A. J. R. ATKIN's discussion on "Grading Analyses and Their Application" ... 458

Diagrams illustrating "Measurement of Pulp and Tailing," by Dr. W. J. SHARWOOD ... 470

SUPPLEMENT TO THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA

LIBRARY
FEB 1[illegible] 1910
UNIVERSITY OF TORONTO

LIST OF MEMBERS

11TH NOVEMBER, 1909.

JOHANNESBURG:
R. G. McKOWEN & Co., Printers, 121, Fox Street.

1909.

SUMMARY OF MEMBERSHIP AT 11th NOVEMBER, 1909.

Hon. Members	23
Life Members	8
Ordinary Members	673
Associates	367
Students	43
Total	1,114

Members, Associates and Students are requested to notify the Secretary of any inaccuracies in the following List, and also to immediately advise him of any change in address, otherwise it will be impossible to guarantee the delivery of Notices or Journals.

LIST OF OFFICERS FOR THE YEAR ENDING JUNE, 1910.

Hon. President:

H.E. The Earl of Selborne, K.G., P.C., G.C.M.G.

Hon. Vice-Presidents:

J. W. S. Langerman M.L.A. (President, Transvaal Chamber of Mines).

Geo. Albu,
F. D. P. Chaplin, M.L.A.,
F. Eckstein,
Sir George Farrar, M.L.A.,
Sir J. P. FitzPatrick, M.L.A.,
J. N. de Jongh,
Hennen Jennings,
Lionel Phillips,
L. Reyersbach,
Harold F. Strange,
Hon. J. de Villiers, M.L.A.,
Minister of Mines.

Past-Presidents:

Andrew F. Crosse,
W. R. Feldtmann,
Charles Butters,
Dr. J. Loevy,
John R. Williams,
S. H. Pearce,
W. A. Caldecott,
William Cullen,
Edward H. Johnson,
John Yates.
R. G. Bevington.

President:

A. McA. Johnston.

Vice-Presidents:

James Moir.
W. R. Dowling.
C. B. Saner.

Ordinary Members of Council:

F. F. Alexander,
E. H. Croghan,
M. Cullen,
K. L. Graham,
E. J. Laschinger,
A. Richardson,
G. O. Smart,
G. H. Stanley,
J. E. Thomas,
A. Whitby,
H. A. White,
J. A. Wilkinson.

Corresponding Members of Council:

E. G. Banks.
T. L. Carter,
G. T. Holloway,
Walter Neal,
H. M. Leslie,
R. B. Pollitt,
G. M. Taylor,
Gerard W. Williams,
A. Yates.

Hon. Treasurer:

James Littlejohn.

Hon. Auditor:

Alex. Aiken, F.S.A.A., F.C.I.S.

Hon. Legal Advisers:

Messrs. van Hulsteyn, Feltham & Fry.

Editorial Committee:

A. McA. Johnston,
W. R. Dowling,
J. Moir,
A. Whitby.

Publications Committee:

F. F. Alexander,
W. R. Dowling,
K. L. Graham,
E. J. Laschinger,
A. Richardson,
C. B. Saner,
H. A. White,
J. A. Wilkinson.

Finance Committee:

R. G. Bevington,
J. Littlejohn,
C. B. Saner,
J. A. Wilkinson.

Secretary:

Fred. Rowland, Fellow, Chartered Inst. Secretaries.
9, Transvaal University College Building, Eloff Street, Johannesburg.

HON. MEMBERS.

His Excellency, The Earl of Selborne, KG., P.C., G.C.M.G.

Bauerman, Prof. Hilary, A.R.S.M., Hon. M.I.M.M., F.G.S. 14 Cavendish Road, Balham, London, S.W.

Bovey, Henry T., LL.D., M.I.C.E., D.C.L. F.R.S. Imperial College of Science and Technology, South Kensington, London, S.W.

Christy, Professor Samuel Benedict, M.I.M.M. University of California, Berkeley, California, U.S.A.

Crookes, Sir William, D.Sc., F.R.S. 7 Kensington Park Gardens, Notting Hill, London, W.

Eckstein, F. 1 London Wall Buildings, London, E. C.

Farrar, Sir George, Kt., D.S.O., M.L.A. P. O. Box 305, Johannesburg.

Fitzpatrick, Sir J. Percy, Kt., M.L.A. Hohenheim, Parktown, Johannesburg.

Hahn, Prof. P. D., Ph.D., M.A. South African College, Cape Town.

Haldane, John Scott, F.R.S., M.A., M.D. Cherwell, Oxford, England.

Jennings, Hennen, M.I.M.M. Barney House, 1626, Rhode Island Avenue, Washington, D.C., U.S.A.

MacArthur, James Stewart, M.I.M.M. 6, Broad Street Place, London, E.C.

Phillips, Lionel. P. O. Box 149, Johannesburg.

Redmayne, Richard A. S., M.Sc., M.I.M.M., M.I.M.E., F.G.S. Whiteholme, Edgbaston, Birmingham.

Richards, Professor R. H., S.B , LL.D. Mass. Institute of Technology, Boston, Mass., U.S.A.

Rose, T. Kirke, D.Sc., A.R.S.M., M.I.M.M. The Royal Mint, London, E.

Villiers, Hon. J. de, M.L.A. Minister of Mines, Pretoria.

The President, American Institute of Mining Engineers. 99, John Street, New York, U.S.A.

The President, Institution of Mining Engineers. Albany Chambers, 39 Victoria Street, London, S. W.

The President, Institution of Mining and Metallurgy. Salisbury House, London, E.C.

The President, Mining and Metallurgical Society of America, New York, U.S.A.

The President, The Chemical Society. Burlington House, London, W.

The President, The Society of Chemical Industry. 9 Bridge Street, Westminster, London, S.W.

MEMBERS.

Corrected to November 11, 1909.

Year of Election.

1904 Acutt, Sidney. Barrett Gold Mining Co., Ltd., Kaapsche Hoop. Mine Manager.

1894 Adair, Alfred. 16, Overbeek Street, Troyeville, Johannesburg. Chemist and Metallurgist.

1909 Adams, Alfred Charles. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1909 (Assoc. 1902) Adams, Arthur E. Messrs. Lowenstein, Adams & Co., P. O. Box 644, Johannesburg. Pharmaceutical Chemist.

1909 Adams, Ernest Lawrence, A.I.M.M. Sabi Mine, Belingwe, Rhodesia. Mining Engineer.

1909 Adams, Hirzel Allen, S.I.M.M. Village Main Reef G. M. Co , Ltd., P. O. Box 1091, Johannesburg. Assistant Surveyor.

1904 Adamson, Robert, M.I.M.E. Susanna Mine, Gwanda, Rhodesia. Mine Manager.

1906 Adendorff, John Ernest R. Randfontein Estates & G. M. Co., Ltd., P. O. Box 39, Randfontein. Analytical Chemist.

1902 Adler, Heinrich Wolff, M.E. (Cape). Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Mine Surveyor.

1898 **Aiken**, Alexander, F.S.A.A., F.C.I.S. (*Hon. Auditor*). P. O. Box 2636, Johannesburg. Accountant and Auditor.

1903 Aimetti, Stefano C. B. New Heriot G. M. Co., Ltd., P. O. Box 10, Cleveland. General Manager.

1898 Albu, Georg. General Mining and Finance Corporation, Ltd., P. O. Box 1242, Johannesburg. Director, Mining Companies.

1897 **Alexander**, Fraser Frederick (*Member of Council*). Crown Mines, Ltd., P. O. Box 102, Fordsburg. Metallurgist and Reduction Works Manager.

1903 Alexander, Harry John. Robinson Gold Mining Co., Ltd., P. O. Box 1024, Johannesburg. Cyanide Manager.

1903 Allen, Francis. East Rand Proprietary Mines, Ltd., P. O. Box 30, East Rand. Mill Manager.

1906 Allen, Herbert Edmund, A.O.S.M.(N.Z.) c/o S. Richards, Esq., Jumpers G. M. Co., Ltd. Cleveland. Assayer.

1904 Allen, John Andrew. Castano Nuevo, Prov. San Juan, Argentine Republic, South America. Assayer.

1909 Allen, Robert, M.A., B.Sc., M.I.M.M. Consolidated Goldfields of S. A., Ltd , P. O. Box 1167, Johannesburg. Metallurgical Engineer.

1907 Allen, William Howard, M.A. P. O. Box 1080, Johannesburg. Chemist.

1894 Alston, Robert A. 14, Margaret Street, Greenock, Scotland. Mining Engineer.

1902 Anderson, George Smith Strachan. East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Assayer.

1909 Anderson, Percy. Durban-Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Surveyor.

1905 Anderson, Peter von Maltitz, B.Sc. New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Mining Engineer and Assistant Manager.

1903 Anderson, William Thomas. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mine Manager.

1897 Andreoli, Gabriel. P. O. Box 2012, Johannesburg. Electro-Metallurgist.

1907 Angus, William Hay. Foreman Amalgamator. *(Address wanted.)*

1907 Argall, Philip, M.I.M.M. 730, Majestic Building, Denver, Colorado, U.S.A. Consulting Mining Engineer and Metallurgist.

Year of Election.

1904 Arkell, Dennis J. The Abels Kop G. M. Co., Ltd., P. O. Schweizer Reneke. General Manager.

1898 Atkin, Austin J. R., M.Aus.I.M.E. East Rand Proprietary Mines, Ltd., P. O. Box 134, East Rand. Metallurgist.

1905 Atkin, Randal Walter Roberts. Knights Deep, Ltd., P. O. Box 143, Germiston. Assistant Smelter.

1905 Atkinson Charles Edwards. Globe and Phœnix Mine, Queque, Rhodesia. Battery Manager.

1903 Atkinson, Robert Charles, M.Am.I.M.E., M. I. Marine E. Messrs. H. Eckstein & Co, P. O. Box 149, Johannesburg. Assistant Consulting Mechanical Engineer.

1904 Aulsebrook, Ernest Edward. Mill Manager. (*Address wanted.*)

1905 Auret, Abraham Anthony. P. O. Box 838, Johannesburg.

1908 (Assoc. 1907) Avent, Alec. Robinson Deep G. M. Co., Ltd., P. O. Box 1188, Johannesburg. Cyanider.

1902 Avesson, Rudolf. 17c, Fabrikstrasse, Reval, Estonia, Russia. Mining Engineer.

1904 Bacon, Arthur Drax. Randfontein Central G.M. Co., Ltd., Randfontein. Mine Manager.

1895 Bagshaw, J. F., B.Sc. (Edin.). Magnesite Mines of S. A., Ltd., Malelane, Transvaal. Chemical Engineer and Manager.

1899 Banks, Edwin Gripper, M.I.M.M., M.Am.I.M.E. (*Corresponding Member of Council*). Waihi G. M. Co., Ltd., P. O. Box 4, Waihi, Auckland, New Zealand. Metallurgist.

1909 Barclay, George Moore. Knights Deep, Ltd, P. O. Box 143, Germiston. Mining Engineer.

1908 Barratt, Rowland Lorraine. Bushtick Mine, per private bag, Bulawayo. Assayer and Surveyor.

1898 (Assoc. 1898) Barrett, Percy. Van Ryn G. M. Estate, Ltd., P. O. Box 22, Benoni. Cyanide Manager.

1896 Barrie, D. McLaurin. New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Technical Chemist & Cyanide Manager.

1903 Barron, William Ernest. Van Ryn Gold Mine Estates, Ltd., P. O. Box 22, Benoni. Mine Surveyor.

1905 Bartlett, Sydney Gordon, Stud.I.M.M. York G. M. Syndicate, Ltd., P. O. Box 54, Krugersdorp. Mine Captain.

1903 Bateman, Edward Latrobe. Allis-Chalmers Co., Ltd., P. O. Box 6659, Johannesburg. Mechanical Engineer.

1897 Baumann, Max, M.E. Elandshoogte Mine, P. O. Alkmaar. Mining Engineer.

1905 Bawden, Ernest Robson. Poldice House, St. Day, Scorrier, Cornwall. Miner.

1908 (Assoc. 1903) Bawden, Frederick Alexander, A.I.M.M. Knights Deep, Ltd., P. O. Box 143, Germiston. Mining Engineer.

1903 Bay, Bernhard, Ph.D. P. O. Box 5513, Johannesburg. Chemist.

1904 Bayldon, Harold Cresswell, M.I.M.M. c/o Troitzk Goldfields, Ltd., 20 Copthall Avenue, London, E.C. Mine Manager.

1905 Bayley, Reginald Francis Gordon, A.R.S.M., A.I.M.M. M.M., Redjang Lebong, Lebong Donok, Benkoelen, Sumatra, Dutch East Indies. Cyanider.

1905 Beaton, Samuel. P. O. Box 6202, Johannesburg. Mine Manager.

1908 Beatty, George Hewitt. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Mine Surveyor.

1903 Beatty, John William Scott. Village Deep, Ltd., P. O. Box 1115, Johannesburg. Mine Sampler.

1903 Beaver, William, A.I.M.M. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Mill Manager.

1902 Behr, Hans Charles, M.I.M.M., M.I.Mech.E. Consolidated Goldfields of South Africa, Ltd., P. O. Box 1167, Johannesburg. Consulting Mechanical Engineer.

1906 Bell, H. D. York G. M. Syndicate, Ltd., P. O. Box 54, Krugersdorp. Assayer.

Year of Election.

1908 Bennett, Randolph White. Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Assayer.

1904 Bennett, William Frederick. New Rietfontein Estates, Ltd., P. O. Rietfontein Mines. Shift Boss.

1904 Bentley, Alfred Mulock. Bechuanaland Exploration Co., Ltd., Broken Hill, N. W. Rhodesia. Resident Secretary.

1909 Bentley, Frederick William. Robinson Deep G. M. Co., Ltd., P. O. Box 4648, Johannesburg. Electrical Engineer.

1902 Betty, William Kemmis, M.I.M.M. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Assistant Consulting Metallurgist.

1894 BEVINGTON, Richard George, M.I.M.M., M.Am.I.M.E. (*Past President*). Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Metallurgist and Mill Manager.

1909 Bisset, Arthur Charles. East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand. Surveyor.

1903 Black, James. Bucks Reef Mine, per private bag, Colleen Bawn Siding, Rhodesia. Manager.

1904 Blackie, John Bwana M'kubwa Mine, Ndola, N. W. Rhodesia. Chemist and Metallurgist.

1903 Blackmore, James Gairdner, A.S.A.S.M. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Metallurgical Chemist and Assayer.

1904 Blackwell, Charles Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Mechanical Engineer.

1908 Blaeser, Cornelius Alidus. Assistant Assayer. *(Address wanted.)*

1907 Blaine, Gilbert Rodney. Kokstad, C.C. Mining Engineer.

1908 Blau, Max. Mijnbouw Maatschappij Siman, Lebong Tandai, *via* Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1902 Bleloch, William Edwin. P. O. Box 3692, Johannesburg. Mining Company Manager.

1908 Blewett, Edward. West Nicholson, Rhodesia. Chemist and Cyanide Operator.
(Assoc. 1906)

1903 Blume, Eugen E. A., A.I.M.M. P. O Box 908, Johannesburg. Metallurgist.

1905 Blundun, Basil Vivian. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Amalgamator.

1905 Boer, Ernest Charles. Maluti Reef, near Colleen Bawn, Gwanda, Rhodesia. Mechanical Engineer.

1908 Boright, Sherman H. Great Eastern Collieries, Ltd., P. O. Box 5, Springs. Mining Engineer.

1902 Bosqui, Francis L. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Metallurgical Engineer.

1909 Bowen, H. O. Robinson G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Mining Engineer.

1909 Bowness, Arthur James. Ferreira Deep Ltd., P. O. Box 5977, Johannesburg. Cyanide Foreman.

1906 Boyd, Alex. Geduld Proprietary Mines, Ltd., P. O. Box 11, Springs. Cyanide Foreman.

1898 Brakhan, Amandus P. O. Box 1156, Johannesburg. Managing Director, Ad. Goerz & Co., Ltd.

1906 Brett, Henry Tyndall. P. O. Box 135, Bulawayo. Mining Engineer and Metallurgist.

1905 Bristol, Frerick Amof., B.A. Consolidated Goldfields of S. A., Ltd., P. O. Box 1167, Johannesburg. Mining Engineer.

1903 Bristol, John James, B.Sc. Reno, Nevada, U.S.A. Mine Sampler.

1902 Brooks, Edwin. Golden Valley, Gatooma, Rhodesia. Assayer.

1902 Broom, William. P. O. Box 890, Johannesburg. Analytical Chemist and Assayer.
(Assoc. 1897)

1905 Brown, George Duncan. French Bobs Mine, Louw's Creek, Barberton. Mine Manager.

1904 Brown, James. French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei. Mill Manager.

1904 Brown, Dr. Johnstone. P. O. Box 94, Jeppestown. Medical Practitioner.

1904 Browne, Ernest. Barrett G. M. Co., Ltd., Kaapsche Hoop, Transvaal. Cyanide Manager.

Year of Election.

1908 Browning, Edward. Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Mining Engineer.

1905 Brunton, Walter George. Premier Diamond Mining Co., Ltd., P. O. Box 43, Premier Mine. Manager, Sorting Dept.

1898 Brunton, Walter O. Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Cyanide Manager.

1902 Bruun, Andreas Frederic. Standard Bank of South Africa, Ltd., P. O. Box 1031, Johannesburg. Assayer.

1898 Burnett, Digby V., M.I.M.M. Piggs Peak Development Co., Ltd., Swazieland, *via* Barberton. General Manager.

1902 Burnett, George Lindsay, A.I.M.M. Wolhuter G. M., Ltd., P. O. Box 1160, Johannesburg. Cyanide Manager.

1909 Busch, Gustave Edgar, M.I.M.M. New Lisbon Berlyn, Limited, Pilgrims Rest. Mining Engineer.

1906 Butler, Joseph John. May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston. Chief Assayer.

1909 Butler, William Mahony. Simmer Deep, Ltd., P. O. Box 50, Germiston. Cyanider.

1894 BUTTERS, Charles, B.Ph., M.I.M.M., A.M.I.C.E., M.Am.I.M.E., M.I.M.E *(Past-President).* 333, Kearney Street, San Francisco, California, U.S.A. (Private address, Rose Lawn, Berkeley, California).

1909 Buuren, William van. Crown Mines, Ltd., P. O. Box 102, Fordsburg. Mill Foreman.

1906 Byerley, Benjamin George. Bulawayo Club, Bulawayo. Mining Engineer.

1894 CALDECOTT, William Arthur. B.A., F.C.S., M.I.M.M., M.Am.I.M.E. *(Past-President).* Consolidated Goldfields of South Africa, P. O. Box 67, Johannesburg. Consulting Metallurgist.

1908 Calder, William Lawrence, Abercorn Banket G. M. Co., Ltd., Shamva, S. Rhodesia. Mining Engineer.

1903 Cameron, William McCallum, B.Sc., E.M., M.Am.I.M.E. Messrs. A. Goerz & Co., Ltd., P. O. Box 1156, Johannesburg. Consulting Mining Engineer.

1906 Cannon, Arthur Beaufoy, A.I.M.M. Barberton. Cyanide and Battery Manager.

1909 Cantor, Bertie Charles Moxon. Blanket Mine, Gwanda, South Rhodesia. Mining Engineer.

1909 Caris, Sydney. Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

1908 Carr, William. Salisbury and Jubilee Mines, P. O. Box 1122, Johannesburg. Mine Manager.

1895 Carrick, J. T., Ph.D., M.I.M.M. Consulting Geologist.

1902 Carruthers, James Thomas. 670, High Street, Brixton, Johannesburg. Cyanider.

1903 (Assoc. 1898) Carter, Benjamin Palmer, E.M. Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Mine Manager.

1898 Carter, George. Nourse Mines, Ltd., P. O. Box 32, Denver. Mining Engineer and Surveyor.

1898 (Assoc. 1897) CARTER, Thomas Lane, A.I.M.M. (*Corresponding Member of Council*). Hammond, Louisiana, U.S.A. Mining Engineer.

1903 Catlin, Robert Mayo, B.Sc., E.M., C.E., M.I.M.M. New Jersey Zinc Co.,Franklin Furnace, New Jersey, U.S.A. Mining Engineer.

1899 Cazalet, Percy. Nourse Mines, Limited, P. O. Box 32, Denver. Mining Engineer and Mine Manager.

1908 Cellier, Jacobus Stephanus, M.E. Nigel Deep, Ltd., P. O. Box 50, Nigel. Mining Engineer and Government Land Surveyor.

1896 Chambers, Arthur Leo, F.R.C.I., M.I.M.M. Subeni Coppers (Natal), Subeni P. O., *via* Vryheid. Mining Engineer and General Manager.

1909 Chandler, Walter Mark. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Mining Engineer.

1905 Chaplin, F. D. P., M.L.A. Consolidated Goldfields of South Africa, Ltd., P. O. Box 67, Johannesburg. Joint Manager.

Year of Election.

1906 Chapman, Frederick Thomas. Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Assayer.

1904 Chew, Beverley. Moodies Fortuna G. M. Co., Ltd., P. O. Box 124, Barberton. Tributor.

1904 Chew, Robert Withers. c/o African Banking Corporation, Ltd., London, E.C. Mill Manager.

1909 (Assoc. 1905) Chilton, James. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Miner.

1905 Christopherson, Douglas. Consolidated Goldfields of South Africa, Ltd., P.O. Box 67, Johannesburg. Assistant Joint Manager.

1902 Christopherson, Evelyn Herbert. West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp. Mining Engineer.

1903 Cindel, Frederick Walter. Crown Mines, Ltd., P. O. Box 1124, Johannesburg. Mill Foreman.

1903 Clarkson, Ralph, New Rietfontein Estate G. M. Ltd., P. O. Rietfontein Mines. Mine Manager.

1902 Claudet, Arthur Crozier, A.R S M., F.I.C., M.I.M.M. 6 and 7, Coleman Street, London, E.C. Assayer.

1907 Coaton, Arthur Allen. Crown Mines, Ltd., P. O. Box 102, Fordsburg. Sampler.

1908 Coats, Joseph. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Chief Sampler.

1904 Cobbett, William Geoffrey Pitt. Rand Mines, Ltd., P. O. Box 1056, Johannesburg. Surveyor.

1908 Cockburn, Paul Alexander. Gaika Mine, Queque, Rhodesia. Mine Manager.

1904 Cohen, Lionel, F.R.G.S. P. O. Box 2945, Johannesburg. Stockbroker and Director of Mining Companies.

1909 Coll, Daniel. Jumpers G. M. Co., Ltd., Cleveland. Cyanider.

1903 Collings, Burton Ireland, A.I.M.M. 15 Harvard Court, West Hampstead, London, N.W., England. Mining Engineer.

1902 Colquhoun, Ludovic. B.S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Chemist.

1908 Colvile, Cecil Augustus Bradshaw. P. O. Box 158, Salisbury, Rhodesia. Mining Engineer.

1905 Cooke, Reginald Cyril Herbert, A.I.M.M. 17, St. Edmund's Terrace, Regent's Park, London, N.W. Metallurgical Chemist.

1905 Coombe, Martin Henry. Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Mine Manager.

1909 Corder, Cecil Arthur. East Rand Proprietary Mines, Ltd , P. O. Box 152, East Rand. Mining Engineer.

1903 Corstorphine, George Steuart, Ph.D., F.G.S. National Bank Buildings, Simmonds Street, P. O. Box 1758, Johannesburg. Consulting Geologist.

1904 Cotter, Arthur John. General Mining and Finance Corporation, Ltd., P. O. Box 1242, Johannesburg. Mine Auditor.

1908 Coulter, William McCarrison. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Foreman.

1904 Crank, Albert Filmore, M.E. 334 Pacific Electric Building, Los Angeles, California, U.S.A. Mechanical Engineer.

1908 Creed, John Percy. Village Main Reef G. M. Co., Ltd., 157a, Church Street, Turffontein. Cyanider.

1898 Creswell, F. H. P., A.R.S.M., A.I.M.M. P. O. Box 6499, Johannesburg. Mining Engineer.

1904 (Assoc. 1903) CROGHAN, Edward Henry *(Member of Council)*, The Laboratory, Simmonds Street, P. O. Box 2187, Johannesburg. Analytical Chemist.

1909 Cropper, Cecil Howe, B.E. (Sydney), A.I.M.M. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mining Engineer.

1894 CROSSE, Andrew Frederick, M.I.M.M. (*Past-President*). P. O. Box 598, Johannesburg. Consulting Chemist and Metallurgist.

Year of Election.

1908 CULLEN, Matthew M.I.M.E. (*Member of Council*). Clydesdale Colliery, Springs. Colliery Manager.

1902 CULLEN, William (*Past-President*), British South African Explosives Co., Ltd., Dynamite Factory, Modderfontein. Chemist and General Works Manager.

1908 Cumings, Charles Ernest George. P. O. Box 305, Bulawayo. Engineer.

1907 Cunnack, James T. (*Address wanted.*) Assayer.

1907 Cunningham, William. c/o Bechuanaland Trading Co., Salisbury, Rhodesia. Representative, Nobel's Explosives Co., Ltd.

1908 Curnow, Robert. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. General Manager.

1894 Curtis, J. S. Natal Buildings, Commissioner Street, P. O. Box 73, Johannesburg. Mining Engineer.

1909 Cusworth, William Henry. New Lisbon Berlyn, Ltd., Pilgrims Rest. Construction Works Foreman.

1904 Dalrymple, Hon. William, M.L.C. Anglo-French Exploration Co., Ltd, P. O. Box 2927, Johannesburg. Director of Companies.

1909 Daniel, James. Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Surveyor.

1894 Darling, George A., M.I.M.M. P. O. Box 3720, Johannesburg. Chemist and Cyanide Manager.

1908 Darrow, Wilton Elmore. Sutter Creek, Amador County, California, U.S.A. Gold Metallurgist.

1906 Davies, Edward. Yankee Doodle G. M. Co., Selukwe, Rhodesia. Surveyor and Assayer.

1906 Davis, Carl Raymond, E.M., A.I.M.E. Lancaster G. M. Co., Ltd., P. O. Box 347, Krugersdorp. General Manager.

1906 Dawbarn, Oscar S., A.I.M.M. Sungei Besi Mines, Kuala Lumpur, Federal Malay States. Experimental Chemist.

1904 Dawe, William Henry. Messrs. S. Neumann & Co., P. O. Box 485, Johannesburg. Manager.

1904 Dinwoodie, James Herbert, F.C.S. 71, Bree Street, P. O. Box 466, Johannesburg. Pharmaceutical Chemist.

1902 Dixon, Clement, M.I.M.M., M.Aus.I.M.E. General Manager, Anglo-French Matabeleland Co., Ltd., P. O. Box 305, Bulawayo, Rhodesia. Consulting Mining Engineer, and Representative, Messrs. Farrar Bros.

1909 Dodd, Herbert. City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Mine Captain.

1903 Dodds, James Bowmaker Kerr, M.I.M.M. P. O. Box 922, Johannesburg. Mill Manager.

1909 Donaldson, Thomas. B.S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Technical Chemist.

1898 Douglas, Robert Stuart. Consolidated Main Reef Mines and Estates, Ltd., P. O. Box 2, Maraisburg. Assayer.

1905 Douglass, Ross Earle, A.I.M.M. 2330 La Mirada Avenue, Los Angeles, Cal., U.S.A. Mining Engineer.

1908 Dowdle, Phillip John. East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand. Cyanide Foreman.

1894 DOWLING, Walford Robert (*Vice-President*). Knights Deep, Ltd., P. O. Box 143, Germiston. Reduction Works Manager.

1909 Dowsett, Rowland Bertram. New Goch G. M., Ltd., P. O. Box 1096, Johannesburg. Cyanide Manager.

1909 Doyle, Michael Joseph. Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

1897 Drake, Francis, M.I.M.M., M.Am.I.M.E. 310, Kohl Buildings, San Francisco, California, U.S.A. Consulting Mining Engineer.

Year of Election.

1897 Drought, James J., F.C.S , A.I.M.M., F.R.C.I. Mt. Landiani Estate, Molo, *via* Mombassa, British East Africa.

1907 Dunning, Frank George. Knights Deep, Ltd., P. O. Box 143, Germiston. Mining Engineer.

1896 Durant, Henry Thomas, F.C.S , M.I.M.M., M.Am.I.M.E. Consulting Metallurgist. (*Address wanted.*)

1897 Dures, Robert. Wanderer G. M. Co., Ltd., Selukwe, Rhodesia. Assayer.

1908 Dyer, Sidney Charles, A.I.M.M. P. O. Box 19, Bulawayo, Rhodesia. Mining Engineer.

1896 Dykes, James, Jr. 11 Royal Exchange Square, Glasgow. Mine Surveyor and Assayer.

1903 Eaton, James Murdoch, A.I.M.M., M.Aust.I.M.E. Etna G. M. Co., Hartley, Rhodesia. Manager.

1908 (Assoc. 1903) Edwards, Albert Llewellyn. P. O. Box 727, Johannesburg. Assayer and Cyanide Manager.

1904 Edwards, Peter Livingstone. Rose Deep, Ltd., P. O. Box 6, Germiston. Mine Secretary.

1903 Eisenstaedter, John Ernest. New Transvaal Chemical Co., Ltd., Delmore. Metallurgist.

1903 Elmore, Frank Edward, M.I.M.M., A.M.I.E.E. 72, Gloucester Terrace, Hyde Park, London, W. Electro-Metallurgist.

1905 Elphinstone, Edmund. Transvaal Gold Mining Estates, Ltd., P. O. Box 42, Pilgrims Rest. Assayer.

1898 Erskine, James Kerr, F.C.S. New Kleinfontein Co., Ltd., P. O. Box 88, Benoni. Analytical Chemist and Assayer.

1906 Evans, Alan Davies, M.I.M.M. Highmead, Llanybyther, South Wales. Mining Engineer.

1907 Evans, A. Winter. City Deep, Ltd., P. O. Box 5492, Johannesburg. Assistant Surveyor.

1905 Evans, George Stephen, A.I M.M. c/o J. Harris, Alma Cottage, West Horsley, near Leatherhead Surrey. Mine Manager.

1898 Evans, Lewis, A.I.M.M. Rand Club, P. O. Box 1032, Johannesburg. Mining Engineer.

1907 (Assoc. 1904) Evans, Thomas. Knights Deep, Ltd , P. O. Box 143, Germiston. Assayer.

1905 Evans, William M. Clonlara, Harrietville, Victoria, Australia. Mining Contractor.

1894 Feldtmann, W. R., M.I.M.M., M.Am I.M.E. (*Past-President*). 132, Salisbury House, London Wall, London, E.C. Consulting Mining Engineer and Metallurgist.

1909 Ferguson, James Frederick. P. O. Box 2309, Johannesburg. Metallurgist.

1903 Fergusson, James. "The Cedars," High Wycombe, Bucks, England. Director of Companies.

1903 Fergusson, Malcolm. P. O. Box 164, Germiston. Deputy Inspector of Mines.

1909 Ferris, Guy Grindley. Ferreira Deep, Ltd., P. O Box 1056, Johannesburg. Smelter.

1902 Ferris, Vivian Grindley. Consolidated Gold Fields of S.A , Ltd., P. O. Box 1167, Johannesburg. Surveyor.

1908 Filet, Gerrit. Redjang Lebong, Benkoelen, Sumatra, Dutch East Indies. Amalgamator.

1907 Fitch, Frank. c/o Ironton School, Denver, Colorado, U.S.A. Cyanide Superintendent.

1902 Fisher, Joseph Simpson. Office of Inspector of Mines, P. O. Box 164, Germiston. Inspector of Mines.

1898 Fleischer, Spencer Charles William. Van Ryn G. M. Estate, Ltd., P. O. Box 22, Benoni. Mill Manager.

1895 Fleming, John, M. A. Norman House, Florida. Consulting Mining Engineer.

1908 (Assoc. 1907) Foran, John Henry. M.M. Redjang Lebong, Lebong Donok, Benkoelen, Sumatra, Dutch East Indies. Assayer.

1909 Ford, F. C. P. O., Langlaagte. Cyanider.

1907 Ford, Milverton, Aurora West United G. M. Co., P. O. Box 26, Maraisburg. Mining Engineer.

Year of Election.

1904 Ford, Stanley H., A.R.S.M., M.I.M.M., M.I.M.E. 5 Osborne Terrace, Ashley Down Road, Bristol, England. Mining Engineer.

1909 Fox, George C. Messrs. Lewis & Marks, P. O. Box 2567, Johannesburg. Consulting Mechanical Engineer.

1898 Fraser, Archibald. French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei. Assayer.

1906 Frerichs, Jacob A. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Mechanical Engineer.

1903 Frey, Frederick Charles, B.Sc. Simmer Deep, Ltd., P. O. Box 178, Germiston Mine Manager.

1908 Furner, Edward Arthur. Crown Mines Ltd., P. O. Box 1145, Johannesburg. Mill Manager.

1903 Galbreath, Neil Macphail, A.I.M.M. 20, Union Street, Greenock, Scotland. Metallurgical Chemist and Cyanider.

1903 Gardiner, Cecil Trevor. Cinderella Deep, Ltd , P. O. Box 75, Boksburg. Assayer.

1894 Garny, Fritz, Ph.D. Lancaster West G. M. Co., P. O. Box 360, Krugersdorp. Cyanide Manager and Assayer.

1906 Garthwaite, Edwin Hatfield, M.I.M.M , M.Am.I.M.E. Oakland Bank of Savings, Oakland, California, U.S.A. Consulting Mining Engineer.

1894 Gaze, James. P. O. Box 3788, Johannesburg. Cyanide Manager.

1902 Gazzam, Joseph P., M.I.M.M. (*Life Member*). 5027, Westminster Place, St. Louis, Mo., U.S.A. Mining Engineer.

1904 Gerlings, H. 60, Derby Road, Bertrams, Johannesburg. Technical Chemist and Assayer.

1906 Giblin, Norman Ernest. c/o L. K. Ward, Geological Survey, Launceston, Tasmania. Assayer.

1909 Gibson, Arthur. Knights Deep, Ltd., P. O. Box 143, Germiston. Smelter.

1903 Gibson, James Richard. Standard Bank of South Africa, Ltd., P. O. Box 222, Bulawayo, Rhodesia. Chemist and Assayer.

1903 Gillies, Albert Sydney, Messrs. Lennon, Limited, P. O. Box 2187, Johannesburg. Chemist and Assayer.

1907 Gilmore, Arthur David. Robinson Deep G. M. Co., Ltd., P. O. Box 2584, Johannesburg. Cyanider.

1908 Glyn, C Glen Deep, Ltd , P. O. Box 184, Germiston. Mine Manager.

1905 Goodwin, Grantham. West Rand Central G. M. Co., Ltd., P. O. Box 1, Krugersdorp. Mine Secretary.

1906 Goodwin, Harry. West Rand Central G. M. Co., Ltd , P. O. Box 1, Krugersdorp. Cyanide Manager.

1904 Gordon, William Shirvinton. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Assayer.

1906 Gosney, Percy. Crocodile Reef, Essexvale, Rhodesia. Mill Manager.

1895 GRAHAM, Kenneth Lancaster, M.I.M.M. (*Member of Council*), Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Metallurgist and Cyanide Manager.

1909 Grammel, Paul. W. Sumatra Mijnen Syndikat, Fort de Kock, W. Sumatra, Dutch East Indies. Mining Engineer.

1906 Grant, James. Simmer & Jack Proprietary Mines, Limited, P. O. Box 192, Germiston.

1904 Gray, James, F.I.C. P. O. Box 3427, Johannesburg. Analytical Chemist and Assayer.

1907 Greene, Francis Mackenzie. East Gwanda Mines, Limited, West Nicholson, Rhodesia. General Manager.

1907 Greer, James Courtney, Jr. c/o W. and G. Elliott, Bank of New Zealand Buildings, Auckland, New Zealand. Chemist.

1903 Griffiths, André Pierre, Wh.Sch., A.R.S.M., A.M.I.C.E., M.I.M.M., M.Am.I.M.E. 781/2, Salisbury House, London, E.C. Mining Engineer.

Year of Election.

1903 Griffiths, Harry Denis, Wh.Sch., A.R.S.M., A.M.I.C.E., M.I.Mech.E., M.I.M.M., M.Am.I.M.E., c/o Standard Bank of S.A., 10 Clements Lane, London, E.C. Consulting Mining and Mechanical Engineer.

1905 Griffiths, William Price. Transvaal G. M. Estates, Ltd., P. O. Box 12, Pilgrims Rest. Sampler.

1899 Grix, Harry Richard. Roodepoort U. M. R. Gold Mining Co., P. O. Box 115, Roodepoort. Mill Manager.

1906 Guthrie, Frederick Gysbert. Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Assayer.

1906 Haddon, Theodore. Globe and Phœnix G. M. Co., Ltd., Que Que, Rhodesia. General Manager.

1905 Hahn, Werner Arnold Aneck, B.Sc. South West Africa Company, Grootfontein, *via* Swakopmund, German South West Africa. Mining Engineer.

1909 Hailey, Rupert Clode, A.I.M.M. Poleleh, North Celebes, Netherlands India. Mining Engineer.

1894 Hall, Robert E. Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg Metallurgist and Cyanide Manager.

1904 Hallimond, William Tasker, M.I.M.E. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Mining Engineer.

1897 Hambly, Frank N. 355 Commissioner Street, Fairview, Johannesburg. Mine Manager. Metallurgist and Chemist.

1906 Hamill, Joseph Holmes. Nkandhla, Zululand. Mine Sampler.

1903 Hamilton, William. Glen Deep, Ltd., P. O. Box 184, Germiston. Assayer.

1897 Hamilton, William Laurie. P. O. Box 678, Johannesburg. Civil and Mining Engineer.

1904 Hanau, Carl. Coronation Syndicate, Ltd., P. O. Box 4602, Johannesburg. Director of Companies.

1903 Hanlon, Claude, A.I.M.M. P. O. Box 113, Klerksdorp. Cyanide Manager.

1903 Hardach, Edward Emery, B.S. Knight Central, Ltd., P. O. Box 91, Germiston. Mine Manager.

1898 Hardwick, William. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mill Manager.

1903 Harland, Henry Lumley. Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Mechanical Engineer and Battery Manager.

1905 Harris, James Vaughan. Blanket Mine, *via* Gwanda, Rhodesia. Cyanide Manager.

1908 Harris, Julian Herbert. Jumpers Deep, Ltd., P. O. Box 30, Cleveland. Amalgamator.
(Assoc. 1906)

1896 Hartley, Alexander Holmes. New Goch G. M., Ltd., P. O. Box 1096, Johannesburg. Assayer.

1903 Hastings, Bernard John. P. O. Box 5337, Johannesburg. Assayer.

1904 Haugh, Henry. Pipeclay Creek, *via* Pambula, New South Wales. Metallurgist.

1903 Hay, Errol. Johannesburg Consolidated Investment Co., Ltd., P. O. Box 231, Johannesburg. Assistant Consulting Engineer.

1909 Healey, John Edward. Vogelstruis Consol Deep, P. O. Box 3, Florida. Mine Manager.

1903 Hebbard, Austin. Block B Langlaagte Estate and G. M. Co., Ltd., P. O. Box 58, Langlaagte. Assayer.

1903 Heim, Felix. Messrs. Wernher, Beit & Co., 1, London Wall Buildings, London, E.C.

1898 Hellmann, Frederick, M.I.M.M. 530 Mills Buildings, 15 Broad Street, New York, U.S.A. Consulting Mining Engineer.

1903 Henderson, Henry. Geldenhuis Estate and G. M. Co., Ltd., P. O. Box 5, Cleveland. Mine Secretary.

1908 Henderson, John McClelland, PhD., F.G.S., M.I.M.M. P. O. Box 1146, Johannesburg. Consulting Mining Engineer and Geologist.

1909 Herald, Andrew James. Knights Deep, Ltd., P. O. Box 143, Germiston. Mill and Tube Foreman.
(Assoc. 1908)

Year of Election.

1903 Herbert, Sydney A. Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanide Foreman.

1909 Herring, Frank. Crown Mines, Ltd., P. O. Box 5, Fordsburg. Cyanide Foreman.

1909 Hesom, William Wade. Sheba G. M. Co., Ltd., Eureka. Amalgamator.

1903 Heymann, Alexander, M.Ch., M.Ph., M.A. The Laboratories, Fox Street, P. O. Box 3427, Johannesburg. Consulting Analytical Chemist.

1898 Higham, James. City & Suburban G. M. & E. Co., Ltd., P. O. Box 1026, Johannesburg. Cyanide Manager.

1905 Hilditch, Charles Henry. 82 Fourth Street, Boksburg North. Cyanider.

1897 Hill, Howard. Sheba G. M. Co., Ltd., Eureka. General Manager.

1908 (Assoc. 1898) Hilliard, Cecil Brewitt. Crown Mines, Ltd. (C. Section) Reduction Works, Fordsburg. Cyanide Manager.

1904 Hilner, Ernest George. P. O. Box 1145, Johannesburg. Mine Surveyor.

1898 Holford, William George, A.M.I.C.E. Apex Mines, Ltd. (Gold Section), Benoni. Manager.

1902 Hollis, Ernest. Bank of Africa, Ltd., P. O. Box 1123, Johannesburg. Assayer.

1907 HOLLOWAY, George Thomas, M.I.M.M. (*Corresponding Member of Council*). 57-58 Chancery Lane, London, W. C.

1903 Honnold, William Lincoln, M.I.M.M. Consolidated Mines Selection, Ltd., P. O. Box 2269, Johannesburg. Mining Engineer.

1905 Houghton, John Henry. P. O. Box 5237, Johannesburg. Manufacturer's Agent.

1909 Hövig, Peter. Benkoelen, Sumatra, Dutch East Indies. Government Mining Engineer.

1909 Howard, Harold Arthur Liddon. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1906 Hoyer, Anton Godfrey. Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Surveyor.

1902 Hughes, Hugh, M.Am.I.M.E. Ventanas Mining and Exploration Co., Ltd., Villa Corona, via Chavarria, Durango, Mexico. Mining Engineer.

1898 Hulsteyn, Sir Willem van, Kt. P. O. Box 46, Johannesburg. Solicitor.

1904 Hunter, Archibald Moses. Langlaagte Estate & G. M. Co., Ltd., P. O. Box 98, Langlaagte. Storekeeper.

1899 Hunter, Charles, A.I.M.M. New Lisbon Berlyn, Ltd., Pilgrims Rest. Metallurgical Chemist.

1904 Hunter, George James. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Assayer.

1909 Hurlbatt, Charles. Eldorado Mine, Lomagundi, S. Rhodesia. Mill Manager.

1898 Hutton, Charles Edward, M.I.M.M. Van Ryn Gold Mines, Ltd., P. O. Box 22, Benoni. Mine Manager.

1906 Hyland, Lawrence. Gatooma Station, Rhodesia. Assayer and Cyanide Chemist.

1904 Hynd, John. Consolidated Gold Fields of South Africa, Ltd., P. O. Box 67, Bulawayo, Rhodesia. Mechanical Engineer.

1909 (Assoc. 1897) Inglis, Alexander Brownlee. Thornton Little G. M. Co., P. O. Box 77, Barberton Mill Manager.

1908 Inglis, Malcolm Canmore. New Club, P. O. Box 1078, Johannesburg. Mine Manager.

1903 I'Ons, John. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Battery Manager.

1905 Irvine, Louis G., M.A., M.D. P. O. Box 320, Johannesburg. Medical Practitioner.

1906 Isaacs, Robert McIntosh, B.E. (Sydney). 63 Charlton Road, Blackheath, London, S.E. Mining Engineer.

1904 Jarman, Edwin Arthur. Crown Mines, Ltd., P. O. Box 102, Fordsburg. Mill Foreman.

1906 James, Christopher Gordon, A.I.M.M. Metallurgist. *(Address wanted.)*

Year of Election.

1905 James, William Henry Trewartha, M.I.M.M., M.I.M.E. Finsbury House, Blomfield Street, London, E.C. Consulting Engineer.

1907 Jansen, Petrus Johannes. Moeara Aman, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1897 Jennings, Sidney Johnston, M.Am.I.M.E., M.I.M.M. c/o Messrs. Smith & Perkins, Mills Building, Broad Street, New York, U.S.A. Mining Engineer.

1905 Johns, James. Ginsberg G. M. Co., P. O. Box 16, Knights. Mine Foreman.

1897 Johns, J. Harry, M.I.M.M., A.M.I.C.E. Consolidated Investment Co., Ltd., P. O. Box 231, Johannesburg. Consulting Engineer.

1904 Johnson, Alex. Downing. Jessie Mine, per private bag, Bulawayo, Rhodesia. Sampler.

1908 Johnson, Arthur John. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Foreman Amalgamator.

1894 JOHNSON, Edward H. (*Past President*). East Rand Proprietary Mines, Ltd., P. O. Box 134, East Rand. Consulting Metallurgist.

1904 Johnson, Henry Howard, A.M.I.Mech.E. c/o R. R. Hollins, Standard Bank Buildings, Johannesburg. Mining Engineer.

1908 Johnson, Hugh Michell Luttman. Rand Club, P. O. Box 1032, Johannesburg. Mining Engineer.

1897 Johnson, J. H. Princess Estate & G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanide Manager.

1899 Johnson, Thomas, P. O. Box 370, Johannesburg. Mine Manager.

1898 JOHNSTON, A. McArthur M.A., M.I.M.M., F.C.S. (*President*). P. O. Box 108, Germiston. Reduction Works Manager. (*Temporary address to January, 1910*: Woodside, Johnston, Scotland.)

1908 Johnston, Herbert. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Shift Boss.

1907 Jolly, Harold R. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Chief Assayer.

1898 Jolly, Thomas S. Jubilee Gold Mining Co., Ltd., P. O. Box 1122, Johannesburg. Analytical Chemist and Assayer.

1905 Jones, Charles Kenelm Digby, F.G.S., A.I.M.M. Jumbo G. M. Co., Ltd., Salisbury, Rhodesia. Mining Engineer.

1903 Jones, Henry Ewer, A.R.S.M., M.I.M.M. The Lodge, Culmington Road, Ealing, London, W. Mining Engineer.

1904 (Assoc. 1902) Jones, John Acheson. Cinderella Deep G. M. Co., Ltd., P. O. Box 75, Boksburg. Cyanide Manager.

1908 Jones, William Harry Richardson. Public Works Department, Standerton. Civil Engineer.

1906 Jongh, J. N. de. P. O. Box 1030, Johannesburg. Director of Companies.

1903 Jordan, Trevor William. 7, Hillier Street, Troyeville, Johannesburg. Mining Engineer.

1897 Jory, James Vine. East Rand Proprietary Mines, Ltd., P. O. Box 109, East Rand. Cyanide Manager.

1906 Kennedy, J. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube Mill Foreman.

1904 Kern, Percival Ernest. Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Assayer.

1899 Kerr, James. New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Analytical Chemist and Cyanide Manager.

1908 Kerr, Thomas. Benoni Consolidated G. M., Ltd., Benoni. Mine Manager.

1908 Kewley, Robert Rigby. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Sampler.

Year of Election.

1905 King, Andrew Clark. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Battery Manager.

1906 Kingston, Charles Burrard, B.A., B.A.Sc., M.I.M.M. Messrs. Lewis & Marks, P. O. Box 1030, Johannesburg. Consulting Engineer.

1903 Kisch, Charles Herbert Moses, F.M.Chart.Inst.P.A., (London). Messrs. D. M. Kisch & Co., P. O. Box 668, Johannesburg. Patent Agent.

1908 Klein, Max Marcus, M.D., Ph.D. P.O. Box 16, Brakpan. Medical Practitioner.

1902 Kotzé, Robert Nelson, B.A., M.E. Mines Department, Winchester House, P. O. Box 2807, Johannesburg. Government Mining Engineer.

1905 Kriekhaus, Karl. Ketahoen Mining Co., Ltd., Post Ketaun, West Coast Sumatra, Dutch East Indies. Mine Manager.

1903 Krige, William Adolph, B.A., B.Sc., M.E. Van Ryn Deep, Ltd., P. O. Box 225, Benoni, Johannesburg. Mine Manager.

1903 Lace, John Dale. P. O. Box 855, Johannesburg. Director of Mines.

1903 Lacy, Edward D. de. P. O. Box 206, Roodepoort. Mine Manager.
(Assoc. 1897)

1904 Lamont, Cyril Howard. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Amalgamator.

1904 Langermann, Hon. Max., M.L.C. P. O. Box 1365 Johannesburg.

1904 LASCHINGER, Edgar Jacob, B.A.Sc., M.E., M.I.M.M. *(Member of Council).* Messrs. H. Eckstein
(Assoc. 1904) & Co., P. O. Box 149, Johannesburg. Mechanical Engineer.

1906 Laurie, Everatt *(Life Member).* Siamese Tin Syndicate, c/o Messrs. Adamson, Gilfillan & Co., Penang, Straits Settlements. Mining Engineer.

1906 Lawson, George Alexander. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand.
(Assoc. 1898) Cyanide Foreman.

1903 Lea, Henry. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 2036, Johannesburg. Cyanider.

1903 Lea, James. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Manager.
(Assoc. 1898)

1902 Lee, Clifford Walker. Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Assayer.

1902 Lee, Howard John. North Randfontein G. M. Co., Randfontein. Battery Manager.

1898 Leitch, Quinton James. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Cyanide Manager.

1905 Lenz, Otto. Messrs. Reunert & Lenz, P. O. Box 92, Johannesburg. Engineer.

1903 Leslie, Charles Duff. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Mining Engineer.

1909 LESLIE, Hugh Mackenzie, A.I.M.M., M.Am.I.M.E. (*Corresponding Member of Council*), Kolar
(Assoc. 1898) Gold Field Central Metallurgical Establishment, Marikuppam, Mysore State, S. India. Consulting Metallurgical Engineer.

1906 Lewis, Henry Haselden. Namaqua Copper Co., Ltd., Concordia, Little Namaqualand, Cape Colony. Metallurgist.

1903 Lewis, Francis Blount. New Unified (Main Reef) G. M. Co., Ltd., P. O. Box 5, Maraisburg. Mining Engineer.

1897 Leyson, William, A.R.S.M., M.R.A.C., M.R.A.S.E. P. O. Box 3, Florida. Mine Manager.

1903 Lichtenstein, Alfred, M.E. (Freiberg). Van Ryn G. M. E., Ltd., P. O. Box 22, Benoni. Mining Engineer.

1908 Lightbody, Thomas. Luipaardsvlei Estate & G. M. Co., Ltd., P. O. Box 268, Krugersdorp. Tube-miller.

1907 Lindsay, Robert. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.
(Assoc. 1904)

Year of Election.

1897 Little, James Berkeley. The Wanderer G. M. Co., Selukwe, Rhodesia. Mine Manager.

1894 Littlejohn, James (*Hon. Treasurer*). Chemical and Metallurgical Laboratories, Simmonds Street, P. O. Box 849, Johannesburg. Chemist and Assayer.

1905 Lloyd, William D'Arcy. East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Cyanider.

1909 Locker, James Thomas. Warrington, England. Screening Manufacturer.

1902 Lockhart, Thomas Lamb, F.C.S. Glencairn Main Reef G. M. Co., Ltd., P. O. Box 191, Germiston. Cyanide Manager.

1894 Loevy, J., Ph.D., M.A. (*Past President*). 21 Luisenstrasse, Berlin, N.W. 6, Germany. Analytical and Metallurgical Chemist.

1909 Lomas, Albert Arthur. Luipaardsvlei Estate & G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Cyanide Manager.

1909 Lowdon, George Webster. Moodie's G. M. & Exploration Co., Ltd., P. O. Box 117, Barberton. Joint Manager.

1908 Lothrop, Percy Dunbar. Crown Mines, Ltd., P. O. Box 5, Fordsburg. Surveyor.

1895 Ludlow, L. Standard Bank of S. A., Ltd., Bulawayo, Rhodesia. Assayer.

1903 Macaulay, Donald, M.A., M.B., C.M. (Edin.), M.L.A. Cleveland. Physician.
(Assoc. 1898)

1898 MacCarthy, George Desmond Richard. Rand Klipfontein Co , P. O. Box 88, Benoni. Electrical Engineer.

1908 MacDonald, Bernard. Apartado 33, Guanajuato, Mexico. Mining Engineer.

1896 Macdonald, F. G., F.C.S. Durban Roodepoort G. M. Co., Ltd., P. O. Box 111, Roodepoort. Analytical Chemist, Assayer and Cyanide Manager.

1908 Macdonald, Peter Archibald. Etna Development Co.. Ltd., Gatooma, Rhodesia. Mill and Cyanide Manager.

1897 Macfarlane, Thomas John M. P. O. Box 1032, Johannesburg. Mining Engineer.

1909 MacGregor, Robert Duncan. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Electrical Engineer.

1905 Mack, John. Rouge Mine, Gatooma, Rhodesia. Mine Manager.

1906 Mackintosh, Harry George. Dunraven Hotel, Selukwe, Rhodesia. Cyanide Manager.

1909 Maclatchy, Allan Gilmour. South Rand Exploration Co., Ltd., Balfour, Transvaal. Mine Manager.

1903 Macleod, Donald McKay. New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Battery Manager.

1906 Macqueen, William Pirrie Otto, A.R.S.M., S.I.M.M. c/o Gillanders, Arbuthnot & Co., P. O. Box 174, Calcutta, India.

1905 Mahaut, Henry. Le Champ d'Or (French) G. M. Co., Ltd., P. O. Box 2, Luipaardsvlei. Mining Engineer and General Manager.

1905 Main, Alfred F., M.I.M.M. El Oro Mining and Railway Co., Ltd., El Oro, Estado de Mexico, Mexico. Assistant General Manager.

1907 Mance, John Courtenay, A.I.M.M. c/o Messrs. Guthrie & Co., Ltd., Singapore. Mining Engineer.

1906 Manners, Charles Edward. West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp. Battery Manager.

1898 Marks, Samuel. P. O. Box 379, Pretoria. Capitalist.

1902 Marquard, Johann David. Glen Deep, Ltd., P. O. Box 184, Germiston. Mine Captain.

1898 Marriott, Hugh Frederick, A.R.S.M., A.R.C.S., F.G.S., M.I.M.M. c/o Messrs. Wernher, Beit & Co., London Wall Buildings, London, E.C. Consulting Mining Engineer.

Year of Election.

1903 Martin, Ernest Holroyd. Langlaagte Estate and G. M. Co., Ltd., P. O. Box 98, Langlaagte. Battery Manager.

1908 Martin, Henry Stuart, M.I.M.M. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Consulting Mining Engineer.

1908 Mather, Thomas W. Crown Mines Ltd., P. O. Box 102, Fordsburg. Mining Engineer.

1897 Matthews, Ernest Lindley. Nourse Mines, Ltd., P O. Box 32, Denver. Assayer, Analytical Chemist and Cyanide Manager.

1899 McCracken, John. Sandycroft Foundry Co., Corner House, P. O. Box 1976, Johannesburg. Merchant.

1906 McCrae, John, Ph.D. Government Laboratories, P. O. Box 1080, Johannesburg. Government Analyst.

1906 McEwen, Moses Thomas. Bonsor Mine, Selukwe, Rhodesia. Mine Manager.

1909 McGuire, James Edward, B.Sc. Messrs. S. Neumann & Co., P. O. Box 485, Johannesburg. Mining Engineer.

1907 McKellow, William Henry. Robinson Deep G M. Co., Ltd., P. O. Box 5168, Johannesburg. Cyanider.

1907 McKerrell, David. Galston Road, Hurlford, Ayrshire, Scotland. Amalgamator.

1902 McLennan, John. Roodepoort United Main Reef G. M. Co., Ltd., P. O. Box 115, Roodepoort. Assayer.

1903 McMillan, Quinton Clark. P. O. Box 3663, Johannesburg. Battery Manager.

1894 McNaughtan, A. Luipaardsvlei. Cyanide Manager.

1909 Mehliss, Max, M.D. Rietfontein Lazaretto, P. O. Box 1076, Johannesburg. Medical Superintendent.

1895 Melvill, George. French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei. Cyanide Manager.

1907 Metcalf, Joseph Ernest. Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Analytical Chemist.

1904 Meyer, Charles Edward. c/o L. C. Meyer, P. O. Box 199, East Rand. Analytical Chemist and Assayer.

1899 Meyer, Hans, A.R.S.M. Rand Mines, Ltd., P. O. Box 1056, Johannesburg. Mining Engineer.

1907 Michaux, Dighton. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Shift Boss.

1904 Michelmore, P. J. Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg.

1897 Milligan, John Thomas, M.I.M.M. New West Bonanza G. M. Co., Ltd., P. O. Box 137, Klerksdorp. Consulting Engineer.

1905 Mills, Arthur L. University of Melbourne, Carlton, Victoria, Australia. Metallurgist.

1906 Mills, Frank. North Randfontein G. M. Co., Randfontein. Cyanide Manager.
(Assoc. 1898)

1908 Minards, William Ivey Kennedy. Bennallack, Grampound Road, Cornwall. Cyanider.

1908 Mitchell, Andrew. Dunraven Mine, Selukwe, Rhodesia. Mine Owner.

1903 Mitchell, John Thomas. East Rand Proprietary Mines, Ltd., East Rand. Engineer.

1898 Mitchell, William Edward Clifton, D.S.O., M.I.M.M., M.Am.I.M.E., Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Mine Manager.

1902 MOIR, James, M.A, D.Sc.(Aber.), F.C.S., F.R.S.S.Af. *(Vice-President).* Mines Department, Johannesburg. Chemist to Department.

1899 Morison, Stephen. Crown Mines, Ltd., P. O. Box 1145, Johannesburg. Smelter.

1908 Morkel, Arthur Loreht Rubidge. P. O. Joker Mine, Salisbury, Rhodesia. Mine Owner and Tributor.

Year of Election.

1897 Morrell, Henry H. Rose Deep, Ltd., P. O. Box 6, Germiston. Cyanide Manager.

1903 Morris, James, A.R.S.M., A.I.M.M. East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Joint General Manager.

1909 Morris, James William Henderson. East Rand Proprietary Mines, P. O. Box 37, East Rand. Assayer.

1906 Morris, Thomas Walters. Nigel Deep, Ltd., P. O. Box 50, Nigel. Assayer.
(Assoc. 1904)

1898 Morrisby, Arthur Clayton. Hamburg Mine, Golden Valley, Rhodesia. Mine Owner.

1896 Morrisby, Percy Tasman B. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanide Manager.

1904 Morse, Willard S., M.I.M.M., M.Am.I.M.E., F.S.A., M.I.M.E. Seaford, Delaware, U.S.A. Mining Engineer.

1906 Morton, W. D. P. O. Box 1331, Johannesburg. Accountant and Auditor.

1908 Mosenthal, Berthold Philip. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assayer.

1908 Mossop, William Clement. Treasury Gold Mines, Ltd., Cleveland. Mining Engineer.

1905 Mumford, Frank Townsend, A.I.M.M. Athenæum Club, Johannesburg. Metallurgist.

1909 Murdoch, David. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1909 Murning, John. Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Tubemiller.

1909 Murray, Myles Thornton, M.Sc., Transvaal University College, P. O. Box 1176, Johannesburg. Lecturer and Demonstrator in Metallurgy and Assaying.

1905 Nahnsen, Dr. George, 7 Heinrich Hertzstr, Hamburg, Germany.

1909 Napier, William Floranee. Simmer Deep, Ltd., P. O. Box 4397, Johannesburg. Assistant Smelter.

1909 Neal, Walter (*Corresponding Member of Council*). Las Dos Estrellas, Apartado 34, El Oro, Mexico. Superintendent, Cyanide Department.
(Assoc. 1909)

1908 Neale, Arthur Leggett. Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. General Manager.

1905 Nel, Pieter Otto, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Mine Captain.

1903 Newhall, Percy Melrose, B.Sc. Messrs. S. Neumann & Co., P. O. Box 485, Johannesburg. Mining Engineer.

1905 Newman, Arthur Dudley. Messrs. Fraser & Chalmers, P. O. Box 619, Johannesburg. Engineer.

1907 Newson, James Aldrovandi. P. O. Box 250, Potchefstroom. Assayer.

1902 Newton, Samuel. Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Cyanide Manager.

1907 Nichol, Thomas T. Springs Mines, Ltd., P. O. Box 54, Springs. Sampler.

1904 Nicholls, Charles Beresford. Le Champ d'Or G. M. Co., Ltd., P. O. Box 2, Luipaardsvlei. Mining Engineer.

1903 Nicholls, Reginald. Queen's Mine, P. O. Box 220, Bulawayo, Rhodesia. Metallurgist.

1909 Nicholson, Arthur Steele. Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Foreman.
(Assoc. 1907)

1903 Nicklin, William. P. O. Box 1908, Johannesburg. Assayer.

1909 Nicol, Gilbert. Simmer Deep, Ltd., P. O. Box 11, Germiston. Cyanider.

1905 Nicolaus, Richard Clement, M.I.M.M. P. O. Box 3143, Johannesburg. Consulting Mining Engineer.

1902 Niven, Robert, P. O. Box 4022, Johannesburg. Merchant.

Year of Election.

1901 Normand, James Normand. Surprise Mine, Surprise Siding, Gwelo, Rhodesia. Mine Manager.

1899 North, Walter Joseph Richard. Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort. Mill Manager.

1907 (Assoc. 1898) Ogle, Frank Bray. Simmer Deep, Ltd., P. O. Box 178, Germiston. Smelter.

1897 O'Hara, John Dennis. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Cyanide Manager.

1903 Olds, Henry Francis *(Life Member)*, M.I.M.M., Chapel Road, St. Just, R.S.O., Cornwall, England. Mining Engineer.

1903 Oliver, William Henry, A.I.M.M. 10, Avondale Road, Truro, Cornwall, England. Metallurgist.

1908 Orr, John, B.Sc., M.I.Mech.E., A.M.I.C.E. Transvaal University College, P. O. Box 1176, Johannesburg. Professor of Mechanical Engineering.

1898 Osborn, Sidney S. Glen Deep, Ltd., P. O. Box 184, Germiston. Cyanide Manager.

1909 (Assoc. 1903) Osterloh, Ernest Albert. Simmer Deep, Ltd., P. O. Box 178, Germiston. Slimes Foreman.

1906 Packard, Harold Turner. Nourse Mines, Ltd., P. O. Box 32, Denver. Shift Boss.

1905 Pam, Edgar. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mining Engineer.

1905 Parker, William Frank. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Surveyor.

1906 Parry, Charles Frank. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mine Captain.

1906 Parsons, Cyril Edward, M.I.M.M., M.Am.I.M.E. Chevender, Chislehurst, Kent, England. Mining Engineer.

1907 Patterson, Philip Francis, A.R.S.M., A.I.M.M. c/o Messrs. Turrell & Patterson, Mining Engineers, Penhalonga, Rhodesia. Mining Engineer.

1903 Payne, Albert Edward, A.R.S.M., M.I.M.M. New Primrose G. M. Co., P. O. Box 193, Germiston. Mine Manager.

1894 Pead, Clement H. P. O. Box 252, Bulawayo. Metallurgist and Cyanide Manager.

1895 Pearce, John, F.R.C.I. Higher Knowle, Kingsbridge, Devon, England. Gentleman.

1894 PEARCE, Sidney Herbert, M.I.M.M. *(Past President; Life Member)*. Messrs. Eckstein & Co., P. O. Box 149, Johannesburg. Consulting Chemist and Metallurgist.

1908 Pearson, Thomas. Cloverfield Mines, Ltd., P. O. Geduld. Mine Manager.

1904 Pedrini, Joseph. Luipaardsvlei Estate & G. M. Co., Ltd.. P. O. Box 53, Krugersdorp. Mine Captain.

1903 Pencier, Henry Percy de, M.Sc. 145 Bayle Street, Montreal, Canada. Mining Engineer.

1898 Penman, William. Luipaardsvlei Estate & G. M. Co., Ltd, P. O. Box 53, Krugersdorp. Assayer.

1899 Pepler, Daniel Joseph. Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Mill Manager.

1908 Perkin, Dan Ward. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Assistant Surveyor.

1897 Petersen, Hans Theodor. Geldenhuis Estate & G. M. Co., Ltd., P. O. Box 5, Cleveland. General Manager.

1909 Phillips, Frederick David. Knights Deep, Ltd, P. O. Box 143, Germiston. Cyanider.

1907 Phillips, John Moyle. Transvaal G. M. Estates, Ltd., P. O. Box 10, Pilgrim's Rest. Mine Captain.

1908 Phillips, Josiah Charles. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanider.

1909 Philp, Henry. Lebong Donok, Benkoelen, Sumatra, Dutch East Indies. Underground Manager.

1905 Philpott, John William. Glen Deep, Ltd., P. O. Box 184, Germiston. Cyanide Foreman.

1904 Pidsley, Harry William. Geldenhuis Estate & G. M. Co., Ltd., P. O. Box 5, Cleveland. Battery Manager.

Year of Election.

1909 Piele, William. East Rand Proprietary Mines, Ltd., P. O. Box 138, East Rand. Mining Engineer.

1898 Pill, Robert. West Rand Consolidated Mines, Ltd. P. O. Box 38, Krugersdorp. Mining Engineer.

1904 Pinkvos, Ernst A. Th. K., M.E. Ketahoen G. M. & Exploration Co, Ltd, Lebong Siman, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1897 Pope, William Toll, M.I.M.M. P. O. Box 2448, Johannesburg. Mining Engineer.

1909 (Assoc. 1908) Price, Walter Samuel Vincent. Cinderella Deep, Ltd., P. O. Box 75, Boksburg Miner.

1904 **Pollitt**, Robert Barnabas, A.M.I.C.E., F.I.C. (*Corresponding Member of Council*). Compania Nacional Mexicana de Dinamita y Explosivos (S.A.), Dinamita (*via* Noe), Durango, Mexico. General Manager.

1908 Popham, John Leyborne, A.I.M.M. P. O. Box 669, Bulawayo, Rhodesia. Mining Engineer.

1909 Potter, Henry Samuel. 32 Plein Street, Johannesburg. Mechanical Engineer and Works Manager.

1902 Powell, Owen Price. Knights Deep, Ltd., P. O. Box 143, Germiston. Mining Engineer.

1897 Price, Henry B. Crown Mines, Ltd., P. O. Box 901, Johannesburg. Mining Engineer.

1895 Prister, Professor August. 13 Via Poste Nuove, Trieste, Austria. Chemist.

1897 Prout, William Morris, A.M.I C E., M.Am.I.M E. Witwatersrand G. M. Co., Ltd, P. O. Box 1, Knights. Mining Engineer and General Manager.

1909 Pryce, Leslie. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Resident Engineer.

1902 Pyles, Jay F. Village Deep, Ltd, P. O. Box 1145, Johannesburg. Cyanide Manager.

1906 Quin, Stanley Cleveland Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg. Cyanider.

1905 Quinn, Jno. W., J.P., M.L.A. P. O. Box 1454, Johannesburg. Merchant.

1898 Raine, Robert. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Mine Manager.

1894 Rand, Ernest T. P. O. Box 9, Boksburg North. Cyanide Manager and Engineer.

1909 Rascher, Louis Hugo, M.E Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Assistant Mine Surveyor.

1903 Reading, Richard Washington. c/o R. S. Reading, Redding, Shasta County, California, U.S.A. Assayer and Mining Engineer.

1903 Reyersbach, Louis J. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Director of Companies.

1909 Reynolds, James. Consolidated Langlaagte Mines, Ltd, P. O. Box 13, Langlaagte. Mill Foreman.

1907 Reynolds, James Percival. Milky Way Gold Mine, Golden Valley, Gatooma, Rhodesia. Assayer.

1907 Richards, Fred., M.I.M E. "Blaenau," 43 Compton Road, Canonbury, London, N. (*temporary address*, P. O. Box 67, Middelburg, Transvaal). Director of Mining Companies.

1902 **Richardson**, Alexander, M.I.M.M. (*Member of Council*). Treasury G. M. Ltd., Cleveland. Mining Engineer and Surveyor.

1898 (Assoc. 1897) Roberts, Frederick Charles, M.I.M.M. 2317 Piedmont Avenue, Berkeley, California, U.S.A. Mining Engineer.

1904 Roberts, Stephen, M.I.M.E. Midas Deep, Ltd, P. O. Box 8, Randfontein. Surveyor.

1909 Robertson, David. Treasury Gold Mines, Ltd., Cleveland. Mill Manager.

1902 Robeson, Antony Maurice, M.I.M.M. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Consulting Mechanical Engineer.

Year of Election.

1904 Robinson, Leonard Joseph. Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston.
(Assoc. 1903) Assayer.

1904 Robinson, Reginald William, A.R.S.M., A.I.M.M. Penhalonga, Umtali, Rhodesia. Mining Engineer.

1907 Robinson, Rupert Frere. New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Mining Engineer.

1909 Robson, John Newton. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Mill Foreman.

1903 Roche, H. Fraser. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Surveyor.

1903 Roe, Wilfred Henry. Jumpers G. M. Co., Cleveland. Cyanide Manager.

1904 Rogers, Arthur. Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights. Assayer.

1908 Rogers, Nelson. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Surveyor.

1904 Rose, Arthur Francis. Fanti Consolidated Mines, Tarkwa, West Africa. Mining Engineer.

1906 Rosendorff, Hans. General Mining & Finance Corporation, Ltd., P. O. Box 1118, Johannesburg. Buyer.

1909 Ross, David George Fraser. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1906 Ross, Oswin Douglas. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Smelter.

1907 Rouillard, R. A. Klipval Mine, P. O. Louwsburg, *via* Vryheid, Natal. Mining Engineer.

1903 Royse, George Herbert, Assoc.M.&M.O.S.M.(N.Z.) Mine Surveyor. *(Address wanted.)*

1904 Rusden, Charles Edgar. *(Address Wanted.)* Assayer.

1903 Russell, Samuel. New Kleinfontein Co., Ltd., P. O. Box 2, Benoni. Battery Manager.

1904 Sale, George Geoffrey. P. O. Box 3308, Johannesburg. Engineer.

1898 Salkinson, A. Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Chemist, Metallurgist and Cyanide Manager.

1902 SANER, Charles B., M.I.M.M., A.I.M.E., F.R.C.I, *(Vice-President)*. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mining Engineer.

1906 Sartorius, Kurt K. Hohnbaum, M.E. Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs. Mining Engineer.

1907 Scarf, Henry A. Robinson Deep G. M. Co., Ltd, P. O. Box 1488, Johannesburg. Cyanide Foreman.

1894 Schlesinger, Jacques, Ph.D. New Transvaal Chemical Co., Ltd., P. O. Box 1455, Johannesburg. Industrial Chemist and Managing Director.

1906 Schmitt, Charles Oscar. Consolidated Gold Fields of S A., Ltd., P. O. Box 6404, Johannesburg. Mechanical Engineer.

1899 Schumacher, Raymond W. Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Director
(Assoc. 1897) of Companies.

1907 Schwabe, Septimus Burdett. West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp. Tube Mill Amalgamator.

1909 Scotland, Robert Duncan. East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Assayer.

1909 Scott, Lawrence E. East Rand Proprietary Mines. Ltd., P. O. Box 56, East Rand. Mechanical Engineer.

1905 Sellers, John Howard. Western Rand Estates, Ltd., P. O. Box 32, Randfontein. General Manager.

1908 Semple, William Boyd. Luipaardsvlei Estate & G. M. Co., Ltd., P. O. Box 53, Krugersdorp.
Assoc. 1905) Chemist and Assayer.

Year of Election.

1895 Shallcross, Vincent Fairfield, M.I.M.M. (*Life Member*). Stock Exchange Buildings, 113 Pitt Street, Sydney, New South Wales, Australia. Metallurgist.

1904 Sharp, Edward Miles. Warren Hill Syndicate, P. O. Ariston, Klerksdorp. Mine Foreman.

1906 Sharp, William Park Fulton. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Foreman.

1904 Sharwood, Walter Sydney, A.I.M.M., M.Am.I.M.E. Macdonal Street, Jeppes Extension. Mining Engineer.

1904 Shlom, Solomon. P. O. Box 1989, Johannesburg. Analytical and Manufacturing Chemist.

1905 Sim, Anthony C. B. Malcolm. City & Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Assistant Assayer.

1904 Simmons, Frederick Lance. Geldenhuis Deep, Ltd., P. O. Box 17, Geldenhuis. Amalgamator.

1903 Simmons, Thomas William. Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Battery Manager.

1909 Simpson, Charles Butement. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Foreman.
(Assoc. 1904)

1908 Simpson, Ffreebairn Liddon. Knight Central G. M. Co., Ltd., P. O. Box 91, Germiston. Amalgamator.

1903 Simson, Leslie, M.I.M.M. Consolidated Gold Fields of S. A., Ltd., P. O. Box 67, Johannesburg. Mining Engineer.

1909 Sly, Alfred William. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Surveyor.

1903 SMART, George Osler, A.I.M.M., M.Am.I.M.E. (*Member of Council*). Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Reduction Works Manager.

1908 Smith, Edward. French Rand G. M. Co., Ltd., P. O. Box 125, Luipaardsvlei. Amalgamator.

1909 Smith, John Arthur. Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

1904 Smith, Mather. Worcester Exploration & G. M. Co., Ltd., P. O. Box 85, Barberton. Cyanide Manager.

1905 Smith, William Henry. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mine Foreman.

1905 Smythe, John James Robert. Warren Hill Syndicate, P. O. Ariston, *via* Klerksdorp. Mechanical Engineer.

1903 Solly, Bertram Campbell Travers. May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston. Assistant Surveyor.

1909 Solomon, Lionel. P. O Box 503, Johannesburg. Surveyor.

1898 Sörensen, Stanley Severin, M.I.E.E., M.C.M.I. Steptoe Valley Smelting & Mining Co., McGill, White Pine County, Nevada, U.S.A. General Manager and Consulting Engineer.

1896 Spengel, Hermann Georg, M.I.E E. P. O. Box 3498, Johannesburg. Consulting Electrical Engineer.

1903 Spoor, Alfred Leonard. Messrs. D. M. Kisch & Co., P. O. Box 668, Johannesburg. Patent Agent.

1897 Stacpoole, Adam Richard W., M.Am.I.M.E. Metallurgist and Cyanide Manager. *(Address wanted.)*

1905 STANLEY, George Hardy, A.R.S.M., M.I.M.M., M.I.M.E. (*Member of Council*). Transvaal University College, P. O. Box 1176, Johannesburg. Professor of Metallurgy.

1909 Steel, Walter Scott. Vogelstruis Estate & G. M. Co., Ltd., P. O. Box 116, Roodepoort. Cyanide Manager.

1903 Steels, Sidney Harold. Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Cyanide Manager.

1898 Stephan, Henri. Eldorado Banket G. M. Co , Ltd., Eldorado, Rhodesia. Mining Engineer.

Year of Election.

1904 Stevens, James Howard. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Storekeeper.

1894 Stewart, Alexander Forson. East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Chemist and Assayer.

1908 Stiell, William, Jumbo Mine, Mazoe, Rhodesia. Mill Manager.

1904 Stokes, Ralph Shelton Griffin. Messrs. H. Eckstein & Co.'s Central Administration, P. O. Box 1145, Johannesburg. Mining Engineer.

1908 Stokoe, John Calvert. Pigg's Peak Development Co., Pigg's Peak, Swaziland. Metallurgist.

1906 Stout, William Henley. Robinson Deep G. M. Co., Ltd., P. O. Box 1188, Johannesburg. Mill Foreman.

1906 Strout, Ernest Allen. Guanajuato Development Co, Apartado 25, Guanajuato, Mexico. Mine Foreman.

1906 St. Quintin, Erroll Tracey. Cyanider. *(Address wanted.)*

1908 Strang, John. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

1898 Strange, Harold F. Johannesburg Consolidated Investment Co., Ltd., P. O. Box 2527, Johannesburg. Joint General Manager.

1909 Streeter, John Radford. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Mine Captain.

1908 Suart, Arthur B., M.I.M.M. Messrs. Johnson & Sons Smelting Works, Ltd., 23 Cross Street, Finsbury, London, E.C. Bullion Refiner.

1906 Swinney, Leslie Alfred, A.R S.M., A.I.M.M. Lorne Villa, Sidcup, Kent. Metallurgical Chemist.

1907 Sydenham, Charles Theodore. Knights Deep, Ltd., P. O. Box 143, Germiston. Assayer.

1904 Taggart, John. Crown Mines, Ltd., P. O. Box 102, Fordsburg. Amalgamator.

1909 Taggart, Thomas. Porges Randfontein G. M. Co., Ltd., Randfontein. Mill Manager.

1909 Tayler, Walker Alfred Charles. Simmer & Jack Proprietary Mines, Ltd, P. O. Box 192,
(Assoc. 1908) Germiston. Cyanide Foreman.

1908 TAYLOR, George M. *(Corresponding Member of Council).* Portland Mill, Colorado Springs, Colorado, U.S.A. Mill Superintendent.

1905 Taylor, Henry. Raleigh & Rockey Agency, Ltd., P. O. Box 790, Johannesburg. Accountant.

1902 Taylor, John Augustine. Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanide Manager.

1909 Taylor, Robert McGeown. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Mill Manager.

1902 Taylor, Wilfrid. Three Sisters Mine, *via* Barberton. Metallurgist and Assayer.

1906 Tennant, Aubrey, M.E. Lancaster G. M. Co., Ltd., P. O. Box 347, Krugersdorp. Surveyor.

1905 Tennant, Vincent. Glencairn G. M. Co., Ltd., P. O. Box 191, Germiston. Amalgamator.

1903 Tennent, Bruce Edward, M.Aus.I.M.E. Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Metallurgical Chemist.

1907 Terry, Charles Warwick, New Found Out Mine, Gadzema, Rhodesia. Mining Engineer.

1908 Thomas, Abram. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Smelter.
(Assoc. 1902)

1904 Thomas, Frederic J. Surprise G. M. Co., Ltd., Selukwe, Rhodesia. Chemist and Cyanide
(Assoc. 1898) Manager.

1907 Thomas, Harry Musson. City & Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Mining Engineer.

1902 Thomas, James. Mines Department, P. O. Box 1132, Johannesburg. Inspector of Explosives.

1898 THOMAS, James Edward, A.I.M.M., M.Am.I.M.E. *(Member of Council).* Simmer Deep, Ltd.,
(Assoc. 1898) P. O. Box 178, Germiston. Reduction Works Manager.

Year of Election.	
1899	Thomas, James Frederick, A.I.M.M. Ginsberg G. M. Co., P. O. Box 16, Knights. Mine Manager.
1908 (Assoc. 1905)	Thomas, John. Cyanide Manager. (*Address wanted.*)
1904	Thompson, George de Roos. Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Cyanide Manager.
1899	Thomson, Samuel Clifton, M.I.M.M. Messrs. S. Neumann & Co., P. O. Box 485, Johannesburg. Mining Engineer.
1909 (Assoc. 1908)	Thorburn, John Mackay. Heymann's Laboratories, Fox Street, P. O. Box 3427, Johannesburg. Analytical Chemist.
1903	Thurlow, John Robert, A.I.M.M. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Metallurgist.
1903	Thurston, George H., A.S.M.E. Consolidated Gold Fields of S. A., Ltd., P. O. Box 1167, Johannesburg. Assistant Consulting Mechanical Engineer.
1906	Tippett, John M. Portland G. M. Co., Colorado Springs, Colorado, U.S.A. Chief Chemist and Assayer.
1909	Tonnesen, Karl Oscar, C.E. Knight Central, Ltd., P. O. Box 6647, Johannesburg. Construction Engineer.
1908	Tonnesen, Otto Stuhaug. East Rand Proprietary Mines, Ltd., 4 London Wall Buildings, London, E.C. Mine Surveyor.
1909 Assoc. 1906)	Toombs, Christopher. East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand. Metallurgical Chemist.
1907	Torin, Richard M., A.I.M.M. P. O. Box 28, Benoni. Mining Engineer.
1902 (Assoc. 1897)	Torrente, Miguel. P. O. Box 1891, Johannesburg. Consulting Metallurgist.
1899	Townsend, Edward Charles, A.I.M.M. Cyanide Manager. (*Address wanted.*)
1902	Trevor, Tudor G. Mines Department, P. O Box 401, Pretoria. Inspector of Mines, Transvaal.
1904 (Assoc. 1902)	Truscott, Samuel John, A.R.S.M., M.I.M.M., F.G.S. (*Life Member*). 65 Aberdare Gardens, South Hampstead, London, N.W. Mining Engineer.
1906	Turner, Arthur Harold. Luipaardsvlei Estate & G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Cyanider.
1905	Turvey, William Edward. P. O. Box 192, Germiston. Mining Engineer.
1909	Twining, David Hopkins. Rosetta Sheba G. M. Co., P. O. Box 136, Barberton. Mine Captain.
1902	Upton, Prescott, A.I.M.C.E., M.I.M.E., F.S.A., F.I.S. Mining Engineer. (*Address Wanted.*)
1903	Vaughan, James Evelyn, A.R.S.M., A.I.M.M. P. O. Box 204, Boksburg. Deputy Inspector of Mines.
1904	Veasey, John Herbert, A.I.M.M. Gaika Mine, Que Que, Rhodesia. Manager.
1904 (Assoc. 1903)	Viney, Arthur Donald, B.Sc. P. O. Box 3233, Johannesburg. Assayer.
1903	Visser, Wilhelmus Hendrikus. Johannesburg Consolidated Investment Co., Ltd., P. O. Box 231, Johannesburg. Mining Engineer and Government Land Surveyor.
1905	Wade, Richard Armytage. P. O., Bulawayo, Rhodesia. Surveyor.
1907	Walker, John Fairfax. P. O. Box 30, East Rand. Tube Mill Foreman.
1907	Wallace, Alexander. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Engineer.
1909	Walsh, Peter Arthur. Ferreira Deep, Ltd., P. O. Box 2473, Johannesburg. Engineer.
1904	Ward, Joseph Philip. Aurora West G. M. Co., Ltd., P. O. Box 26, Maraisburg. Mine Manager.

Year of Election.

1899 Warren, Howard. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Mining Engineer.

1908 Warwick, John Abraham, D.S.O. Bush Tick Mines, Ltd., Private Bag, Bushtick Siding, *via* Bulawayo, Rhodesia. Mining Engineer and Mine Owner.

1903 Wason, Hugh, A.G.T.C. East Rand Proprietary Mines, Ltd., P. O Box 37, East Rand. Assayer.

1903 Watson, Edward South. Simmer & Jack East, Ltd., P. O. Box 47, Germiston. Mine Sampler.

1903 Watson, Frederick William, B.Sc., F.I.C. Consolidated Goldfields Laboratory, P. O. Box 108, Germiston. Analytical Chemist.

1909 Watson, James Henry Arnand. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Electrician.

1897 Watson, John, F.I.C. (by Exam.). City & Suburban Estate & G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Chief Assayer and Chemist.

1908 Way, Edward John, M.I.M.M., A.M.I.C.E. Kleinfontein Group, Benoni. Consulting Engineer.

1904 Wayne, Thomas Henry Babington. Claremont Mine, Insiza, Claremont private bag, Bulawayo. Metallurgist.

1899 Webb, Harry Howard, M.I.M.M. Consolidated Gold Fields of S. A., Ltd., 8, Old Jewry, London. Mining Engineer.

1909 Webb, Herbert Douglas. Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

1895 Webber, George Edward, M.I.M.M. Rand Mines, Ltd., P. O. Box 1056, Johannesburg. General Manager.

1909 Webster, David. Robinson Deep G. M. Co., Ltd., P. O. Box 6316, Johannesburg. Electrician.

1902 Weiskopf, Erich, Ph. D. Dynamite Factory, Modderfontein. Chief Laboratory Chemist.

1907 Welch, J. O. P. O. Box 1935, Johannesburg. Foreman Timberman.

1909 Wells, Frank. Robinson Deep G M. Co , Ltd., P. O. Box 1488, Johannesburg. Amalgamator.

1909 Wernher, Sir Julius C , Bart. 1, London Wall Buildings, London, E.C. Financier.

1909 Wessels, Johannes Jacobus. Nigel Deep, Ltd., P. O. Box 50, Nigel. Mining Engineer.

1908 Weston, Eustace Moriarty. McCreedy Tin Mines, Ltd., Mbabane, Swaziland. Mine Manager.

(Assoc. 1904)
1905 Weston, Robert Ogilvy, M.I.M.M. Globe & Phoenix G. M. Co., Ltd., Que Que, Rhodesia. Engineer and Mill Manager.

1897 WHITBY, Arthur (*Member of Council*). Assay Offices, 10, Simmonds Street, P. O. Box 849, Johannesburg. Chemist and Assayer.

1895 White, Franklin, M.I.M.M. P. O. Box 669, Bulawayo, Rhodesia. Consulting Mining Engineer.

1897 WHITE, Henry Arthur (*Member of Council*). May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston. Reduction Works Manager.

1903 Whyte, Percy H. M. P. O. Box 679, Johannesburg. Cyanide Foreman.

1904 Wiley, Harry. Selukwe G. M. Co., Ltd , Selukwe, Rhodesia. Shift Boss.

1907 Wight, A. Randfontein Estates & G. M. Co , Ltd., P. O. Box 2, Randfontein.

1904 Wilkinson, Arthur. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Mill Foreman.

1903 Wilkinson, David, M.I.M.M. Messrs. S. Neumann & Co., 241, Salisbury House, London, E.C. Mining Engineer.

1902 WILKINSON, James Alfred, M.A. (Cantab.), F.C.S. (*Member of Council*). Transvaal University College, P. O. Box 1176, Johannesburg. Professor of Chemistry.

1897 Wilkinson, W. Fischer, M.I.M.M. 43, Threadneedle Street, London, E.C. Mining Engineer.

1905 Williams, Alpheus Fuller, B.Sc , A.S.M.E., A.I.M.M., A.I.M.E. De Beers Consolidated Mines, Ltd., Kimberley. General Manager.

Year of Election.

1902 WILLIAMS, Gerard W., A.I.M.M., F.C.S. (*Corresponding Member of Council*). c/o London *Times*, Falmouth Chambers, Pitt Street, Sydney, New South Wales, Australia. Analytical Chemist and Metallurgist.

1894 WILLIAMS, John Richard, M.I.M.M., M.I.M.E., M.Am.I.M.E. (*Past President and Life Member*). c/o Standard Bank of S. A., 10 Clements Lane, E.C. Consulting Metallurgist.

1908 Williams, Richard St. John Barnard. Hauser Tribute Co., P. O. Box 160, Johannesburg. Manager.

1909 Williams, Walter Owen. Meyer & Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg.
(Assoc. 1902) Cyanide Manager.

1908 Williamson, Robert. Vlei Street, Luipaardsvlei Township, Krugersdorp. Tube-miller.

1906 Williamson, William Duncan, M.I.M.M., M.Am.I.M.E. Mount Boppy G. M. Co., Ltd., Boppy Mountain, New South Wales, Australia. Metallurgical Engineer

1908 Willis, Harry Thomas, A.I.M.M. Champaign, Ill., U.S.A. Mining and Metallurgical Engineer.

1908 Wilmoth, Leslie James, A.I.M.M. Simmer & Jack Proprietary, Mines, Ltd., P. O. Box 192,
(Assoc. 1905) Germiston. Assayer.

1906 Wilson, Lyell Newton. Nigel Deep, Ltd., P. O. Box 50, Nigel. Cyanider.

1898 Wilson, Nathaniel, D.S.O., M.I.M.E., M.Am.I.M.E. 2, East India Avenue, Leadenhall Street, London, E.C. Consulting Mechanical Engineer.

1903 Winckworth, William Henry, Treasury G. M., Ltd., Cleveland. Cyanider.

1903 Winslow Gilbert, B.A. (Cantab.), A.I.M.M., A.M.I.C E., (*Life Member*). Messrs. Golden Read, Forde & Co., 75, Aldermanbury, London, E.C. Mining Engineer.

1905 Wisdom, George Edward. Yankee Doodle Mine, Selukwe, Rhodesia. Metallurgist.

1899 Wormald, Robert, M.I.M.E. Worcester Exploration & G. M. Co., Ltd., P. O. Box 85, Barberton. Mining Engineer and Manager.

1908 Wyndham, Herbert Henry. Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg. Cyanide Manager.

1899 YATES, Arthur, M.I.M.M. (*Corresponding Member of Council*). Lebong Donok, Benkoelen, Sumatra, Dutch East Indies. Metallurgist.

1896 YATES, John, A.R.S.M., M.I.M.M., F.G.S. (*Past President*). Transvaal University College, P. O. Box 1176, Johannesburg. Mining Engineer.

1904 Yeatman, Pope, M.E., M.I.M.M., M.Am.S.C.E., M.Am.I.M.E. Room 1714, No. 165, Broadway, New York, U.S.A. Mining Engineer.

1904 Yule, James Alexander, Langlaagte Estate & G. M. Co., Ltd., P. O. Box 98, Langlaagte. Mechanical Engineer.

ASSOCIATES.

Corrected to November 11, 1909.

Year of Admittance.

1909 Adams, Angus Robert. Lancaster West G. M. Co., Ltd., P. O. Box 360, Krugersdorp. Assistant Assayer.

1899 Alderson, George. Globe and Phœnix G. M. Co., Que Que, Rhodesia. Mine Secretary.

1908 Allan, John Richard. Village Deep, Ltd., Barnes Road, Brixton, Johannesburg. Miner.

1903 Allingham, John, A.I.M.M. San Pedro Guanacevi, Durango, Mexico. Cyanide Manager.

1897 Ancarani, Henri. 78, Rue d'Amsterdam, Paris, France. Managing Director.

1902 Anderson, Alexander Gillies, A.I.M.M. Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Metallurgical Chemist.

1908 Anderson, James Stuart. c/o A. J. Barlow, Samox, Victoria Road, Eccles, Manchester, England. Reduction Officer and Cyanide Chemist.

1908 Anderson, Robert. East Rand Proprietary Mines, Ltd., P. O. Box 30, East Rand. Tube-miller.

1908 Anderson, William. Antelope Mine, Matopos, Rhodesia. Surveyor.

1906 Andrews, Alan A. Forbes Reef, Swaziland. Cyanider.

1898 Andrews, Thomas William. Louws Creek, per private bag, Barberton. Mine Owner.

1898 Angus, Thomas. Civil and Mining Engineer. (*Address wanted*).

1905 Arderne, Ralph Alexander Sandilands. Bon Accord Mine, Greylingstad. Cyanider.

1908 Aspland, Christopher Hatton. Simmer Deep, Ltd., P. O. Box 178, Germiston. Mine Surveyor.

1906 Atkinson, John. (*Address wanted.*) Cyanider.

1907 Ayers, Gilbert Frederick. B.S.A. Explosives Co., Dynamite Factory, Modderfontein. Chemist.

1906 Baker, Harold Edward, A.R.S.M. Veracity Mine, Lower Gwelo, Rhodesia. Cyanider.

1909 Barrett, Sidney Stephen. Sheba G. M. Co., Ltd., Eureka. Amalgamator.

1904 Barry, Ernest Albert. Knights Central, Ltd., P. O. Box 91, Germiston. Mine Captain's Assistant.

1905 Bawden, Livingstone Booth Lawry. Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Assayer.

1907 Bayly, Charlton F. Knight's Deep, Ltd., P. O. Box 143, Germiston. Mill Foreman.

1909 Bell, Guy Folkard. Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

1904 Bell, Septimus Walter, B.Sc., A.I.M.M. Ashanti Goldfields Corporation, Ltd., Obuassi, *via* Sekondi, Gold Coast Colony. Assayer.

1909 Bell, Trevor Griffith. Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

1908 Bell, Willoughby George, B Sc. Crœsus South G. M. Co., Ltd., Kalgoorlie, West Australia. Metallurgist and Mine Manager.

1909 Berry, William James. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Carpenter.

1899 Bettel, Arthur Percy. corner Hay and Kennedy Streets, Turffontein. Cyanider.

1909 Bird, Frank Arthur. Ancona Cottage, Ivy Mine, Barberton. Cyanide Manager.

1905 Blomfield, Alfred Leslie, B.C.E. (Melb.), M.Aus.I.M.E., M.Am.I.M.E., Golden Cycle Mining Co., Colorado Springs, Colorado, U.S.A. Mining Engineer and Metallurgist.

1906 Blyth, William Birch. Great Fingall G. M., Ltd., Day Dawn, West Australia. Assayer.

1905 Bolitho, E. J. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel.

1904 Bosanquet, Florian Gustavus, B.Sc. c/o Rev. G. Bosanquet, Lyde Vicarage, Hereford, England. Assayer.

Year of Admittance.

1908 Bowyer, James. East Rand Proprietary Mines, Ltd., P. O. Box 131, East Rand. Mill Foreman.

1908 Brading, Thomas. Vectis Reef, P. O. Colleen Bawn, Gwanda, Rhodesia. Miner.

1897 Brangham, A. F. 193, Mackintosh Place, Roath Park, Cardiff.

1907 Brazier, Henry. Crown Mines, Ltd., P. O. Box 102, Fordsburg. Cyanider.

1905 Brickhill, Hector Gordon. Nigel Deep, Ltd., P. O. Box 50, Nigel. Mining Engineer.

1909 Bridgers, Marcus Milton. Sheba G. M. Co., Ltd., Eureka. Ore Roaster.

1899 Brodigan, Charles Bernard. Brakpan Mines, Ltd., P. O. Box 2, Brakpan. Mine Manager.

1904 Brown, Arthur Henry, M.E., M.Am.I.M.E. 133, Kent Street, London, Ont , Canada. Metallurgist.

1908 Brown, William Sinclair, M.I.M.M. Lebong Soelit, Ketaun, via Benkoelen, Sumatra, Dutch East Indies. Metallurgist.

1907 Brown, William Waugh. Mysore G. M. Co., Ltd., Marikuppam, Mysore State, India. Chief Reduction Officer.

1909 (Stud. 1908) Browne, Claude Melville, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider.

1908 Brunton, Charles. Theta G. M. Co , Ltd., Que Que, Rhodesia. Cyanider.

1908 Bullock, Laurence N.B. Butters Copala Mines, Copala, Estado de Sinaloa, Mexico. Metallurgist.

1906 Burmester, Gustav. B S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Chief Engineer.

1907 Burrows, William Edmund. Crown Mines, Ltd., P. O. Box 53, Fordsburg. Amalgamator.

1905 Burt, Edwin. El Oro Mining and Railway Co., Ltd., El Oro, Estado de Mexico, Mexico.

1907 Burt, Wilfrid. Wanderer Mine, Selukwe, Rhodesia. Sampler.

1907 Cameron, Lindsay Duncan. Greenmount Mines, Southern Cross, West Australia. Metallurgist.

1907 Cameron, Sidney James. Robinson Deep G. M. Co., Ltd., P. O. Box 2772, Johannesburg. Assistant Assayer.

1906 Campbell, Hugh. Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Amalgamator.

1898 Carbis, William. Sheba G. M. Co , Ltd., Eureka. Battery Manager.

1904 Cardwell, Robert Holgate. Lizant Rectory, Launceston, Cornwall. Mine Sampler.

1903 Carter, James Wilton. (*Address wanted.*) Assistant Assayer.

1909 Cawthorne, John Russell. Cinderella Deep, Ltd., P. O. Box 82, Boksburg. Amalgamator.

1906 Chapple, Albert John Morphett, A.I M.M., M.Aus.I.M.E. Occidental G. M. Co., Cobar, New South Wales. Metallurgist.

1909 Chomley, William Burgh. Day Dawn, West Australia. Cyanider.

1909 (Stud. 1906) Clarence, John Vaughan Gerald. Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Mining Engineer.

1904 Clark, Allan Jay, E.M., M.I.M.M., M.Am.I.M.E. Homestake Mining Co., Ltd., P. O. Box 481, Lead, South Dakota, U.S.A. Chief Assayer and Reduction Officer.

1903 Clayton, Robert Henry, B.Sc. Manchester Oxide Co., Ltd., Canal Street, Miles Platting, Manchester, England. Chemist.

1904 Clennell, John Edward, B.Sc. (Lond.), A.I M.M. 6, Great James Street, Bedford Row, London, W.C. Metallurgical Chemist.

1909 Coalbank, Vincent Richards. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider.

1907 Cochrane, A. C. Block "A," P. O. Box 23, Randfontein. Sampler.

1906 Coe, Walter Charles. The Rowdy Boys Mine, near Eldorado, South Rhodesia. Mine Manager.

1907 Cogle, Charles Thomas. New Kleinfontein Co., Ltd., P. O. Box 2, Benoni. Mining Engineer.

Year of Admittance.

1898 Comrie, Alexander. Rosalind Mine, Essexvale, South Rhodesia. Cyanide Manager.

1909 Cook, Basil. Vogelstruis Consolidated Deep, Ltd., P. O. Box 3, Florida. Assayer.

1897 Cook, Junius F. Messrs. S. Neumann & Co., P. O. Box 485, Johannesburg. Mechanical Engineer.

1898 Coppin, Joseph Tuman. South Randfontein G. M. Co., Ltd., P. O. Box 18, Randfontein. Cyanide Manager.

1907 Cox, Charles Thomas. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

1909 Crea, John M. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tubemiller.

1898 Cribbes, George C. Robinson Randfontein G. M. Co., Ltd., Randfontein. Cyanider.

1906 Crocker, Cecil John. Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

1905 Cronin, John. P. O. Box 4800, Johannesburg. Cyanider.

1905 Daniels, Ivor Charles. Rose of Sharon and Shamrock G. M., Ltd., Gwelo, Rhodesia. Assayer.

1899 Davidson, John. New Unified G. M. Co., Ltd , P. O. Box 5, Maraisburg. Cyanider.

1906 Davidson, John Lindsay. The Eiffel Blue Mine, Gatooma Siding, Rhodesia. Cyanide Manager.

1904 Davis, Robert. Waterfall Syndicate Mill, Penhalonga, Rhodesia. Mill Manager.

1897 Day, Alfred Moggridge. Roodepoort United Main Reef G. M. Co., Ltd., P. O. Box 115, Roodepoort. Cyanide Manager.

1904 Deakin, Charles Edward. Messrs. Deakin & Furnival, Orchard Chambers, Church Street, Sheffield, England. Amalgamator.

1906 Dennehy, Michael John. corner Alexander Road and Biccard Streets, Turffontein. Cyanider.

1903 Devenish, Charles William. Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort. Assayer.

1905 Dewar, Charles Lachlan. Ferreira G. M. Co., Ltd., P. O Box 3554, Johannesburg. Mill Foreman.

1909 Dick, Otto Emile. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Mill Foreman.

1909 Dickson, Victor Richard. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider.

1909 Dixon, John McKay. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assistant Mill Foreman.

1909 Donald, Henry Stone Hutcheon. Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Mining Engineer.

1905 Dorr, John van Nostrand, B.Sc., A.I.M.M., M.Am.I.M.E. 204, Bank Block Buildings, Denver, Colorado, U.S.A. Metallurgical Engineer.

1908 Dowling, Arthur Edward. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Millwright.

1908 Dowling, Douglas. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.

1908 Dowsett, Charles William. Knights Central G. M. Co., Ltd., P. O. Box 91, Germiston. Cyanide Foreman.

1904 Dudgeon, William Francis Henley. Consolidated Gold Fields of South Africa, Ltd., P. O. Box 67, Bulawayo. Mining Engineer.

1909 Duncan, William Scott. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Fitter.

1907 Dunnachie, William James Nimmo. Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Sampler.

1909 Durell, Harold Lancelot Vavasour. East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand. Surveyor.

1898 Durham, Benjamin Walter. Bushtick Mines, Ltd., *via* Bulawayo. Metallurgist.

Year of Admittance.

1908 Easten, Gordon Travers. East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Tube Mill Foreman.

1906 Edginton, David, Simmer and Jack Proprietary Mines Ltd., P. O. Box 192, Germiston. Amalgamator.

1905 Ellis, Bernard. New Rietfontein Estate G. M. Co., Ltd., P. O. Rietfontein Mines. Sampler.

1906 Evans, Albert Charles. Theta Mine, Que Que, Rhodesia. Secretary.

1898 Evans, Samuel. Messrs. H. Eckstein & Co., P. O. Box 1067, Johannesburg. Director of Companies.

1898 Evans, Walter Bowen, M.I.M.M. Victorian Cornish Gold Mines, Ltd., Daylesford, Victoria, Australia. Metallurgical Chemist.

1904 Ferguson, Archibald McAulay. 20a, Johannes Street, Troyeville, Johannesburg. Civil Engineer and Surveyor.

1903 Fiander, Robert. Geldenhuis Estate and G. M. Co., Ltd., P. O. Box 5, Cleveland. Cyanider.

1906 Finlay, Norman Walker. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.

1906 Fletcher, Harold Clarkson. P. O. Box 224, Bulawayo, Rhodesia. Mechanical Engineer.

1908 Foote, Arthur B., Jr. North Star Mine, Grass Valley, California, U.S.A. Mining Engineer and Superintendent.

1909 Fox, George William. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1907 Fox, Herbert W. United States Reduction and Refining Co., Colorado Springs, Colorado, U.S.A., Superintendent.

1904 Francis, William. South Arab Mine, Penhalonga, Rhodesia. Manager.

1908 Freusberg, Adolph. Mynbouw Maatschappij "Siman," Lebong Tandai (Ketaun), West Coast Sumatra, Dutch East Indies. Assayer.

1897 Fricker, Robert George. Consolidated Gold Fields of S.A., P. O. Box 198, Johannesburg. Director of Companies.

1905 Gemmell, David. Transvaal G. M. Estates, Ltd., Pilgrims Rest. Cyanider.

1908 Gepp, Herbert William, A.I.M.M. de Bavay Treatment Co., Ltd., Broken Hill, New South Wales. Chemist.

1907 Gerardy, Baron George. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

1906 Gibb, John Alexander Philip, A.I.M.M. Anglo-French Exploration Co., Ltd., P. O. Box 2927, Johannesburg. Mine Surveyor.

1897 Gibbings, George William, F.C.S. Standard Bank of S A., Ltd., P. O. Box 190, Salisbury, Rhodesia. Bank's Assayer.

1898 Gibson, James, M.I.M.E., A.S.M. (Ballarat). Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Engineer and Surveyor.

1904 Graham, William Hastie. Gaika Mine, Queque, Rhodesia. Cyanide Manager.

1902 Gray, James Wensley. Waihi G. M. Co., Ltd., Waihi, New Zealand. Metallurgical Chemist.

1906 Greathead, Cyril Harding. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Shift Boss.

1909 Greig, Douglas William. B.S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Technical Chemist.

1908 Grierson, Norman Victor. Higginsville, Western Australia. Metallurgist and Mining Engineer.

1905 Gross, John, E.M., M.Am.I.M.E. 417, Jackson Building, Denver, Colorado, U.S A. Mining and Metallurgical Engineer.

Year of Admittance.

1903 Grumitt, P. H. P. O. Box 1677, Johannesburg. Cyanider.

1908 Halford, James Edward. Midas Deep G. M. Co., Randfontein. Amalgamator.

1898 Hall, William Frederic. Wanderer Gold Mines, Ltd., Selukwe, Rhodesia. Assayer and Cyanide Manager.

1902 Hamill, James Clark. New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Cyanider.

1897 Hamilton, Edward Montague, B.A. (Oxon.), M.I.M.M. Messrs. Charles Butters & Co., 333, Kearney Street, San Francisco, Cal., U.S.A.. Consulting Metallurgist.

1906 Hamilton, William. 45, Scarsdale Villas, Kensington, London, W. Mining Engineer.

1908 Hanna, Joseph. Treasury G. M., Ltd , Cleveland. Cyanider.

1902 Hardy, James Gordon, F.C.S., M.Am.I.M.E , A I.M.M. Dolores Mining Co., Dto. Guerrero, San Isidro, Chihuahua, Mexico. Manager, and Mining and Metallurgical Engineer.

1906 Harrison, Oliver. East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Cyanider.

1904 Hart, James Arthur. London & S. A. Agency, Ltd , P. O. Box 765, Johannesburg. Mechanical Engineer.

1908 Hauton, George. Union Club, Eloff Street, Johannesburg. Mill Manager.

1907 Hawthorne, James Winchester. Kerang, Victoria, Australia. Chief Sampler.

1906 Hay, Alexander. c/o Bank of Africa, Johannesburg. Cyanider.

1904 Heeley, Alec William. Crocodile Reef, Essexvale, Rhodesia. Mine Surveyor.

1903 Hiddleston, John. Langlaagte Estate & G. M. Co., Ltd., P. O. Box 140, Fordsburg. Cyanider.

1903 Hill, John Whitelaw. 26 Newark Drive, Pollockshields, Glasgow, Scotland. Analytical Chemist and Metallurgist.

1907 Hogenraad, Gerard Bernard. Gold Exploration Co., Lebong Kandis, *via* Post Ketahoen, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1898 Homersham, Edwin Collet, M.I.M.M. Randfontein Estates, Ltd., Randfontein. Mining Engineer.

1898 Hopkins, Herbert William. Waihi G. M. Co., Auckland, New Zealand. Metallurgist.

1909 Hopkins, Joseph Huxley. Consolidated Rand Brick, Pottery & Lime Co., Ltd., P. O. Box 609, Johannesburg. Business Manager and Secretary.

1899 Hunt, Bertram, F I.C., M.Am.I.M.E., M.I.M.M. 709 Mills Buildings, San Francisco, California, U.S.A. Analytical Chemist and Metallurgist.

1909 Hunter, William John Russell. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192. Germiston. Cyanider.

1907 Ingle, Frederick Charles William. Cyanider. (*Address wanted.*)

1908 Ingram, John. Durban Roodepoort G. M. Co., Ltd., Roodepoort, P. O. Box 111, Roodepoort. Assayer.

1905 Irwin, Harold Beresford. Niwri, Armstrong Street, N., Ballarat, Victoria, Australia. Amalgamator.

1904 Jackson, George Thomas. Alaska Perseverance Mine, Juneau, Alaska, U.S.A. Assistant Manager.

1898 James, Alfred, M.I.M.M., F.G S. 2, Broad Street Place, London, E C. Consulting Mining and Metallurgical Engineer.

1908 Jameson, Charles Godfrey. Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Mining Engineer.

1906 Jane, Walter Hooper. Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

1898 Jansen, Henry. Robinson Randfontein G. M. Co., Ltd., Randfontein. Cyanide Manager.

1905 Jarvis, William. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

Year of Admittance.

1906 Jenkins, Charles Warren Bowen, B.E., M.I.M.E. Hillview, Turramurra, New South Wales, Australia. Mining Engineer and Metallurgist.

1904 Jennings, T. Bryant. P. O. Box 1565, Johannesburg. Mining Engineer.

1909 Johnson, Cecil Robert. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1898 Johnston, James. Butters' Cupola Mines, 30 Church Street, New York, U.S.A. Engineer.

1907 Jones, George Francis. Beresford Buildings, Johannesburg. Chemist.

1909 Jones, William Thomas. P. O. Box 28, Pilgrims Rest. Mine Overseer.

1907 Jones, Wyndham Arthur. Gaika Mine, Que Que, Rhodesia. Cyanider.

1909 Jupp, Herbert Lauder. French Rand G. M. Co., Ltd., P. O. Box 30, Luipaardsvlei. Cyanider.

1909 Kay, Herbert George, Premier Diamond Mining Co., P.O. Box 182, Premier Mine. Surveyor.

1909 King, Andrew. Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

1908 Kirkland, Henry George. Geldenhuis Deep. Ltd., P. O. Box 54, Cleveland. Clerk.

1902 Krause, Herbert Louis. A S.M. (Ballarat). Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg. Mining Engineer.

1898 Kyle, John Tullis. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assayer.

1906 Ladds, Lawrence Henry. Bell Mine, Que Que, Rhodesia. Cyanider.

1905 Lakeland, William John. Burma Mines, Ltd., Bawdwin, *via* Hsipaw, N. Shan States, Burma. Mining Engineer.

1905 Laret, Harry. Zeerust Lead Proprietary Mine, Ltd., P. O. Box 44, Zeerust. Mining Engineer.

1906 Lathbury, Frank Herbert. P. O. Box 213 Bulawayo, Rhodesia. Mining Engineer.

1909 Lean, Thomas Frank Rodgers. Occidental G. M. Co., Ltd., Lewis Street, Cobar, New South Wales. Assayer.

1897 Lee, Ernest. Crown Mines, Ltd., P. O. Box 1145, Johannesburg. Cyanider.

1903 Leng, Robert William. 30 Primrose Hill, Skinningrove, Carlin How, R.S.O., England. Miner.

1909 Lewis, Arthur. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Millman.

1908 Lindberg, Birger. Mutual Life Office, Helsingfors, Finland. Mining Engineer.

1908 Linton, Robert. 616 Bixel Street, Los Angeles, California, U.S.A. Metallurgist.

1898 Locke, Arthur Nutter. New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Cyanider.

1908 Lofts, Horace F. Inglenook, Bradford Road, Trowbridge, Wilts, England. Metallurgist.

1909 Lomberg, Henry. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assistant Smelter.

1898 Lucke, Percy Kent, A.I.M.M. Apartado 320, Monterrey, Mexico. Surveyor and Assayer.

1909 Macaulay, Murdo. Mount Morgan Mine, P. O. Box 95, Barberton. Miner and Tributor.

1906 Macfarlane, Thomas Murdoch Marshall, A.I.M.M. c/o Messrs. H. S. King & Co., 65, Cornhill, London. Miner, &c.

1908 Macfarlane, Walter Graeme. Jumbo Mine, Mazoe, Rhodesia. Assayer and Surveyor.

1897 Macintyre, James, B Sc., C.E. Tati Concessions, Ltd., Francistown, Bechuanaland Protectorate. Mining Engineer and General Manager.

1904 Mackay, Alexander Neil, A.R.S.M., A.I.M.M. Cherokee Goldfields, Ltd. San Julian, *via* Parral, Chihuahua, Mexico. Assayer, Chemist and Assistant Manager.

1909 Mackilligan, Hector Rannie. South Kalgurlie G. M. Co, Fimiston, W. Australia. Assayer.

Year of Admittance.

1905 Mann, William Seward, B.Sc., Met. E., M.I.M.M., M.Am.I.M.E. El Conde Mining Company, S. A., Tepehuanes, Durango, Mexico. General Manager.

1898 Mannheim, Hermann, Charles. P. O. Box 585, Bulawayo, Rhodesia. Metallurgist.

1905 Marriott, Frederick Armidale, A.S.M. (Ball.), M.Aus.I.M.E., M.Am.I.M.E. Lancefield G. M. Co., Laverton, W. Australia. Metallurgist.

1898 Marshall, Charles C., M.I.M.E Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs. Mining Engineer and Surveyor.

1909 Martin, William Alexander. Robinson Deep G. M. Co., Ltd , P. O. Box 1488, Johannesburg. Cyanider.

1902 Massey, Frank Byrne. P. O. Box 41, Jeppestown. Cyanider.

1909 Masterson, Charles William. Subeni Copper Mine, Vryheid, Natal. Assayer and Surveyor.

1898 Maxwell, F. A. G., M.I.M.M. Randfontein Estates G. M. Co., Ltd., P. O. Box 2, Randfontein. Cyanide Manager.

1907 Maxwell, Robert Kerr. Chemist. (*Address wanted.*)

1907 Maxwell, Robert William. The Walls, Hampton Court, Middlesex, England. Mechanical Engineer.

1909 McAlister, Alister Gordon. Jumpers Deep, Ltd., P. O. Box 1. Cleveland. Assistant Surveyor.

1897 McConnell, Thomas, Sheba G. M. Co., Ltd., Eureka. Cyanide Foreman.

1898 McCormick, A. R. Pilgrims Rest. Cyanider.

1908 McCrindle James. Knights Deep, Ltd., 15 Long Street, Germiston West. Amalgamator.

1903 McDonnell, Henry Ernest. Elands Drift, P. O. Sabie. Assayer.

1904 Meeser, J. N. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Contractor.

1898 Mein William Wallace, B.Sc., M.Am.I.M.E., M.I.M.M. Messrs. H. Eckstein & Co., Central Administration, P. O. Box 1145, Johannesburg. Consulting Engineer.

1909 Meredith, Vaughan Wilson. Princess Estate & G. M. Co., Ltd., P. O. Box 112, Roodepoort. Mill Manager.

1897 Merrill, Charles W., B.Sc., M.I.M.M., M.Am.I.M.E. 143, Second Street, San Francisco, California, U.S.A. Metallurgical Engineer.

1897 Michell, Richard T. Standard Bank of S.A., Ltd., Bulawayo, Rhodesia. Assistant Assayer.

1898 Michie, James. Cyanider. (*Address Wanted.*)

1898 Miller, Joseph, M.Am.I.M.E. Mangana, Tasmania. Mining Engineer.

1906 Millroy, Alfred Taggart, F.R.Met.S. c/o Lyman, Sons & Co., St. Paul Street, Montreal, Canada. Chemical and Metallurgical Analyst.

1905 Mills, Edwin Walter, S.B.(Harvard), A.I.M.M., M.Am.I.M.E., Korean Exploration, Co., Chiksan Mines, Chiksan, Korea. Mining Engineer and Superintendent.

1903 Mills, Louis David, M.E., M.Am.I.M.E. 502, Rincon Buildings, 143, Second Street, San Francisco, U.S.A. Mining Engineer and Metallurgist.

1908 Moncrieff, Roger Murray. 21, Kensington Park Gardens, London, W. Mining Engineer.

1905 Moore, Charles George James. Ferreira G. M. Co., Ltd., P. O. Box 3554, Johannesburg. Cyanider

1903 Moore, Karl Byron. 421 Collins Street, Melbourne, Australia. Mill Foreman.

1907 Moore, William Robert, P. O. Box 2586, Johannesburg. Foreman Mason.

1898 Murdoch, Robert. Kimberley Reefs, Mazoe, Rhodesia. Cyanide Manager.

1908 Murphy, Thomas Daniel. c/o The Aguacate Mines, San Mateo, Costa Rica, Central America. Mining Engineer.

1907 Murray, George Daly. Ivanhoe Gold Corporation, Fimiston, Western Australia. Metallurgist.

1909 Nakamoto, Hidehiko. Sado Gold Mine, Aikawa, Sado, Japan. Sub-Manager.

Year of Admittance.	
1909	Nalty, Frederick John. P. O. Box 253, Bulawayo, Rhodesia. Cyanider.
1908	Nellmapius, Arthur Philip. Messrs. Eckstein & Co., P. O. Box 149, Johannesburg. Mining Engineer.
1909	New, William Stanley. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Underground Learner.
1905	Newcomb, Clive Seymour. c/o Messrs. White & Newcomb, 32 Avenida Cinco de Mayo, Mexico City, D. F. Mining Engineer.
1907	Newland, Noel Nelson. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.
1908	Nicholas, Dan. Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Assayer.
1907	Nicholls, John Carroll. New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Assistant Mine Captain.
1909	Nixon, George Henry. P. O. Box 11, Pilgrim's Rest. Mine Overseer.
1908	O'Connor, Bernard. P. O. Box 6510, Johannesburg. Cyanider.
1905	Odgers, William Hearn. Village Deep, Ltd., P. O. Box 1145, Johannesburg. Cyanider.
1898	Ogilvie, Duncan F. Glencairn G. M. Co., Ltd., P. O. Box 191, Germiston. Mill Manager.
1905	Olsen, Arnold Carl Louis, M.Inst.M.E. Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Engineer.
1898	Ovendale, Charles. Steytler's Buildings, P. O. Box 3146, Johannesburg. Patent Agent.
1904	Owen, Harold Alfred. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Cyanider.
1898	Packard, George Arthur, B.S., M.Am.I.M.E. 18 Lafayette Street, Wakefield, Mass., U.S.A. Mining Engineer and Metallurgist.
1909	Packman, Augustus Frederick. Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanider.
1908	Parfitt, Walter. Cinderella Deep G. M. Co., P. O. Box 75, Boksburg. Cyanider.
1897	Park, Walter E. Messrs. Fraser & Chalmers, Ltd., P. O. Box 619, Johannesburg. Mechanical Engineer.
1897	Parry, Charles Stuart. Robinson Randfontein G. M. Co., Ltd., P. O. Box 46, Randfontein. Assayer.
1909	Parsea, Arthur Edward. Sheba G. M. Co., Ltd., Eureka. Vanner Foreman.
1904	Patrick, William. Elandsdrift, P. O. Sabie. Mining Engineer and Surveyor.
1903	Peebles, James Gibb. Geldenhuis Estate & G. M. Co., Ltd., P. O. Box 32, Cleveland. Cyanider.
1898	Peele, Robert, E.M., M.I.M.M. 36 Wall Street, New York City, U.S.A. Mining Engineer and Adjunct Professor of Mining, Columbia University School of Mines.
1903	Perry, Mulford Frank. c/o Messrs. Chas. Butters, Ltd., Minas Prietas Works, Torres, Sonora, *via* Nogales, Mexico. Cyanide Manager.
1907	Pinto, Michele. Komarovo Mining Co., Avzianopetrovsk, Government Orenbourg, Russia. Manager.
1909	Plinke, August Adolph Phillip. P. O. Box 273, Johannesburg. Director of Mining Companies.
1909	Pooler, Frederick John, B.Sc., F.C.S. Jeppe High School, 358 Commissioner Street, Johannesburg. Science Teacher.
1907	Pope, John. Brakpan Mines, Ltd., P. O. Box 3, Brakpan. Surveyor and Assayer.
1908	Powrie, Donald E. Rose Deep, Ltd., P. O. Box 6, Germiston. Cyanider.
1907	Powter, Harold Bowen. Knights Deep, Ltd., P. O. Box 145, Germiston. Engineer.

Year of Admittance.

1908 Pryce, Wynn St. Lawrance. Treasury G. M. Ltd., Cleveland. Assayer.

1909 Ramsden, Ernest. Sheba G. M. Co., Ltd., Eureka. Cyanider.

1905 Ranson, William Rowell. Transvaal Hotel, Krugersdorp. Cyanider.

1909 Read, Frederick George. Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Mill Manager.

1908 Reece, Frederick Bennett, A.I.M.M P. O. Box 955, Lead, S. Dakota, U.S.A. Assistant, Refinery Department.

1909 Richardson, John Stephen. Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Mining Engineer.

1903 Richardson, Robert. c/o Thos. Andrews, private bag, Barberton. Mining Engineer.

1908 Richardson, Willie Park. Forbes Reef, near Mbabane, Swaziland. Amalgamator.

1907 Rhodes, Clarence Edgar. Guanajuato, Estado de Guanajuato, Apartado 25, Mexico. Cyanide Manager.

1899 Roberts, Frank G. A. Knights Central, Ltd., P. O. Box 91, Germiston. Mine Manager.

1907 Robertson, John Rankin Hastie, A.I.M.M. Rezende, Ltd., and Penhalonga Proprietary Mines, Ltd., Penhalonga, Umtali, Rhodesia. Metallurgist.

1905 Robertson, Thomas. Glencairn Main Reef G. M. Co., Ltd., P. O. Box 191, Germiston. Miner.

1902 Robinson, Clifford A. Meyer & Charlton G. M. Co., P. O. Box 1127, Johannesburg. Assayer and Metallurgist.

1907 Robinson, Sydney Greenwood. c/o F. L. Smidth & Co., 112 Palace Chambers, Westminster, London, S.W. Engineer.

1898 Roessler, Dr. Friedrich Johann. Deutsche Gold and Silver Scheider Austalt, Frankfort a/m Chemist. Germany.

1909 Ross, Arthur William. P. O. Box 9, Day Dawn, Western Australia. Mining Engineer.

1906 Rossiter, David William. Consolidated Gold Fields of S.A., Ltd., P. O. Box 1167, Johannesburg. Head of Intelligence Dept.

1908 Roy, James. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Cyanider.

1904 Rusden, Armine Grose. Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Assistant Surveyor.

1897 Rusden, Harry. Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Chief Assayer.

1906 Russell, William. Assistant Assayer. (*Address wanted.*)

1906 Ryan, William. Lonely Reef, Inyati, *via* Bulawayo, Rhodesia. Mine Manager.

1898 Saunders, Frederick William. Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Mill Manager.

1909 Sawyer, Ronald Ernest. East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand. Mining Engineer.

1903 Sayers, Joseph. Ardeer Factory, Stevenston, Ayrshire, Scotland. Technical Chemist.

1899 Seaer, Valentine Edmund, M.E., M.I.M.M. P. O. Box 69, Bulawayo, Rhodesia. Mining Engineer and Mine Manager.

1908 Scatterty, Peter. Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort. Cyanide Manager.

1909 Schaefer, Benjamin. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

1909 Scholtz, Ewald Vos. P. O. Box 28, Pilgrims Rest. Mining Student.

1908 Scott, Andrew. P. O. Box 2361, Johannesburg. Assayer.

1897 Searle, Barry. 36 Lake Avenue, Montrose, Susy Co., Penna., U.S.A. Consulting Mining Engineer.

1904 Seear, Herbert Charles. Chicago Mine, Umswezwe, Rhodesia. Assayer and Surveyor.

Year of Admittance.

1906 Sharpley, Harold. Broken Hill South Blocks, Ltd., Broken Hill, New South Wales, Australia. Mining Engineer.

1899 Sharwood, William John, Ph.D , A.R.S.M., M.I.M.M., M.Am.I.M.E., M.Am.C.S. Homestake Mining Co., Lead, S. Dakota, U.S.A. Metallurgical Chemist.

1906 Shaw, John Frederick. Cassilis G. M. Co , Tongio West, Victoria, Australia. Assayer.

1907 Shed, William Burns. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.

1908 Short, Arthur Reginald. Crown Mines, Ltd., P. O. Box 36, Fordsburg. Cyanider.

1907 Simpson, Charles Edward, Concession Hill Mine, Hartley, Rhodesia. Mine Tributor.

1904 Smith, Ewing. 4 Queen Street, Craigie, Perth, Scotland. Chemist.

1897 Smith, Rowland. The Big Blow, Umswezwe, Rhodesia. Cyanide Manager.

1908 Smith, Sidney A. Modderfontein "B" G. M. Co., Ltd., P. O. Box 14, Brakpan. Miner.

1905 Smith, William James Wright. Wanderer Mine, Selukwe, Rhodesia. Cyanider.

1907 Soeterik, Hendrik Willem Cornelis, M.I.M.M. Mijnbouw Maatschappij, Redjang Lebong, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1906 Southwell, Bailey. York Mine, P. O. Box 54, Krugersdorp. Sampler.

1903 Spicer, Harold Norman, M.I.M.M., M Am.I.M.E. 434 Symes Building, Denver, Colorado, U.S.A. Consulting Mining Engineer.

1908 Stacey, Douglas William. P. O. Box 188, Bulawayo, Rhodesia. Amalgamator.

1905 Stadler, Herman. P. O. Box 4240, Johannesburg. Architect-Engineer.

1905 Staver, William H. M.Am.I.M.E.. Freeport, Ills, U.S.A. Mining Engineer.

1904 Steele, William H., Cyanider and Smelter. *(Address wanted.)*

1906 Stevens, Thomas Barnbrook. Bewick, Moreing & Co., P. O. Box 57, Kalgoorlie, Western Australia. Metallurgist.

1909 Stidolph, Neville Harold. Yankee Doodle Mine, Selukwe, South Rhodesia. Amalgamator.

1898 Stockett, A. W., M.Am.I.M.E., A.M.Am.S.C.E. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Civil and Mining Engineer.

1905 Summers, Edward. P. O. Box 94, Klerksdorp. Assayer and Cyanider.

1906 Surmon, Cecil Barber. *(Address wanted.)* Sampler.

1905 Sutherland, Robert. Transvaal G. M. Estates, Ltd., Pilgrims Rest. Mining Engineer.

1909 Symonds, Melville Carter. East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

1904 Tait, William Raitt. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mining Engineer.

1909 Taylor, Hamish Ernest Oliff-. Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanider.

1908 Thayer, Harry Stanley. U.S. Reduction and Refining Co., Ltd., Colorado Springs, Colorado, U.S.A. Assistant Superintendent.

1907 Thomas, Alfred Messer. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assistant Assayer.

1908 Thomas, David John. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Amalgamator.

1906 Thomas, David Lewis. P. O. Box 108, Johannesburg. Cyanider.

1909 Thomas, George Gordon. Wolhuter G. M., Ltd., P. O. Box 1160, Johannesburg. Sampler.

1904 Thomas, Henry Trenear, A.I.M.M., M.Am.I.M.E. Champion Reef G. M. Co., Champion Reefs, Mysore State, S. India. Chemist and Metallurgist.

1898 Thomas, Joseph. Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights. Cyanide Manager.

Year of Admittance.	
1907	Thomas, William. Champion Reef G. M. Co., Champion Reefs, Mysore State, S. India. Assistant Cyanide Chemist.
1909	Thomas, Victor Israel. Wrightville, Cobar, New South Wales. Cyanider.
1907	Thorburn, Frederick Harrison, A.R.S.M. Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Assayer.
1907	Thorpe, William Edward. East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand. Assistant Metallurgical Chemist.
1902	Tillard, Maximilian Ogilvie., M.E. (Cape). Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Mining Engineer and Mine Captain.
1908	Tipper, Loton. Nigel G. M. Co., Ltd , P. O. Box 20, Nigel. Cyanider.
1907	Tom, Isadore, A.R.S.M., F.G.S., M.Am.I.M.E., A.I.M.M. Associated Gold Mines of W. A., Kalgoorlie, Western Australia.
1906	Tonkin, John Lewis, A.I.M.M. Mysore G. M. Co., Marikuppam, S. India. Reduction Officer.
1909	Townsend, Charles Thomas. Brilliant Extended Cyanide Works, Charters Towers, Queensland. Metallurgist and Cyanide Manager.
1905	Trapman, Lewis Alexander. Morgan Crucible Co., Ltd., Battersea Works, London, S.W. Director.
1908 (Stud. 1907)	Turner, Hubert Frederick Sidney. New Primrose G M. Co., Ltd., P. O. Box 193, Germiston. Sampler.
1906	Urquhart, William Graham. Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.
1908	Wacher, Harry. Jumbo Mine, Mazoe, Rhodesia. Cyanider.
1903	Wallace, Harry Vincent, M.Am.I.M.E. Nogales, Santa Cruz Co., Arizona, U.S.A. Consulting Mining Engineer and Metallurgist.
1909	Ward, Harry. Vogelstruis Estates & G. M. Co., Ltd., P. O. Box 116, Roodepoort. Amalgamator.
1904	Warden, Philip S. Ardpatrick Mine, Queque, Rhodesia.
1905	Wartenweiler, Fred., M.Am.I.M.E., A.I.M.M. Granite-Bimetallic Consolidated Mining Co., Philipsburg, Montana, U.S.A. Metallurgist.
1898	Warriner, Ruel Chaffee, B.Sc., M.Am.I.M.E. Crown Mines, Ltd., P. O. Box 1145, Johannesburg. General Manager.
1904	Waters, William. B.S.A. Explosives Co., Ltd., P. O. Box 4575, Johannesburg. Explosives Representative.
1906	Watkins, H. T. Glynn's Lydenburg, Ltd., P. O. Sabie. Surveyor.
1898	Watts, John Whidbourne, J.P., M.I.M.E., F.R.C.I. United Ivy Reef G. M. Co., Ltd., P. O. Box 179, Barberton. Mining Engineer.
1898	Waymark, Edgar C. Messrs. Fraser & Chalmers, Ltd., London Wall Buildings, London Wall, London, E.C. Mechanical Engineer.
1897	Weldon, Horace. Mines Department, P. O. Box 331, Johannesburg. Chief Inspector of Mines.
1907	Welhaven, Alf. Oriental Consolidated Mining Co., Tabowie, Unsan, Korea. Mine Superintendent.
1906	Wertheim, Clement Jean Marie. Hilversum, Holland. Mining Engineer.
1909	Westwood, William. Robinson G. M. Co., Ltd., 4, Central Road, Fordsburg. Cyanider.
1908	Wheler, Aubrey Stuart, A.I.M.M. c/o S. W. Jameson, 52, Ashley Gardens, Westminster, London, S.W. Mining Engineer.
1902	White, Thomas Ayrton. Simmer & Jack Proprietary Mines, Ltd , P. O. Box 192, Germiston. Secretary.
1907	Whitehouse, James. Vryheid, Natal. Mining Engineer.
1906	Whittinghame, James. Cyanider. (*Address wanted.*)

Year of Admittance.

1909 Whittuck, William Henry Giffard. The Troitzk Goldfields, Ltd., Kotchkar, Orenburg Govt., Russia. Metallurgist.

1903 Whytock, Peter Robertson, B.Sc. P. O. Box 19, Rawhide, Nevada, U.S.A. Mine Reporter.

1907 Wilkie, Donald Cook. Straits Trading Co., Ltd., Kuala Lumpur, Selangoe, Federal Malay States. Mining and Electrical Engineer.

1909 Willcox, Alexander. New Rietfontein Estates G. M. Co., Ltd., P. O. Rietfontein Mines. Tubemill Foreman.

1906 Williamson, Reginald, S.M.B. Great Boulder No. 1, Ltd., Mount Magnet, Western Australia. Assayer.

1903 Wilson, John Hawthorne. Simmer Deep, Ltd., P. O. Box 178, Germiston. Engineer.

1907 Wilson, Peter. Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Foreman.

1897 Wilson, Robert Alexander. Eldorado G. M. Co., Ltd., Lomagundi, Rhodesia. Cyanider.

1908 Wiseman, Edward James. Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Foreman Trammer.

1906 Wisnom, Robert James. c/o C. Butters & Co., Ltd., Virginia City, Nevada, U.S.A. Cyanider.

1908 Wolvekamp, Hendrik. Ketahoen, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

1904 Womble, Lloyd Alexander, M.Am.I.M.E. Van Dyk Mines, P. O. Box 115, Boksburg. Mine Manager.

1902 Wood, Gilbert Peter. Assayer. (*Address wanted.*)

1909 Wood, Sydney. Sheba G. M. Co, Ltd., Eureka. Amalgamator.

1906 Woodard, Bernard Nathaniel. Bonanza Mine, Blanket Junction, Lomagundi, Rhodesia. Surveyor.

1904 Woodburn, James Allan, M.I.M.M., M.I.M.E. Messina Development Co, Ltd., Grenfell Camp, *via* Pietersburg. Mine Manager.

1904 Woodley, Reginald C. Cyanider. (*Address wanted.*)

1908 Woods, Wallace. East Rand Proprietary Mines, Ltd., P. O. Box 185, East Rand. Tube Mill Foreman.

1898 Wright, Alexander Leckie. Vogelstruis Consolidated Deep, Ltd., P. O. Box 3, Florida. Cyanide Manager.

1903 Young, Henry Stow, A.I.M.M. Borneo Co., Ltd., Bau Gold Mines, Sarawak, Borneo. Mining and Metallurgical Engineer.

STUDENTS

Corrected to November 11, 1909.

Year of Admittance.	
1908	Ball, Harry Standish. P. O. Box 2536, Johannesburg.
1909	Bass, Kenneth Christian. Three Sisters Mine, via Barberton. Assay Learner.
1906	Bell, Herbert Charles Fitzwilliam. French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei.
1909	Bolt, Walter Carl. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.
1909	Botha, Fritz Alexander. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tube mill Learner.
1909	Brennan, Ernest. New Kleinfontein Co., Ltd., P. O. Box 2, Benoni. Cyanide Learner.
1909	Chadwick, Herbert Hamilton. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.
1909	Coombe, Oswald. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Assay Learner.
1909	Craig, Arthur William. Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Learner.
1908	Davies, Cyril Nutcombe. Holcombe, Jubilee Road, Parktown, Johannesburg.
1908	Donovan, Donough Bennet. 103, De Korte Street, Wanderer's View, Johannesburg.
1908	Gaisford, Arthur. Princess Estate & G. M. Co., Ltd., P. O. Box 112, Roodepoort.
1908	Gilbert, Thomas Woodford. 47, Bok Street, Jouberts Park, Johannesburg.
1908	Graham, Harry Woodthorpe. P. O. Box 1155, Johannesburg.
1909	Greyson, Robert. Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanide Learner.
1909	Hains, Arthur Rutherford Bolt. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg.
1909	Hasselt, Carel Louis van. Transvaal University College, College House, 42, Berea Road, Bertrams, Johannesburg.
1905	Innes, John. The Observatory, Johannesburg.
1909	Impey, Robert Lancelot. Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanide Learner.
1907	Johnston, William Hawthorn. 94, Sherwell Street, Doornfontein, Johannesburg.
1908	Lipschitz, Cecil. Transvaal University College, P. O. Box 1176, Johannesburg.
1909	Little, George Heard. Transvaal University College, P. O. Box 1176, Johannesburg.
1909	Lotter, Reginald Godfrey. Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanide Learner.
1908	Malherbe, David Gabriel. *(Address wanted.)*
1909	McCaffery, Cecil J. Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.

Year of Admittance.	
1909	McNeil, William. Wolhuter G. M., Ltd., P. O. Box 1160, Johannesburg. Cyanide Learner.
1909	Medlin, Edward James George. Rose Deep, Ltd., P. O. Box 6, Germiston. Tube-mill Learner.
1908	Mills, Thomas Landall. Royal School of Mines, London, S.W.
1908	Moon, Cyril Leslie. Transvaal University College, P. O. Box 6695, Johannesburg.
1909	Mortimer, Norman. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.
1909	Munro, John Strachan. Transvaal University College, P. O. Box 1176, Johannesburg.
1909	Musson, Gordon. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube-mill Learner.
1909	O'Brien, Horace Vincent. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.
1909	Radley, Andrew. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Reduction Works Learner.
1909	Reeler, William George. Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Extractor House Learner.
1909	Robinson, John Martyn. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube-mill Learner.
1909	Rowe, John Lafflin. Nigel Deep, Ltd., P. O. Box 50, Nigel. Assay Learner.
1909	Smith, Harold James. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.
1909	Stewart, Victor Mallet James. Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.
1908	Sutton, Joseph Bennett. P. O. Box 1176, Johannesburg.
1908	Tapscott, Sidney. (*Address wanted.*)
1903	Wagner, Percy Albert, P. O. Box 1176, Johannesburg.
1909	Webb, Willmer. Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assay Learner.

TOPOGRAPHICAL INDEX OF MEMBERSHIP.

AFRICA.

TRANSVAAL.

Johannesburg—

Adair, A.
Adams, A. E.
Adams, H. A.
Aiken, A.
Albu, G.
Alexander, H. J.
Allan, J. R.
Allen, R.
Allen, W. H.
Anderson, W. T.
Andreoli, G.
Atkinson, R. C.
Auret, A. A.
Avent, A.

Ball, H. S.
Bateman, E. L.
Bay, B.
Beaton, S.
Beatty, J. W. S.
Behr, H. C.
Bell, G. F.
Bennett, R. W.
Bentley, F. W.
Betty, W. K.
Bevington, R. G.
Blackmore, J. G.
Bleloch, W. E.
Blume, E. E. A.
Blundun, B. V.
Bosqui, F. L.
Bowen, H. O.
Bowness, A. J.
Brakhan, A.
Bristol, F. A.
Broom, W.
Browning, E.
Bruun, A. F.
Burnett, G. L.

Caldecott, W. A.
Cameron, S. J.
Cameron, W. McC.
Caris, S.
Carr, W.
Carruthers, J. T.
Carter, B. P.
Chadwick, H. H.
Chandler, W. M.
Chaplin, F. D. P.
Chilton, J.
Christopherson, D.
Cindel, F. W.
Clarence, J. V. G.
Cobbett, W. G. P.
Cohen, L.
Cook, J. F.
Corstorphine, G. S.
Cotter, A. J.
Creswell, F. H. P.
Croghan, E. H.
Cronin, J.
Cropper, C. H.
Crosse, A. F.
Curtis, J. S.

Dalrymple, W.
Darling, G. A.
Davies, C. N.
Dawe, W. H.
Dewar, C. L
Dinwoodie, J. H.
Dodd, H.
Dodds, J. B. K.
Donovan, D. B.
Dowling, D.
Dowsett, R. B.
Doyle, M. J.

Edwards, A. L.
Evans, A. W.
Evans, L.
Evans, S.

Ferguson, A. Mc. A.
Ferguson, J. F.
Ferris, G. G.
Ferris, V. G.
Fox, G. C.
Frerichs, J. A.
Fricker, R. G.
Furner, E. A.

Gaze, J.
Gerlings, H.
Gibb, J. A. P.
Gilbert, T. W.
Gillies, A. S.
Gilmore, A. D.
Gordon, W. S.
Graham, H. W.
Gray, J.
Grumitt, P. H.

Hains, A. R. B
Hall, R. E.
Hambly, F. N.
Hamilton, W. L.
Hanau, C.
Hardwick, W.
Harland, H. L.
Hart, J. A.
Hartley, A. H.
Hasselt, C. L. v.
Hastings, B. J.
Hauton, G.
Hay, A.
Hay, E.
Henderson, J. McC.
Heymann, A.
Higham, J.
Hilner, E. G.
Hollis, E.
Honnold, W. L.
Hopkins, J. H.
Houghton, J. H.
Hulsteyn, Sir W. van

Inglis, M. C.
Innes, J.
Irvine, L. G.

Jennings, T. B.
Jones, G. F.
Johns, J. H.
Johnson, A. J.
Johnson, H. H.
Johnson, H. M. L.
Johnson, T.
Johnston, W. H.

Johannesburg (continued)

Jolly H. R.
Jolly, T. S.
Jongh, J. N. de.
Jordan, T. W.

Kennedy, J.
Kern, P. E.
King, A. C.
Kingston, C. B.
Kotzé, R. N.
Kisch, C. H. M.

Lace, J. D.
Lamont, C. H.
Langermann, M.
Laschinger, E. J.
Lea, H.
Lea, J.
Lee, E.
Leitch, Q. J.
Lenz, O.
Leslie, C. D.
Lipschitz, C.
Little, G. H.
Littlejohn, J.

Macfarlane, T. J. M.
MacGregor, R. D.
Martin, H. S.
Martin, W. A.
McCracken, J.
McCrae, J.
McGuire, J. E.
McKellow, W. H.
McMillan, Q. C.
McNiel, W.
Meeser, J. N.
Mehliss, M.
Mein, W. W.
Meyer, H.
Michelmore, P. J.
Mitchell, W. E. C.
Moir, J.
Moon, C. L.
Morison, S.
Moore, C. G. J.
Moore, W. R.
Mortimer, N.
Morton, W. D.
Mumford, F. T.
Munro, J. S.
Murray, M. T.
Musson, G.

Napier, W. F.
Nel, P. O.
Nellmapius, A. P.
Newhall, P. M.
Newland, N. N.
Newman, A. D.
Newton, S.
Nicholas, D.
Nicklin, W.
Nicolaus, R. C.
Niven, R.

O'Connor, B.
Odgers, W. H.
Olsen, A. C. L.
Orr, J.
Ovendale, C.
Owen, H. A.

Pam, E.
Park, W. E.
Parker, W. F.
Pearce, S. H.
Pepler, D. J.
Plinke, A. A. P.
Pooler, F. J.
Pope, W. T.
Potter, H. S.
Price, H. B.
Pyles, J. F.

Quinn, J. W.

Raine, R.
Reyersbach, L.
Richardson, J. S.
Robeson, A. M.
Robinson, C. A.
Robinson, J. M.
Roche, H. F.
Rosendorff, H.
Ross, O. D.
Rossiter, D. W.
Rusden, A. G.
Rusden, H.

Sale, G. G.
Saner, C. B.
Scarf, H. A.
Schlesinger, J.
Schmitt, C. O.
Schumacher, R. W.
Scott, A.
Shlom, S.
Sim, A. C. B. M.
Simson, L.
Smith, H. J.
Smith, J. A.
Solomon, L.
Spengel, H. G.
Spoor, A. L.
Stadler, H.
Stanley, G. H.
Stevens, J. H.
Stewart, V. M. J.
Stokes, R. S. G.
Stout, W. H.
Strange, H. F.
Streeter, J. R.
Sutton, J. B.

Taylor, H.
Thomas, D. L.
Thomas, G. G.
Thomas, H. M.
Thomas, J.
Thomson, S. C.
Thorburn, J. M.
Thurlow, J. R.
Thurston, G. H.
Tonnesen, K. O.
Torrente, M.

Viney, A. D.
Visser, W. H.

Wagner, P. A.
Wallace, A.
Walsh, P. A.
Warren, H.
Warriner, R. C.
Waters, W.
Watson, J.
Watson, J. H. A.
Webber, G. E.
Webster, D.
Welch, J. O.
Weldon, H.
Wells, F.
Whitby, A.
Whyte, P. H. M.
Wilkinson, A.
Wilkinson, J. A.
Williams, R. St. J. B.
Williams, W. O.
Wiseman, E. J.

Yates, J.

Alkmaar—

Baumann, M.

Balfour—

Maclatchy, A. G.

Barberton—

Andrews, T. W.
Bass, K. C.
Bird, F. A.
Brown, G. D.
Burnett, D. V.

Barberton (continued)
Cannon, A. B.
Chew, B.
Inglis, A. B.
Lowdon, G. W.
Macaulay, M.
Richardson, R.
Smith, M.
Taylor, W.
Twining, D. H.
Watts, J. W.
Wormald, R.

Benoni—
Anderson, P. v. M.
Barrett, P.
Barron, W. E.
Brennan, E.
Cogle, C. T.
Erskine, J. K.
Fleischer, S. C. W.
Holford, W. G.
Hutton, C. E.
Kerr, J.
Kerr, T.
Krige, W. A.
Lichstentein, A.
Locke, A. N.
MacCarthy, G. D. R.
Nicholls, J. C.
Robinson, R. F.
Russell, S.
Torin, R. M.
Way, E. J.

Boksburg—
Beatty, G. H.
Browne, C. M.
Cawthorne, J. R.
Coalbank, V. R.
Coombe, O.
Dickson, V. R.
Gardiner, C. T.
Greyson, R
Hilditch, C. H.
Jones, J. A.
Parfitt, W.
Price, W. S. V.
Rand, E. T.
Sly, A. W.
Taylor, R. McG.
Vaughan, J. E.
Womble, L. A.

Brakpan—
Brodigan, C. B.
Klein, M. M.
Pope, J.
Smith, S. A.

Cleveland—
Adler, H. W.
Aimetti, S. C. B.
Allen, H. E.
Brunton, W. O.
Coll, D.
Coulter, W. McC.
Dick, O. E.
Fiander, R.
Finlay, N. W.
Gibson, J.
Graham, K. L.
Hallimond, W. T.
Hanna, J.
Harris, J. H.
Henderson, H.
Hunter, G. J.
Jameson, C. G.
Kirkland, H. G.
Lee, C. W.
Lindsay, R.
Macaulay, D.
McAlister, A. G.
McCaffery, C. J.
Mossop, W. C.
Peebles, J. G.
Petersen, H. T.
Pidsley, H. W.
Pryce, W. St. L.
Richardson, A.
Robertson, D.
Roe, W. H.
Rogers, N.
Shed, W. B.
Simpson, C. B.
Tillard, M. O.
Winckworth, W. H.

Delmore—
Eisenstaedter, J. E.

Denver—
Carter, G.
Cazalet, P.
Matthews, E. L.
Packard, H. T.

East Rand—
Allen, F.
Anderson, G. S. S.
Anderson, R.
Atkin, A. J. R.
Beaver, W.
Berry, W. J.
Bisset, A. C.
Bowyer, J.
Corder, C. A.
Cox, C. T.
Dowdle, P. J.
Dowling, A. E.
Duncan, W. S.
Durell, H. L. V.
Easten, G. T.
Harrison, O.
Jarvis, W.
Johnson, E. H.
Jory, J. V.
Lawson, G. A.
Lloyd, W. D'A.
Meyer, C. E.
Mitchell, J. T.
Morris, J.
Morris, J. W. H.
Piele, W.
Sawyer, R. E.
Scotland, R. D.
Scott, L. E.
Stewart, A. F.
Strang, J.
Symonds, M. C.
Thorpe, W. E.
Toombs, C.
Walker, J. F.
Wason, H.
Woods, W.

Eureka—
Barrett, S. S.
Bridgers, M. M.
Carbis, W.
Hesom, W. W.
Hill, H.
McConnell, T.
Parsea, A. E.
Ramsden, E.
Wood, S.

Florida—
Cook, B.
Fleming, J.
Leyson, W.
Healey, J. E.
Wright, A. L.

Fordsburg—
Alexander, F. F.
Brazier, H.
Burrows, W. E.
Buuren, W. van.
Coaton, A. A.
Herring, F.
Hiddleston, J.
Hilliard, C. B.
Jarman, E. A.
Lothrop, P. D.
Mather, T. W.

Fordsburg (continued)

Short, A. R.
Taggart, J.
Westwood, W.

Geduld—

Pearson, T.

Geldenhuis—

Simmons, F. L.

Germiston—

Adams, A. C.
Aspland, C. H.
Atkin, R. W. R.
Barclay, G. M.
Barrie, D. McL.
Barry, E. A.
Bayly, C. F.
Bawden, F. A.
Bell, T. G.
Bolt, W. C.
Botha, F. A.
Butler, J. J.
Butler, W. M.
Craig, A. W.
Crea, J. M.
Crocker, C. J.
Dixon, J. McK.
Dowling, W. R.
Dowsett, C. W.
Dunning, F. G.
Edginton, D.
Edwards, P. L.
Evans, T.
Ferguson, M.
Fisher, J. S.
Fox, G. W.
Frey, F. C.
Gerardy, B. G.
Gibson, A.
Glyn, C.
Grant, J.
Greathead, C. H.
Guthrie, F. G.
Hamill, J. C.
Hardach, E. E.
Herald, A. J.
Herbert, S. A.
Howard, H. A. L.
Hamilton, W.
Hunter, W. J. R.
Impey, R. L.
Jane, W. H.
Johnson, C. R.
Johnston A. McA.
Johnston, H.
King, A.
Kyle, J. T.
Lewis, A.
Lockhart, T. L.
Lomberg, H.
Lotter, R. G.
Macleod, D. McK.
Marquard, J. D.
McCrindle, J.
Medlin, E. J. G.
Metcalf, J. E.
Michaux, D.
Morrell, H. H.
Mosenthal, B. P.
Murdoch, D.
New, W. S.
Nicholson, A. S.
Nicol, G.
O'Brien, H. V.
Ogle, F. B.
Ogilvie, D. F.
Osborn, S. S.
Osterloh, E. A.
Parry, C. F.
Payne, A. E.
Phillips, F. D.
Philpott, J. W.
Powell, O. P.
Powrie, D. E.
Powter, H. B.
Pryce, L.
Radley, A.
Reeler, W. G.
Roberts, F. G. A.
Robertson, T.
Robinson, L. J.
Ross, D. G. F.
Schaefer, B.
Sharp, W. P. F.
Simpson, F. L.
Smart, G. O.
Smith, W. H.
Solly, B. C. T.
Stockett, A. W.
Sydenham, C. T.
Tait, W. R.
Tayler, W. A. C.
Taylor, J. A.
Tennant, V.
Thomas, A.
Thomas, A. M.
Thomas, D. J.
Thomas, J. E.
Turner, H. F. S.
Turvey, W. E.
Urquhart, W. G.
Watson, E. S.
Watson, F. W.
Webb, H. D.
Webb, W.
White, H. A.
White, T. A.
Wilmoth, L. J.
Wilson, J. H.
Wilson, P.

Greylingstad—

Arderne, R. A. S.

Jeppestown—

Brown, J.
Massey, F. B.
Sharwood, W. S.

Kaapsche Hoop—

Acutt, S.
Browne, E.

Klerksdorp—

Hanlon, C.
Milligan, J. T.
Sharp, E. M.
Smythe, J. J. R.
Summers, E.

Knights—

Anderson, A. G.
Bawden, L. B. L.
Chapman, F. T.
Coats, J.
I'Ons, J.
Johns, J.
Kewley, R. R.
Morrisby, P. T. B.
Perkin, D. W.
Phillips, J. C.
Prout, W. M.
Rascher, L. H.
Rogers, A.
Salkinson, A.
Saunders, F. W.
Tennent, B. E.
Thomas, J.
Thomas, J. F.

Krugersdorp—

Adams, A. R.
Bartlett, S. G.
Bell, H. D.
Campbell, H.
Christopherson, E. H.
Davis, C. R.

Krugersdorp (continued)

Garny, F.
Goodwin, G.
Goodwin, H.
Lightbody, T.
Lomas, A. A.
Manners, C. E.
Murning, J.
Pedrini, J.
Penman, W.
Pill, R.
Ranson, W. R.
Schwabe, S. B.
Semple, W. B.
Southwell, B.
Tennant, A.
Turner, A. H.
Williamson, R.

Langlaagte—

Blackwell, C.
Coombe, M. H.
Daniel, J.
Dunnachie, W. J. N.
Ford, F. C.
Hebbard, A.
Hunter, A. M.
Martin, E. H.
Reynolds, J.
Simmons, T. W.
Thompson, G. de R.
Yule, J. A.

Luipaardsvlei—

Bell, H. C. F.
Brown, J.
Campbell, H.
Fraser, A.
Jupp, H. L.
Mahaut, H.
McNaughtan, A.
Melvill, G.
Nicholls, C. B.
Smith, E.

Malelane—

Bagshaw, J. F.

Maraisburg—

Davidson, J.
Douglas, R. S.
Ford, M.
Krause, H. L.
Lewis, F. B.
Quin, S. C.
Ward, J. P.
Wyndham, H. H.

Modderfontein—

Ayers, G. F.
Burmester, G.
Colquhoun, L.
Cullen, W.
Donaldson, T.
Greig, D. W.
Weiskopf, E.

Nigel—

Bolitho, E. J.
Brickhill, H. G
Cellier, J. S.
Curnow, R.
Morris, T. W.
O'Hara, J. D.
Robson, J. N.
Rowe, J. L.
Roy, J.
Thorburn, F. H.
Tipper, L.
Wessels, J. J.
Wilson, L. N.

Pietersburg—

Woodburn, J. A.

Pilgrims Rest—

Busch, G. E.
Cusworth, W. H.
Elphinstone, E.
Gemmell, D.
Griffiths, W. P.
Hunter, C.
Jones, W. T.
McCormick, A. R.
Neale, A. L.
Nixon, G. H.
Phillips, J. M.
Read, F. G.
Scholtz, E. V.
Sutherland, R.

Potchefstroom—

Newson, J. A.

Premier Mine—

Brunton, W. G.
Kay, H. G.

Pretoria—

Marks, S.
Trevor, T. G.

Randfontein—

Adendorff, J. E.
Bacon, A. D.
Cochrane, A. C.
Coppin, J. T.
Cribbes, G. C.
Halford, J. E.
Homersham, E. C.
Jansen, H.
Lee, H. J.
Maxwell, F. A. G.
Mills, F.
Parry, C. S.
Roberts, S.
Sellers, J. H.
Taggart, T.
Wight, A.

Rietfontein—

Bennett, W. F.
Clarkson, R.
Ellis, B.
Willcox, A.

Roodepoort—

Anderson, P.
Day, A. M.
Devenish, C. W.
Donald, H. S. H.
Gaisford, A.
Grix, H. R.
Hoyer, A. G.
Ingram, J.
Johnson, J. H.
Lacy, E. D. de
Macdonald, F. G.
McLennan, J.
Meredith, V. W.
North, W. J. R.
Packman, A. F.
Scatterty, P.
Steel, W. S.
Steels, S. H.
Taylor, H. E. O.
Ward, H.

Sabie—

McDonnell, H. E.
Patrick, W.
Watkins, H. T.

Schweizer Reneke—

Arkell, D. J.

Springs—

Boright, S. H.
Boyd, A.
Cullen, M.
Marshall, C. C.
Nicholl, T. T.
Sartorius, K. K. H.

Standerton—
Jones, W. H. R.

Turffontein—
Bettel, A. P.
Creed, J. P.
Dennehy, M. J.

Zeerust—
Laret, H.

BRITISH EAST AFRICA.

Molo (Mombassa)—
Drought, J. J.

CAPE COLONY.

Francistown (Bechuanaland)—
Macintyre, J.

Concordia (Namaqualand)—
Lewis, H. H.

Kimberley—
Williams, A. F.

Kokstad—
Blaine, G. R.

GERMAN SOUTH WEST AFRICA.

Swakopmund—
Hahn, W. A. A.

NATAL.

Vryheid—
Chambers, A. L.
Masterson, C. W.
Rouillard, R. A.
Whitehouse, J.

ZULULAND.

N'kandhla—
Hamill, J. H.

RHODESIA.

NORTH WESTERN RHODESIA.

Broken Hill—
Bentley, A. M.

Ndola—
Blackie, J.

SOUTHERN RHODESIA.

Belingwe—
Adams, E. L

Bulawayo—
Barratt, R. L.
Brett, H. T.
Byerley, B. G
Cumings, C. E. G.
Dixon, C.
Dudgeon, W. F. H.
Dyer, S. C.
Fletcher, H. C.
Gibson, J. R.
Hynd, J.
Johnson, A. D.
Lathbury, F. H.
Ludlow, L.
Mannheim, H. C.
Michell, R. T.
Nalty, F. J.
Nicholls, R.
Pead, C. H.
Popham, J. L
Scaer, V. E.
Stacey, D. W.
Wade, R. A.
White, F.

Bushtick Siding—
Durham, B. W.
Warwick, J. A.

Colleen Bawn—
Black, J.
Brading, T.
Boer, E. C.

Eldorado—
Coe, W. C.
Hurlbatt, C.
Stephan, H.

Essexvale—
Comrie, A.
Gosney, P.
Heeley, A. W.

Gadzema—
Terry, C. W.

Gatooma—
Brooks, E.
Davidson, J. L.
Hyland, L.
Macdonald, P. A.
Mack, J.

Golden Valley—
Morrisby, A. C.
Reynolds, J. P.

Gwanda—
Adamson, R.
Cantor, B. C. M.
Harris, J. V.

Gwelo—
Baker, H. E.
Daniels, I. C.
Normand, J. N.

Hartley—
Eaton, J. M.
Simpson, C. E.

Insiza—
Wayne, T. H. B.

Inyati—
Ryan, W.

Lomagundi—
Wilson, R. A.
Woodard, B. N.

Matopos—
Anderson, W.

Mazoe—
Macfarlane, W. G.
Murdoch, R.
Stiell, W
Wacher, H.

Penhalonga—
Davis, R.
Francis, W.
Patterson, P. F.
Robertson, J. R. H.
Robinson, R. W.

Queque—
Alderson, G.
Atkinson, C. E.
Brunton, C.
Cockburn, P. A.
Evans, A. C.
Graham, W. H.
Haddon, T.
Jones, W. A.
Ladds, J. H.
Veasey, J. H.
Warden, P. S.
Weston, R. O.

Salisbury—

Colville, C. A. B.
Cunningham, W.
Gibbings, G. W.
Jones, C. K. D.
Morkel, A. L. R.

Selukwe—

Burt, W.
Davies, E.
Dures, R.
Hall, W. F.
Little, J. B.
Mackintosh, H. G.
McEwen, M. T.
Mitchell, A.
Smith, W. J. W.
Stidolph, N. H.
Thomas, F. J.
Wiley, H.
Wisdom, G. E.

Shamva—

Calder, W. L.

Umswezwe—

Seear, H. C.
Smith, R.

West Nicholson—

Blewett, E.
Greene, F. M.

SWAZILAND.

Forbes Reef—

Andrews, A. A.
Richardson, W. P.

Mbabane—

Weston, E. M.

Piggs Peak—

Stokoe, J. C.

WEST AFRICA.

Obuassi—

Bell, S. W.

Tarkwa—

Rose, A. F.

AMERICA (NORTH).

CANADA.

London—

Brown, A. H.

Montreal—

Millroy, A. T.
Pencier, H. P. de

ALASKA.

Juneau—

Jackson, G. T.

MEXICO.

Chihuahua—

Hardy, J. G.
Mackay, A. N.

Copala—

Bullock, L. N. B.

Chavarria (Durango)—

Hughes, H.

Dinamita (Durango)—

Pollitt, R. B.

San Pedro Guanacevi (Durango)—

Allingham, J.

Tepehuanes (Durango)—

Mann, W. S.

El Oro—

Burt, E.
Main, A. F.
Neal, W.

Guanajuato—

Macdonald, B.
Rhodes, C. E.
Strout, E. A.

Mexico City—

Newcomb, C. S.

Monterrey—

Lucke, P. K.

Torres (Sonora)—

Perry, M. F.

UNITED STATES.

ARIZONA.

Nogales—

Wallace, H. V.

CALIFORNIA.

Berkeley—

Roberts, F. C.

Grass Valley—

Foote, A. B.

Los Angeles—

Crank, A. F.
Douglass, R. E.
Linton, R.

Oakland—

Garthwaite, E. H.

San Francisco—

Butters, C.
Drake, F.
Hamilton, E. M.
Hunt, B.
Merrill, C. W.
Mills, L. D.

Redding—

Reading, R. W.

Sutter Creek—

Darrow, W. E.

COLORADO.

Colorado Springs—

Blomfield, A. L.
Fox, H. W.
Taylor, G. M.
Thayer, H. S.
Tippett, J. M.

Denver—

Argall, P.
Dorr, J. van N.
Fitch, F.
Gross, J.
Spicer, H. N.

DELAWARE.

Seaford—

Morse, W. S.

ILLINOIS.

Champaign—
Willis, H. T.

Freeport—
Staver, W. H.

LOUISIANA.

Hammond—
Carter, T. L.

MASSACHUSETTS.

Wakefield—
Packard, G. A.

MISSOURI.

St. Louis—
Gazzam, J. P.

MONTANA.

Philipsburg—
Wartenweiler, F.

NEVADA.

McGill—
Sorensen, S. S.

Rawhide—
Whytock, P. R.

Reno—
Bristol, J. J.

Virginia City—
Wisnom, R. J.

NEW JERSEY.

Franklin Furnace—
Catlin, R. M.

NEW YORK.

New York—
Hellmann, F.
Jennings, S. J.
Johnston, J.
Peele, R.
Yeatman, P.

PENSYLLVANIA.

Montrose—
Searle, B.

SOUTH DAKOTA.

Lead—
Clark, A. J.
Reece, F. R.
Sharwood, W. J.

AMERICA (CENTRAL).

COSTA RICA.

San Mateo—
Murphy, T. D.

AMERICA (SOUTH).

ARGENTINE REPUBLIC.

Castano Nuevo—
Allen, J. A.

ASIA.

BORNEO.

Sarawak—
Young, H. S.

BURMA.

Bawdwin (Hsipan)—
Lakeland, W. J.

DUTCH EAST INDIES (SUMATRA)

Benkoelen—
Hövig, P.

Fort de Kock—
Grammell, P.

Ketahoen—
Wolvekamp, H.

Ketaun—
Brown, W. S.
Kreikhaus, K.

Lebong Donok—
Bayley, R. F. G.
Filet, G.
Foran, J. H.
Philp, H.
Soeterik, H. W. C.
Yates, A.

Lebong Kandis—
Hogenraad, G. B.

Lebong Siman—
Pinkvos, E.

Lebong Tandai—
Blau, M.
Freusberg, A.

Moera Aman—
Jansen, J. P.

NORTH CELEBES.

Poleleh—
Hailey, R. C.

MALAY PENINSULA.

Kuala Lumpur—
Dawbarn, O. S.
Wilkie, D. C.

Penang—
Laurie, E.

Singapore—
Mance, J. C.

INDIA.

Calcutta—
Macqueen, W. P. O.

Champion Reef—
Thomas, H. T.
Thomas, W.

Marikuppam—
Brown, W. W.
Leslie, H. M.
Tonkin, J. L.

JAPAN.

Sado—
Nakamoto, H.

KOREA.

Chiksan—
Mills, E. W.

Unsan—
Welhaven, A.

AUSTRALASIA.

AUSTRALIA.

NEW SOUTH WALES.

Boppy Mountain—

Williamson, W. D.

Broken Hill—

Gepp, H. W.
Sharpley, H.

Cobar—

Chapple, A. J. M.
Lean, T. F. R.
Thomas, V. I.

Pipeclay Creek (Pambula)—

Haugh, H.

Sydney—

Shallcross, V. F.
Williams, G. W.

Turramurra—

Jenkins, C. W. B.

QUEENSLAND.

Charters Towers—

Townsend, C. T.

VICTORIA.

Ballarat—

Irwin, H. B.

Carlton—

Mills, A. L.

Daylesford—

Evans, W. B.

Harrietville—

Evans, W. M.

Kerang—

Hawthorne, J. W.

Melbourne-

Moore, K. B.

Tongio West—

Shaw, J. F.

WESTERN AUSTRALIA.

Day Dawn—

Blyth, W. B.
Chomley, W. B.
Ross, A. W.

Fimiston—

Mackilligan, H. R.
Murray, G. D.

Higginsville—

Grierson, N. V.

Kalgoorlie—

Bell, W. G.
Stevens, T. B.
Tom, I.

Laverton—

Marriott, F. A.

Mount Magnet—

Williamson, R.

Southern Cross—

Cameron, L. D.

NEW ZEALAND.

Auckland—

Greer, J. C.
Hopkins, H. W.

Waihi—

Banks, E. G.
Gray, J. W.

TASMANIA.

Launceston—

Giblin, N. E.

Mangana—

Miller, J.

EUROPE.

BRITISH ISLES.

Bristol

Ford, S. H.

Cardiff

Brangham, A F.

arlin How—

Leng, R. W.

Chislehurst—

Parsons, C. E.

Craigie, N.B.—

Smith, E.

Glasgow—

Dykes, J.
Hill, J. W.

Grampound Road—

Minards, W. I. K.

Greenock—

Alston, R.
Galbreath, N. M.

Hampton Court—

Maxwell, R. W.

High Wycombe—

Fergusson, J.

Hurlford, N.B.—

McKerrell, D.

Kingsbridge—

Pearce, J.

Launceston—

Cardwell, R. H.

Llanybyther—

Evans, A. D.

London—

Bayldon, H. C.
Chew, R. W.
Claudet, A. C.
Clennell, J. E.
Collings, B. I.
Cooke, R. C. H.
Elmore, F. E.
Feldtmann, W. R.
Griffiths, A. P.
Griffiths, H. D.
Hamilton, W.
Heim, F.
Holloway, G. T.
Isaacs, R. McI.
James, A.

London (continued)
James, W. H. T.
Jones, H. E.
Macfarlane, T. M. M.
Marriott, H. F.
Mills, T. L.
Moncrieff, R. M.
Richards, F.
Robinson, S. G.
Suart, A. B.
Tonnesen, O. S.
Trapman, L. A.
Truscott, S. J.
Waymark, E. C.
Webb, H. H.
Wernher, Sir J. C.
Wheler, A. S.
Wilkinson, D.
Wilkinson, W. F.
Williams, J. R.
Wilson, N.
Winslow, G.

Lyde—
Bosanquet, F. G.

Manchester—
Anderson, J. S.
Clayton, R. H.

Scorrier—
Bawden, E. R.

Sheffield—
Deakin, C. E.

Sidcup—
Swinney, L. A.

Stevenston, N.B.—
Sayers, J.

St. Just—
Olds, H. F.

Trowbridge—
Lofts, H. F.

Truro—
Oliver, W. H.

Warrington—
Locker, J. T.

West Horsley—
Evans, G. S.

AUSTRIA.

Trieste—
Prister, A.

FINLAND.

Helsingfors—
Lindberg, B.

FRANCE.

Paris—
Ancarani, H.

GERMANY.

Berlin—
Loevy, J.

Frankfort—
Roessler, F. J.

Hamburg—
Nahnsen, G.

HOLLAND.

Hilversum—
Wertheim, C. J. M.

RUSSIA.

Avzianopetrovsk—
Pinto, M.

Kotchkar—
Whittuck, W. H. G.

Reval—
Avesson, R.

ADDRESSES WANTED.

Angus, T.
Angus, W. H.
Atkinson, J.
Aulsebrook, E. E.
Blaeser, C. A.
Carter, J. W.
Cunnack, J. T.
Durant, H. T
Ingle, F. C. W.
James, C. G.
Malherbe, D. G.
Maxwell, R. K.
Michie, J.
Royse, G. H.
Rusden, C. E.
Russell, W.
Stacpoole, A. R. W.
Steele, W. H.
St. Quintin, E. T.
Surmon, C. B.
Tapscott, S.
Thomas, John
Townsend, E. C.
Upton, P.
Whittinghame, J.
Wood, G. P.
Woodley, R. C.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. X. JULY, 1909. NO. 1.

Proceedings
AT
Ordinary General Meeting, July 17, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, July 17th, Mr. R. G. Bevington in the chair. There were also present:—

47 Members: Messrs. A. McA. Johnston, W. R. Dowling, K. L. Graham, E. J. Laschinger, A. Richardson, Prof. G. H. Stanley, H. A. White, W. A. Caldecott, W. Cullen, E. H. Johnson, H. A. Adams, Geo. Albu, D. J. Arkell, A. Avent, F. A. Bawden, W. Beaver, W. Broom, S. Caris, P. Carter, F. T. Chapman, J. Gray, F. G. Guthrie, J. H. Harris, S. A. Herbert, J. H. Johnson, G. A. Lawson, J. E. Metcalf, D. Murdoch, W. Nicklin, F. B. Ogle, A. Pill, E. T. Rand, W. H. Roe, D. G. F. Ross, H. A. Scarf, A. L. Spoor, R. S. Stokes, J. Taggart, H. Taylor, J. A. Taylor, A. Thomas, K. Tonnesen, O. Tonnesen, A. D. Viney, F. W. Watson, and Dr. E. Weiskopf.

12 Associates: Messrs. A. E. Adams, A. G. Anderson, J. Chilton, J. Cronin, D. W. Greig, T. B. Jennings, N. Newland, H. B. Powter, H. Stadler, A. M. Thomas, G. G. Thomas, and C. Toombs.

25 Visitors, including Sir T. Price, K.C.M.G., Mr. J. H. Dobson, President S.A.A. Engineers, and Mrs. and Miss Bevington, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the June *Journal*, were confirmed, after which Mr. Bevington formally introduced the new President, Mr. A. McArthur Johnston, who then took the chair.

NEW MEMBERS.

Prof. G. H. Stanley and Mr. A. Richardson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

ADAMS, ARTHUR E., Messrs. Loewenstein, Adams & Co., Ltd., P. O. Box 644, Johannesburg. Chemist. *(Transfer from Associate Roll.)*

CHANDLER, WALTER MARK, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Mining Engineer.

CUSWORTH, WILLIAM HENRY, Simmer Deep, Ltd., P. O. Box 178, Germiston. Construction Work Foreman.

DOWSETT, ROWLAND BERTRAM, New Goch G. M. Ltd., P. O. Box 1096, Johannesburg. Cyanide Manager.

GIBSON, ARTHUR, Knights Deep, Ltd., P. O. Box 143, Germiston. Smelter.

HERALD, ANDREW JAMES, Knights Deep, Ltd., P. O. Box 143, Germiston. Mill and Tube Foreman. *(Transfer from Associate Roll.)*

INGLIS, ALEXANDER BROWNLEE, Thornton Little G. M. Co., P. O. Box 56, Barberton. *(Transfer from Associate Roll.)*

LOMAS, ALBERT ARTHUR, Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Cyanide Manager.

McGUIRE, JAMES EDWARD, Jupiter G. M. Co., Ltd., P. O. Box 27, Germiston. Mine Manager.

MURRAY, MYLES THORNTON, Transvaal University College, Johannesburg. Lecturer and Demonstrator in Metallurgy and Assaying.

NAPIER, WILLIAM FLORANCE, Simmer Deep, Ltd., P. O. Box 4397, Johannesburg. Assistant Smelter.

NICOL, GILBERT, P. O. Box 11, Germiston. Cyanide Learner.

PHILLIPS, FREDERICK DAVID, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

PRICE, WALTER SAMUEL VINCENT, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Miner. *(Transfer from Associate Roll.)*

STEEL, WALTER SCOTT, Vogelstruis Estate and G. M. Co., Ltd., P. O. Box 116, Roodepoort. Cyanide Manager.

STEELE-NICHOLSON, ARTHUR, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Foreman. *(Transfer from Associate Roll.)*

TWINING, DAVID HOPKINS, Rosetta Sheba G. M. Co., P. O. Box 136, Barberton. Mine Captain.

WEBB, HERBERT DOUGLAS, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

WESSELS, JOHANNES JACOBUS, Nigel Deep, Ltd., P. O. Box 50, Nigel. Mining Engineer.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Durell, Harold Lancelot Vavasour, East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand. Surveyor.

Macaulay, Murdo, Mount Morgan Mine, P. O. Box 95, Barberton Miner and Tributor.

Sawyer, Ronald Ernest, East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand. Mining Engineer.

Stidolph, Neville Harold, Yankee Doodle Mine, Selukwe, S. Rhodesia. Millman.

As a Student.—

Webb, Willmer, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Learner in Assay Office.

General Business.

The President: We will now proceed to general business, and I will be pleased if any gentleman who has anything to bring forward will do so.

Mr. H. A. White (*Member of Council*): It has become a pleasant practice recently for short notes on new ideas to be brought forward here, and Mr. Solly, of the May Consolidated, has handed me the following to read :—

NOTES ON THE USE OF AIR LIFT PUMPS FOR UNDERGROUND WORK.

Mr. B. C. Travers Solly: I am informed that air lifts are not in common use underground on the Rand, and that is my excuse for bringing the small experience I have gained in that connection on the May Consolidated G. M. to the notice of members of this Society.

We were led to utilise them for the purpose of unwatering flooded winzes, by the fact that at the time a small pump was not available for the purpose. We used a 2 in. delivery pipe, to which was strapped a 1 in. air pipe. The latter was bent sharply at the bottom, and entered the delivery pipe for a distance of 3 ft. ; the end of the air pipe was flattened with the object of splitting up the air current to give the greatest possible aeration of the water. We started with one 18 ft. length of delivery pipe, with bend and short length attached at the top, and as the flow of water delivered decreased, added fresh lengths until the necessity for its use in that winze ceased, owing to an ordinary air driven pump being installed. The lift at this stage was a slope distance of 52 ft., in a 40° winze, when the surface of the water was $11\frac{3}{4}$ ft. above the air delivery, or a proportion of 81·5%. The rate of delivery at the start averaged about 600 gals. per hour for the first four hours, gradually decreasing as the distance between surface of water and end of air jet lessened. Very little air was required for the maximum flow at the start, but, as might be expected, an increase was necessary as the water lowered. The lift was then moved to another winze and similar results obtained. This winze, however, was down only a short distance and owing to an order for turning off the air at a certain time on night shift being forgotten, the water was lowered to a distance of 26 ft. from the top of the delivery pipe, being a distance of 7 in. only above the air jet; when the air was turned off, the water still being delivered was in the form of spray, and no doubt this had been the case for some hours, giving, however, a proportion of 97·8%. I do not claim for the air lift thus used any equality with ordinary pumps from the point of view of economy; but considering the facts that practically no supervision is required, no oiling of working parts, and that there is no wear and tear, I do claim that the air lift can be with advantage used for such purposes as the unwatering of flooded winzes, where the question of economy hardly enters, and the question of rapid installation is of first importance.

THE LATE MR. WAGER BRADFORD.

The President, referring to the recent death of Mr. Wager Bradford, said : In the life of the individual, in the life of a family, there are moments when the deepest sorrow comes upon each and all ; so it is in the history of a Society, and to-night it is with extreme regret that we miss the well known face of Mr. Wager Bradford. He was a keen and active member in our midst for many years, and our appreciation of him was shown in his having been our Vice-President for three years. There is little doubt that had not his managerial duties demanded so much of his time and were it not for some diffidence in taking on responsibilities for which he had not the leisure to do justice, he would have been President. We knew him well, and little therefore need be said by us. He was a man remarkable for rectitude and straightforwardness, kindly in demeanour, and a thorough worker. We mourn his loss. I would ask you to night to give your Council permission to forward your sincerest sympathy to his sorrowing wife and family.

The request was acceded to, all standing.

The President : Gentlemen, the next business on the agenda is my inaugural address, during the delivery of which, our Vice-President, Mr. W. R. Dowling, will take the chair.

Mr. Dowling having taken the chair, the President delivered his inaugural address, as follows :—

INAUGURAL ADDRESS
ON THE
CRYSTALLISATION OF IRON AND STEEL.

By A. McArthur Johnston, M.A., M.I.M.M., F.C.S. (President).

Some societies are happy in imposing on their newly elected president a duty in the shape of an inaugural address, whilst others demand a retiring oration. Of the two, I would personally prefer the latter, inasmuch as, by then an opportunity is afforded of talking about the work done during the tenure of office. This Society, the members of which have done me the honour to elect me to the highest position in their power, has an absurd practice of expecting an address, hoping thereby, it would seem, that an auspicious opening may be created.

Casting back over the performances of my predecessors, I find that some have contented themselves with saying little, others have criticised the work of the previous session—an invidious task—whilst there are not wanting instances where the presidential mind has soared into prophetic flights and depicted the future greatness of our industry, or of other industries depending largely on the skill of the chemist, the metallurgist or the miner. Now, I am quite willing to credit all here with having carefully read their *Journals* during the past year, and would therefore excuse myself from reviewing the work done and the papers read. Moreover, we are personally acquainted with the excellence of the year just closed in the number and quality of the papers, a tribute not only to the prestige of the Society but to the reputation of our Past-President, and so, with your permission, I will endeavour to interest you in the study of metals ; in this case, in the formation, shape and size of the crystals formed in iron and steel and the examination of these by means of the microscope.

Most of you are aware, that, in a few years, there is great probability that a new industry —I trust I am not prophesying too much, gentlemen—is likely to spring up in this country, namely, the mining and smelting of iron ores. Already our Government, realising its responsibilities, has taken a paternal interest in the erection of furnaces at the capital for re-melting waste or scrap iron, and I feel sure it is the hope of all of us that this will ultimately develop into the establishment on sound commercial lines of many smelting works not only for waste iron, but for the iron ores known to be present in payable quantities.

Our industry is invariably dependent on many others, and it is only by linking them up and making each a corollary of the others that truest economy can be obtained. We here know the constant striving we have to obtain the last grain of gold from the rock. We know, besides, that this is but one of our economical problems, whilst others lie before us, and I am pleased to say that this Society has in the past not been backward in pointing out the lines on which other economies could be and should be attained. Let us instance the enormous quantities of iron and steel yearly imported into the Colony and imagine the benefit not only to our industry, but to the country at large, were we able to mine and refine our iron ores and to produce therefrom the finished article. We are at the beginning of a new industry it would appear, and it is necessary that we take every precaution that not only are the best methods, the most up-to-date, adopted, but that the lessons learned from gold winning, that what at one time was reckoned waste or rubbish, may in future become valuable and well worth recovery, are remembered. Our aim should be to conserve the resources of the country, for therein true wealth lies, and it is necessary that the initial work thall be thorough and grounded on the best principles.

It were perhaps invidious that I should criticise the action of our Government in, at this juncture, choosing mechanical engineers to study the question of smelting and refining of iron and iron ores, but in view of many failures in the past with the introduction of new industries, I feel that a warning is necessary. This work comes in reality under the sphere of the metallurgist, and is quite beyond the scope of training undertaken by our confrères—the engineers.

To impress upon members the advantages of following on the best principles, I would to-night crave your attention when I deal with one of the latest phases of iron and steel examination. All metallurgists are aware tnat the first importance in iron and steel manufacture is given to the chemical composition. Herein lies the main element of successful commerce, but inasmuch as chemical analyses can definitely control this, there occurs but seldom a steel which gives trouble owing to the false or improper mixing of the elements. Next in importance to this, is the treatment to which the metal or alloy has been subjected during manufacture and herein lies the cause of many otherwise inexplicable failures.

More often than not, rule of thumb methods are adopted, temperatures are taken by visual effects, and a general amount of carelessness, due principally to ignorance, is bound to occur. As a means of determining the composition to some extent, and the treatment which any particular steel has undergone, I now propose to put a few facts before you on the crystallisation of iron and steel, especially when viewed under the microscope. The subject is a vast one and its growth of but recent date, so I would ask you to excuse me if I touch on a very limited section of it, and am a trifle elementary in many parts. It is my intention to deal mainly with the size of the crystals in iron and steel and the effect on these of heat treatment, mechanical treatment, and of the strains and stresses set up during work.

To many, a piece of iron is viewed as a homogeneous mass, made up, it may be, of many small multi-sided crystals, as seen when a fracture is made. Modern microscopic investigation has, however, revealed to us the exact nature of the combinations found in all kinds of iron and steel. Briefly sketched, the method of obtaining this information is as follows. A small piece of any particular sample is taken, due regard being given to its position in the main structure; one of the faces is filed flat and then ground on emery paper and finally polished by the finest rouge until no scratches are seen and a mirror like structure is obtained. This is then observed under the microscope, and not infrequently deductions may then be drawn, as to the nature of the material and the treatment which it has undergone. Thereafter the polished surface is dipped for a very short time—5 to 10 seconds—in an acid, usually nitric or picric, which has the effect of eating away certain portions and not affecting others; the untouched portions are bright in colour, whilst the others are dark or dulled, due to the depth of the furrows or hole. The microscope shows that pure metals consist of polyhedral grains, each of which is separated from the others by a very thin boundary line and which are themselves found to be made up of numbers of minute crystals, each a replica of the large grain. During the cooling from the plastic state, these crystals have separated out and the irregular shapes are due to the lapse of time and the compression in area during the solidification.

The size of these grains is dependent on many circumstances, among which may be noted (*a*) the temperature to which the metal has been heated prior to cooling, (*b*) the rate of cooling, (*c*) the treatment during cooling, (*d*) the presence of certain constituents or impurities which have, according to their nature, the power of increasing or decreasing the size of the grains, and (*e*) the nature of the metal itself. The reason why that variation is capable of taking place is that iron or steel above a certain temperature possesses the property expressed by the term "solid solution." This phrase may not be a happy one, since to the ordinary mind, it expresses a contradictory state, but it is expressive inasmuch as it attributes to the iron in a semi-solid state the properties of a solution. We know that when whiskey and water are mixed, we cannot either by the microscope nor in any other way detect the one from the other. The mixture is complete and a solution has been formed with thorough merging of each in the mixture. Again, when sodium and chlorine are brought into chemical combination, salt is obtained which exhibits none of the characteristics of each of these elements—the identity of each is here also merged. So when iron and carbon are brought together at a high temperature, there is obtained an alloy which, above 700° C. is in a state of solid solution. Below this temperature, however, we find that separation takes place in the solid form attained.

Now we find that all metals in cooling from this state of solid solution, which they have attained at high temperatures, segregate out into crystals or grains. These, viewed under the microscope, are seen to be many sided and of irregular shape. As one of the most simple metals, let us examine a piece of wrought iron, which we take in this instance because pure iron is exceedingly difficult to get. Wrought iron has usually been subjected to a certain amount of mechanical treatment, and it is interesting to note the size and shape of the grains and the effect thereon of the welding or rolling. Let us examine Fig. 1. This represents the structure of wrought iron, the face exposed being a cross-section, or that at right angles to the direction of rolling. It will be noted that the grains, which are distinguished from each other by the boundary lines and by the slight varying degree of light and shade, due to orientation, that is the deflection of the light from the surface planes at different angles, are more or less constant in shape and size, no marked degree of elongation in either direction being apparent. The dark patches are generally sulphides of manganese or iron, and cinder, which are present in all commercial wrought iron, and you will note that although irregular in shape, no pronounced elongation of these areas in any particular direction is noticeable.

Fig. 2 shows the same piece of metal with the longitudinal side exposed to the lens. Note here, first, that the grains are approximately of the same size as in the preceding, and that in this case also they exhibit no inclination to stretch in the direction of rolling. A difference, however, arises in this case in that the slag is very pronounced both in size and shape. All these particles extend in the direction of rolling and present elongated areas., which in themselves conduce not at all to the strength of the iron. The quantity of slag present in wrought iron helps materially to determine the quality, since, as I say, its importance is entirely a negative one. Annealing will alter the shape and condition of the iron or steel crystals, but fails to alter the cinder and slag areas elongated by rolling or forging.

These photo micrographs show distinctly that the opinion held by many engineers that wrought iron is fibrous, is based on a fallacy. In all ordinary material, the crystals are as regularly shaped in longitudinal section as in a transverse section; whilst even in cold worked iron in no case does the elongation of any particular grain amount to more than two or three times its least diameter, and this is quite insufficient to base an assumption that the structure is fibrous, more especially as even in this latter case, the distinctive character of the grain is still maintained. Whence arises this fallacy then? We know that many engineers prefer a wrought iron, which has a stringy or fibrous look. The sellers know this also and they take care that the engineer is not disappointed. With a little practice in breaking, the same piece of steel or iron can be made to show quite opposite structures. For example, wrought iron if nicked on one side and slowly bent, gives a decidedly fibrous looking fracture; whilst if nicked on both sides and sharply broken, a crystalline structure is obtained.

These two photo micrographs represented to you the structure of wrought iron, very often termed carbonless iron, a term rather a misnomer, since in all commercial varieties of wrought iron some carbon is found, though in many, it is true, only to a very limited extent. Let us examine now the appearance of iron containing carbon, that is, steel.

Engineers know well the valuable properties which the presence of carbon bestows on iron. Pure iron is soft and ductile at ordinary temperatures, whilst even a small percentage of carbon, say 0·2%, forming thus a mild steel, gives to the iron greater hardness and strength. Before proceeding further, I would like to draw your attention to the technical names used in steel micro-structure. These are ferrite, pearlite and cementite. Ferrite means pure iron, cementite means iron which is saturated with carbon, and pearlite is a mixture of these two forming a compound to which the term eutectic is given. This cementite contains 6·9% of carbon, and is a chemical compound of iron and carbon, sometimes termed iron carbide, Fe_3C. The steel, therefore, may be in either of three stages, either much ferrite, along with pearlite; pearlite alone, forming the eutectic state, that is when about 0·9% of carbon is present; and pearlite plus cementite. To illustrate the different properties belonging to these, I will quote from H. C. Boynton before the Iron and Steel Institute, 1906, where he gives the hardness of ferrite as 1, pearlite as from 3 to 4, and cementite as 272·8.

As the word eutectic is constantly in use in metallographic work, I would like to explain it in greater detail. Steel, when cooling from the state of solid solution, separates out into ferrite and cementite. Like other simple binary metallic alloys, it, during cooling from a state of molten solution, behaves in the same manner as a solution of salt in water. When an unsaturated solution of salt water is cooled to a certain temperature, it is found that water begins to freeze out, and further lowering of the temperature drives out more of the water until the saturation point is reached, when it is noted that continued cooling separates out a very fine mechanical mixture of salt and ice. This mixture of crystals is called the eutectic and represents a fixed definite ratio of ice to salt. In a low carbon steel the same phenomenon is witnessed. Reduction of temperature during the state of solid solution freezes out ferrite until the critical point is reached when the mother metal solidifies as a mechanical mixture of ferrite and cementite. This is pearlite, and when the whole mass contains no excess ferrite and no excess cementite, the steel is wholly pearlite and is called the eutectic. When iron and carbon only are present eutectic steel is formed with a percentage of 0·9% of carbon, but the presence of other ingredients tends to lower this figure. Pearlite forms crystals in the same way as pure metal does. When more than this percentage of carbon is present, we find brought into existence excess cementite. Note that cementite takes the place of ferrite and forms the boundary lines between the grains of pearlite.

Fig. No. 3 shows a mild steel containing 0·2% carbon. Here will be seen the white areas of ferrite and the darker patches of pearlite, the former occupying by far the greater part of the surface. Pearlite may be distinguished from slag, in this quality of iron, in that it is not rounded or lake like in structure. It shows no slate or dove coloured tints, but rather is seen to consist of small plates of bright and dark colours, this being more evident under higher magnifications. When viewed under oblique light, pearlite is composed of various tints, much resembling the rainbow effects of mother of pearl—hence its name.

Before proceeding further with the crystallisation of steel, I would draw your attention to the photograph, Fig. 4, where a unique break in a cam shaft is seen. The two ends are shown and you will note the longitudinal break right down the middle of the shaft. This shaft was sold as double-faggotted iron and the break occurred about halfway from either end. As you know this class of iron is made from scrap, and heat treated and hammered until, what is claimed to be an excellent shafting, is obtained. Microsections were obtained from each of the broken ends, and I reproduce them here. One, Figs. 5 and 6, shows the structure of wrought iron and the other, Figs. 7 and 8, that of mild carbon steel. In both you will see the crystals of ferrite, but in the former the dark areas are slag, whilst in the latter they are seen to be pearlite. The difference between these is further accentuated in the higher magnifications. *En passant*, I would point out that although this material has been hammered and rolled, no decided drawing out or elongation of the crystals is apparent. Now I have said that the influence of the slag in iron is entirely negative, that is, it weakens the material inasmuch as it has no strengthening effect and takes the place of iron. This cam shaft therefore has one disadvantage in that there is so much of this useless matter present. Had the temperature, during which the mechanical work was performed, been considerably higher, there is every probability that much of this would have been eliminated and the shafting thereby strengthened. I do not hold that the mixing of mild steel with good wrought iron, would have a deleterious effect on this class of material, as one piece examined in which both were found failed to show any junction lines. The blending or merging of one into the other was complete.

A more serious objection to this particular instance lies in the longitudinal break, as this would tend to the conclusion that the temperature of welding was too low, with the result that the different layers failed to unite and thus was formed a line of intergranular weakness. The question of the best material to use for cam shafts is one which our engineers ought to carry to a logical conclusion, as much money is yearly thrown away on these fields on account of not only bad material, but unsuitable material, being supplied. An unkind critic told me the other day that makers are not anxious to improve the quality, since doing so would mean that they would fail to sell as many as they do at present. I, however, put little faith in this kind of criticism as it shows an utter failure to understand the first elements of political economy, and I can only say that if any engineering firm is anxious to conduct tests on the best class of iron for building cam shafts, we here will only be too pleased to help them out to the best of our ability. The item is a serious one in many mills, and any improvement would mean a considerable saving.

A further increase in the carbon content is shown in Fig. 9, where we find a considerably less extent of the white ferrite and an increased pearlitic structure.

A 0·5% carbon steel is seen in Fig. 10 and the main difference here lies in the decreasing area of the ferrite and the increasing expanse of pearlite. More distinctly is noted the size and shape of the pearlite crystals, which are now enclosed by bands of ferrite.

I would next bring to your notice the photo micrograph of a steel, Fig. 11, containing 0·75% carbon, of which I have obtained a magnification of 1,000 diameters. This brings clearly into view the nature of the pearlite, which is here seen both in the laminated and granular form. The laminated consists of alternate thin plates of cementite and ferrite, and the granular of a very short, scrappy looking admixture of the same constituents. You will note that the grains or crystals of pearlite are bounded by lines of ferrite, which points to a network crystallisation of the excess ferrite with, immediately afterwards, the filling in of the interstices by the pearlite grains.

Shortly afterwards we reach the stage when nothing but pearlite is found to exist. This is the eutectic steel, which as I have already noted, occurs in pure iron-carbon alloys when the carbon content reaches 0·9%. Above this point, we find that instead of ferrite being in excess, the cementite is, and forms the boundary lines between the grains of pearlite. This forms a high carbon steel, hard and brittle, and in Fig 12 is seen the microstructure of such, viz., containing 1·25% carbon. The reason for this brittleness is that the walls of cementite, since their coefficient of contraction is greater than the pearlite grains, form a line of weakness.

Having thus briefly dealt with the main elementary composition of iron and steel, as shown by the microscope, I would like to draw your attention next to the size of the crystals and the effect on them of heat, work and chemical composition. It is generally recognised that the smaller the grain, the higher are the physical properties of the steel and iron. The following figures obtained by Sauveur tend to prove this:—

Size of grain in 0·0001 sq. m.m.	Tensile strength, kilograms per sq. m.m.	Elongation % of length.	Reduction, %.
148	69·6	15·0	20
118	70·3	19·0	22
62	77·7	22·5	25

In determining the size of the grain, the influence of heat is of great importance. Before dealing with this, I would, however, like to bring to your notice that in all commercial iron and steel there is what is known as the critical temperature. By this is implied that in cooling from high temperatures a point is reached when an increased evolution of heat takes place. This change is entirely a physical one. Prior to this the alloy is capable of change, chemical or physical, but after passing this point no other change takes place; the metal is set. In pure iron, three critical temperatures are noticed—860° C., 750° C. and 690° C., whilst in hard steel only one is found, namely, 690° C. These points are very much influenced by the chemical constituents, and generally speaking, the less pure the metal, the more tendency is there to one critical temperature being found.

The main influence of heat is seen to be that the higher the temperature, above the critical point, to which the metal is brought, the larger will be the grain, whilst the nearer this point the heat reaches, the smaller is the size attained, that is, the finer is the structure. This of course presupposes equality in the rate of cooling. Figs. 13—15 clearly indicate this phenomenon. These are diagrams by Campbell and show soft steel heated to: Fig. 13, about 900°; Fig. 14, 1,200°; and Fig. 15, 1,300°. It is interesting in this connection to note that when steels containing about 0·8% carbon are heated to between 1,100° C. and 1,200° C., a critical point is reached, above which the injurious effects of overheating are very marked. Such a steel as a jumper or drill steel would come under this class.

Again, provided that steels of similar composition be brought to the same high temperature, then the relative size of the grain depends entirely on the rate of cooling. This is evidenced by the following three photo micrographs, Figs. 16—18. Fig. 16 shows the result of heating to 800° C. and cooling in air. Fig. 17 has been heated to 1,000° C. and cooled in air. Fig. 18 to 1,000° C. and cooled in the furnace.

As following on these principles, we find that the structure of hardened or unhardened steels can be changed by heating them to the critical point, and provided that that point be not greatly exceeded, the coarsest structure can be converted into the finest possible the metal can assume. This is the process generally known as annealing, and by it brittle or overhard steel is converted to practically any degree of hardness required, the final hardness being dependent on the length of time annealing has taken, the degree of heat applied and the subsequent rate of cooling. A very coarse crystalline structure is also obtained by subjecting the steel to a lengthy treatment above 1,000° C. This coarseness implies brittleness, and should the temperature be considerably above this, and the time relatively long, there is very grave danger of the metal being burnt.

The size of the crystals can be reduced by cooling quickly and the burning effect reduced by extended annealing, except when the temperature has been so high that the melting point has been almost reached so that gases have been given off with subsequent oxidation of the metal. As an interesting sidelight on this you may ask why it is that a steel can cool from the melting point without burning, when a lesser degree of heating burns the metal. The most probable explanation is, that when the metal is cooling from the molten state, hydrogen gases are given off, which prevent air from getting in, but in the case of reheating to near the melting point, there is an intake of air between the crystals, and thus oxidation with its concomitant burning takes place.

The next most important influence on the crystallisation of steel is the treatment to which it is subjected during cooling. This may be of two kinds, the degree or rate of cooling, and the mechanical usage it receives. Concerning the former, I have shown that the quicker the rate of cooling the smaller will be the grain size. On this principle depends the reason why the outer layer of steels and cast irons is always smaller in grain size than is found further in. The cooling naturally has been quicker there, hence the largest grains are nearest the centre. This, within limits, explains why the skin of these is always reckoned as the strongest part by many engineers. More importance, however, may be given to the effect of mechanical work on steel. This may be divided into hot working and cold working. Hot working is known as that carried on when the temperature is above the critical point, whilst cold working implies work below this. Let us instance the case of a wire in a rope.

The billet before being mechanically drawn has a certain size of grain. The first passage between the rolls breaks down this grain, and the subsequent grain formed takes the size attained at the temperature at that time. The next rolling or drawing breaks down this grain size, and another grain is formed with the size of that temperature. This would continue during subsequent rolling, until the temperature of the final passage through the gauge gives the size of the grain found in the wire. In this case, the action of the rolling has been felt right through the wire, and the grain throughout has been prevented from forming until the final drawing is finished. It thus follows that the temperature at this time will determine the relative size of the grain. In the case of very hard drawn wires the crystals may be found drawn out into long, narrow forms, which may be called fibres, though, inasmuch as the greatest length of these may not exceed two or three times the least dimension, and the crystal remains intact, except when distortion takes place, the term is rather a misnomer.

I have here for your inspection three photographs showing the structure of wires Fig. 19 shows the section of a wire which has been heated up to a point slightly above the critical. The grains are regular and evenly distributed. In Fig. 20 you will see that the ferrite is found in elongated shapes, resembling fibres, and this shows that cold working has taken place; whilst in Fig. 21 you can note not only two cracks extending across the section, but also a considerable distortion of the ferrite grains. This last photo micrograph depicts a longitudinal section of a wire from a rope, which had been in use here. It is probable that the wires to begin with were hard drawn, and the subsequent bending treatment in passing over the sheaves had rendered them so brittle that not only had distortion of the grains taken place, but cracks at right angles to the length of the wire had developed. It is, of course, needless to point out that once these cracks have reached this stage no annealing will induce improvement in the metal.

A more vigorous action, such as hammering, will, of course, have the same effect on a larger piece of metal, and, provided that this mechanical work is sufficiently strong to make its influence felt to the centre, no crystallisation will begin whilst it continues. Should, however, the action of the blow reach but a certain distance into the material, then crystallisation will begin beyond that point, and this explains why in large diameter shafts the nearer the centre the larger is the grain—the action of the rolling has not been felt there. This is, however, counteracted to a certain extent, by the initial heating of the material not being so great at the centre of the metal as on the outside.

This hammering or mechanical work has, as we all know, a further use in that in many instances it closes up blow-holes or drives out enclosed cinder or slag. To effect this, it is advisable that the alloy be worked as hot as is possible without injuring the metal, since there is less power required and the work is better performed. In finishing off it may be found advisable to cool the metal to very near the critical point, whilst mechanical work is maintained. Thus is obtained a low finishing temperature and consequently a fine grain. We are familiar with the procedure adopted in welding, when the blacksmith thickens the weld, so that the subsequent hammering which he finds advisable may bring the shape to the required size and the grains to the smallest possible.

Mechanical work may also be applied, or we had better say misapplied, where the temperature is below the critical. This is injurious to the steel since the crystals, having by this time definitely set, are destroyed either by flattening, elongating or otherwise distorting them. We find this in boilers, tanks or large pipes which have to undergo great pressure. The cold work performed during rivetting causes deformation, which ultimately leads to the destruction of the iron through fatigue. To a great extent this may be remedied, and the original structure restored by annealing.

The size of the grain may also vary according to the other constituents of the metal, but into this question I shall not enter to-night. Much more research work has yet to be done in this direction, since the influence of but comparatively few of foreign elements, when compounded with iron or with steel, have been determined.

Now there is one point which appeals to all of us, and that is that the crystal is by its very nature brittle. That this is only relatively so, I would now proceed to show you. When we compare the crystals in metals with those found in rocks, we note that marble, which is a crystalline body, is capable of much deformation, through stress, without alteration of structure. This applies to a block of ice also, and we find that the crystals in metals are so elastic that they can withstand a very considerable amount of strain or pressure without deformation. When slow deformation does take place, there is no shattering of the crystal, but only a gradual appearance of cracks within the crystal, or rather, grain, itself. These minute cracks develop especially when there are sudden and severe reversals in the stresses, and we find that the crystalline elements relatively alter their position with each other. It is only after the strain becomes too great, and the number of cracks extend almost right across, that fracture is obtained. This is called slipping, and the cracks slip-bands. Repeated grinding, due mainly to the reversal in stresses, between the faces of the slip-bands, tends to lower the cohesion. Fig. 22 shows a crack which has developed in, say a grain, and has extended right across the section, sometimes cutting through a grain and at other times running along the junction lines.

Should, however, a metal have been but slightly strained, a rest cure is beneficial. An approximate return to its former state is thus secured, the length of time required varying directly with the temperature and the amount of strain. This metal was at that time in an unstable condition, and therefore more liable to correction or alteration than one which had been greatly strained or cracked. This unstable condition is commonly known among engineers as fatigue. A steel spring in a watch will go on for years without losing its elasticity, and so we conclude that its elastic limit has never been exceeded—fatigue has not been generated. The steel spring on a railway car can last many years without breaking or mayhap without losing its spring, if care be taken that too heavy a burden be not imposed on it. Miscroscopic examination shows no deformation. A series of slight jars may, however, cause the elastic limit to be approached, and then we find these slip-bands appearing, and the consequent weakening of the spring with, in time, ultimate failure.

It will be found that slight heating or annealing will often be beneficial to the iron or steel in the "fatigued" condition, provided, of course, deformation has not been too extensive. A very familiar example might be quoted in the case of a razor. The very thin surface of the edge is much more susceptible to fatigue than are larger bodies, but on the other hand a "rest," or heat treatment by dipping in hot water, may renew its efficiency very considerably.

Having thus far sketched briefly the formation, growth and decay of crystals, I would now draw your attention to the popular opinion regarding the continued growth of these crystals due to strains, stresses, vibrations or shocks. This theory presupposes that the vibration allows the crystals to rearrange themselves and to enlarge. Were this the case, then we might reasonably conclude that when any metal had been injured by undue stresses, we could cause a speedy recovery by allowing it to undergo vibration with its consequent rearrangement of crystals. Tests on this line of argument have however shown the uselessness of the treatment, since no effect whatever has been produced on the metal.

I have shown that the fractures due to fatigue favour the breakdown of steel, and as these fractures proceed either across the crystal or along the junction lines we find in these breaks that the crystals are most distinctly marked. The grain is therefore seen to most advantage. Again, it is found that breaks oftenest occur where large crystallisation is evident. In this respect it must be borne in mind that the larger the crystals the less plastic they are and the more liable for the occurrence of slip-bands. This explains why fractures occur mostly at the place where are found the coarsest sized crystals, and what has led to the commonly accepted theory, a theory which mistakes in that the coarse crystallisation is the cause of the break, and not the result of strain or vibration.

To illustrate and drive home this assertion, I here produce for your inspection some photo micrographs of a stem, the one series showing the size of the crystals in the stem before use and the other depicting in similarly selected sections the crystal sizes after the stem had been hammering away for two years in a heavy stamp mill. This particular stem lasted almost exactly the two years before breaking.

The chemical analysis of the steel was as follows :—

Carbon	0·58%
Nickel	3·70 ,,
Manganese	0·57 ,,
Silicon	0·75 ,,
Sulphur	0·04 ,,
Phosphorus	0·05 ,,

This stem was introduced as a test of nickel steel, which combines great hardness, tensile strength and elasticity with high ductility and is in much demand for marine shafting and armour plate. The microstructural composition of the steel, neglecting the constituents other than carbon and iron, would be as follows :—

$$\left.\begin{array}{ll}\text{Cementite } (Fe_3C) & = 8{\cdot}70\% \\ \text{Ferrite } (Fe) & = 55{\cdot}68\text{ ,,}\end{array}\right\}\text{Pearlite} = 64{\cdot}38\%$$

Excess ferrite = 35·52 ,,
Total ferrite 91·20 ,,

A complete section was cut in each sample and the face at right angles to the length of the stem, that is the cross section, was polished in each piece. The sections presented are :—

Outer ...	near the periphery.
Middle ...	$\frac{3}{4}''$ from the periphery.
Inner ...	$1\frac{1}{2}''$,, ,, ,,

The diameter of the stem was $3\frac{3}{4}''$.

In these photo micrographs, Fig. 23-28, I wish you to note first, the gradual enlargement of the grains proceeding from the outside to the inner section, due to slower cooling nearer the centre and the lesser influence of mechanical rolling; next, the greater quantity of ferrite present in the external sections, due probably to a slight decarburization during the heating; and lastly, the great similarity in the relative size and structure of the grain. This goes to prove that, although the stem had been in use over two years and had then become so "fatigued" that breakage had occurred, the size of the grains had not altered, that is to say, the enlargement of these, popularly known as crystallisation, had not taken place. The cause of the break would, in this case, be due to fatigue, that is the distortion of the grains set up by the strains, so that there resulted initially the slip-bands, and in due course such extension of these as to form a crack, with weakening of the metal.

These bear out the openly expressed opinions of metallurgical experts at present. It may be that they will some day be altered when riper experience comes, as there is always the probability that the size of the crystal may be altered during the lapse of a long period of time, but so far as our present knowledge extends everything points to a permanence in the size of the crystals in spite of severe strains and stresses.

Gentlemen, in thanking you for your kind attention and interest shown, I feel that I must acknowledge my indebtedness to the Consolidated Gold Fields for liberty to reproduce many of these photo micrographs, and to Messrs. J. E. Metcalf and Moon for help in preparing the plates and slides illustrating my remarks.

Mr. W. R. Dowling (*Vice-President*) : I wish to propose a very hearty vote of thanks to our new President for his most interesting and instructive address. Mr. Johnston is a very busy man, but he has been able to spare us the time to give us the benefit of his knowledge. I think members are to be congratulated upon his election as President for the year, judging by to-night's paper.

Sir Thomas Price (*Visitor*) : May I be permitted the privilege of seconding the vote of thanks, and at the same time of thanking the Society for the courtesy they have accorded me in allowing me to be present to-night to listen to a particularly instructive and no less interesting lecture. In doing so, may I also join with you, sir, in offering our congratulations to the Society upon Mr. Johnston having accepted the Presidency. If I may be allowed to say so, you have been fortunate, and your fellow societies have been fortunate also. I have very much pleasure in seconding the vote of thanks.

Mr. E. J. Laschinger (*Member of Council*) : I should like to be allowed to make a few remarks. I know the paper is not open for discussion, but I think our President has rather taken advantage of the engineers as to what they know and what they do not know. I only rise for the purpose of asking the President to put it before the Society as to whether the technical part of this paper may not be thrown open for discussion at some future meeting. I think it is too valuable an opportunity to be lost, and that we should have a paper of this kind discussed.

The President : I am afraid that what Mr. Laschinger has asked is rather beyond my powers. We have many precedents before us, and must adhere to them, though, if any engineer likes to bring up the question, I shall be only too pleased to criticise him. Before sitting down I should like to extend a welcome to our guests to-night, the representative of the Chamber of Mines, Mr. G. Albu, Sir Thos. Price, and the representatives of kindred societies. I trust you will all stay to hear Mr. Cullen's reply to the criticisms upon the papers submitted by himself and Dr. Weiskopf some months ago.

Mr. W. Cullen submitted his reply as follows :—

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

(Read at November Meeting, 1908.)

By Wm. Cullen (Past-President).

THE ESTIMATION OF CARBON MONOXIDE IN MINE GAS.

(Read at February Meeting, 1909.)

By E. H. Weiskopf, Ph.D., F.C.S. (Member).

REPLY TO DISCUSSION.

Mr. W. Cullen and Dr. E. Weiskopf : The discussion on these two papers has been somewhat meagre, but that is hardly to be wondered at in view of the fact that few members of the Society possess facilities for carrying on work of this nature. We have, however, to thank those members who took part in the discussion for their courteous criticism of our work. Since the first paper was read in November of last year, experiments have been constantly going on, and our reply must therefore partake of the nature of a fresh paper. Some of the new facts will give the answer to certain questions which were asked by members, but it is most gratifying to us to know that the papers have elicited an interest which is almost world-wide.

Coming now to the discussion—Mr. McArthur Johnston, following immediately after the reading of the first paper, indicated that some reformation was required in the constituents of the explosives used on these fields. With this criticism we are in perfect accord. He also asked for some light on the probable source of the large quantities of CO which we showed were present in so many samples, and this point we shall deal with when we refer to the results of the new work done since the papers were written. He further states that our data, with regard to actual working conditions, were meagre, but if he examines the various appendices, I think he must agree that all essential data have been given. If more analyses are desired, we can only say that they would be only a repetition of those already given. Mr. Kingston asks for information about the *amount* of gas that may be retained by broken rock on the face after blasting, but we regret that we have no data on this point.

We thank Dr. Moir for correcting two printing errors—but neither of them could, we hope, mislead anyone. He asks for the publication of further results of analyses in our possession, as also details of oxygen determinations, but all through we have considered this subsidiary to the main question, viz. the determination of the CO present. We quite agree, however, that light is wanted on the question of oxygen. Regarding his theories to account for the production of CO, we think that we shall be able to give a satisfactory explanation for its almost universal presence. Neither of us, so far as we recollect, has laid claim to be the pioneers in discovering the presence of CO in explosive gases, and we know that many analyses similar to those he quotes, have been published. More than 17 years ago one of us did many analyses of gases from bomb explosions, and one could nearly always predict the results from the elementary composition of the particular explosive under examination. As these results seem now to have a practical importance which was not then realised, they may form the subject of a communication to the Society later on.

Dr. Moir has a grievance against one of us which goes back to 1903—6 years ago—a long time in this country. One of us has looked up the remarks in question and he is willing to admit that they are somewhat strong, nay even pointed, but that was the prevailing tone of the Society at the time. Dr. Moir himself was, and still is, no mean exponent of that particular type of debate, of which the remarks in question are an example. Beyond admitting that these remarks were strong we are unrepentent on every other count, but we think, as we said in the paper, that Dr. Moir did splendid pioneer work in this particular direction.

Mr. Whitehouse quotes certain analyses with which we are cognisant, and as he is a practical man we are bound to accept the conclusions at which he arrives with regard to clay tamping. We have not had an opportunity of pursuing this particular line of investigation further.

We regret that we attributed certain opinions to Dr. Haldane regarding the presence of CO in explosive gases. We had not the report of his commission at hand and merely quoted from memory, which accounts for the quite unintentional error. Our researches have shown conclusively that even with gelignite, if the ingredients are re-arranged scientifically, it is possible to produce gases which contain practically no CO. We shall refer to this later on. He is not quite correct in thinking that we attribute the high mortality of the Rand Mines mainly to the presence of CO and other noxious gases. We hold strongly the same views as he does, viz., silicosis is the main factor, and experience is confirming this daily, but we think also that continuous work in a vitiated atmosphere must through time impair a man's vitality and undermine his health. He asks one pertinent question on page 275, with regard to the conditions under which the samples of *ordinary mine air* were taken, and we have to inform him that in every instance the conditions were what we should call "working conditions." In nearly every case a minimum of 12 hours, but more frequently 20 hours elapsed between the previous blast and the taking of the sample.

Before placing the new results before you, we should like to say that our researches have extended over a period of nearly 3½ years and during most of this time experiments of one kind and another were in almost continuous progress. We do not propose to complicate issues by giving results of footage or explosive consumed per ton of rock broken, as these after all are side issues, but we may say that our experiments were not conducted with fancy compositions, but with serviceable explosives which will be put upon the market at once. Small improvements have been made in the apparatus for sampling and analysing the gases, but if it is thought desirable they can be made the subject of a separate communication to the Society.

On page 147, Table 3 in the first paper, the results of the analyses of the gases produced by gelignite are given, and it will be remembered that the ratio of CO to CO_2 came out as 1 to 4·9 – a result which was described as astounding. Another brand of gelignite gave 1 to 11 2, which is considerably better than the best blasting gelatine quoted on Table I. Both of these gelignites gave practically the same results when tested for strength by the Trauzl block and by footage in practical mining, but from a health point of view the one is infinitely superior to the other. Indeed this was proved very conclusively on a certain badly ventilated mine where both had been used over a period of years. With the former the contractors could reckon on a certain number of boys being gassed every week, while with the other there were no such occurrences. This led us to investigate the subject further and without any sacrifice of strength, we have been able to produce a gelignite which gives a ratio of 1 to 18·1, or practically no carbon monoxide at all from the explosive itself, as we shall see presently. This was admittedly a step forward.

You will, no doubt, recollect that all the blasting gelatines which we investigated produced large amounts of carbon monoxide—but one of them which we designated 13B in Table 1 of the first paper—appeared to give less than any of the others. We followed up this particular line of investigation and have been able to produce a blasting gelatine which gives none at all, but before giving the results obtained we desire to draw your attention to the "causation" of carbon monoxide. Even when the second paper was read in February we were uncertain about the point, but a careful analysis of the results hitherto obtained indicated that the cartridge wrapper might be the cause, and at the risk of being tedious we shall quote what we said on that occasion.

"This reason has tempted many to assume that the above objectionable gases originate from the explosive. Many theories, based on this contention, have been brought forward. It has been suggested that the nitro-cotton in the blasting gelatine does not take part in the chemical reaction involved. Others, in expecting nitric oxide as part product of an incomplete detonation, have argued that their inability to find large amounts of this gas is proof conclusive that carbon monoxide is not present either."

"Counter arguments have been advanced, that nitrogen peroxide, condensing as nitric and nitrous acid will, to a very great extent, escape being caught in the gas sample. Others again, have been puzzled by the deficiency in oxygen, which did not agree with theoretical expectations. A further suggestion has, I find, been made a considerable time ago by Dr. Moir, that the cartridge paper introduced with the explosive is responsible for the formation of the products of incomplete combustion. Although we have experiments on record showing that cartridge paper, even if intimately mixed with blasting gelatine does not take part in the combustion when the mixture is exploded in a lead block, the conditions in a drill hole may be different.

"Here we may have the key to the whole problem before us. The assumption that the cartridge paper takes part in the combustion would not only explain the deficiency of oxygen, the practically constant ratio of CO, and the absence of nitrogen oxide in anything like proportional quantities, but it would also furnish a reasonable explanation of the presence of hydrogen proved in the samples."

We have always contended that, provided reasonable care was exercised in charging and in blasting, the theoretical results were obtained, *i.e.*, blasting gelatine scientifically constructed—if we might so designate it—behaved in the bore-hole exactly as in the steel bomb, producing only permanent gases, carbonic acid and nitrogen. To prove the point conclusively the obvious experiment was therefore to try the effect of blasting gelatine with no cartridge wrappers at all. In the first of these experiments the greatest difficulty was experienced in charging the holes; indeed we may as well confess that they were not properly charged at all—as the cartridges buckled, leaving spaces large and small between the individual cartridges. Nevertheless, a round was blasted under these—the very worst possible conditions—and although it was absolutely hung up, the gases showed a ratio of 1 CO to 18·2 CO_2, the best hitherto obtained. This demonstrated the influence of cartridge paper, but we next went one step further, and wrapped the cartridges in tin-foil in order to enable us to charge the holes more satisfactorily. Our object in using tin-foil was, of course, to obviate the use of carbonaceous matter. The result showed a ratio of 1 CO to 25·7 CO_2. But, after all these experiments, though they, to our minds, show conclusively the influence of the wrapper on the production of CO, hardly indicate a practical way out of the difficulty. The explosive already referred to, viz., 13B, indicated the way, and modifying it still further on the lines of producing still more potential oxygen, we have been able to obtain a ratio of 1 CO to 16·7 CO_2.

Stripping off its paper and substituting tin-foil, we obtained a ratio of 1 CO to 26·2 CO_2. This is the explosive which has been commented on in the press during the past 14 days. After all, however, the practical test is perhaps the best, and we are informed by the miners who have been using it for months, that they can go straight to the face after a blast without feeling any inconvenience. This is not a practice which we recommend, but it certainly does confirm the analytical results. With regard to the strength, we are also assured by miners working on contract on four mines, that it is certainly no lower than that of ordinary blasting gelatine.

The question will now naturally be asked, why should it not be possible to so adjust the excess of oxygen that there is no carbon monoxide produced at all. The answer to this is very simple. Although we have succeeded in providing the new explosive with sufficient potential oxygen to completely oxidise all the combustion products from the wrapper, and render them quite innocuous, no amount of oxygen can possibly affect the products of combustion of the fuse, produced prior to detonation.

On p. 148 of our first paper we gave an analysis of the gases produced in a drive, by the burning of the ordinary amount of fuse used for blasting a round, and it will be remembered that the CO reached the comparatively high figure of ·05%. We have more experiments on this particular point in progress at the moment. It therefore stands to reason that as long as safety fuse is employed in ordinary blasting operations, just so long will the mine air be contaminated by carbon monoxide, and furthermore we think that practically all the carbon monoxide which we have found in our recent experiments, comes from the fuse and not from the explosive. To our minds there is one, and only one way of overcoming this difficulty, to adopt a system of electric blasting, using delay ignitions. We feel so strongly that this is the only remedy that we propose to submit a paper on the subject at an early date.

We think it right to sound a note of warning about the gases produced by our modified blasting gelatine. Some people have assumed that no gases are produced at all, which is manifestly absurd. Large amounts of the permanent gases, nitrogen and carbonic acid, are produced, and it may be, very small amounts of carbon monoxide. Nitrogen is innocuous, though one could not live in such an atmosphere, and carbonic acid though not a noxious gas does not support respiration, so manifestly it would be foolish to return to a face after a blast until the gaseous products of the blast have been at least partially dissipated. The ordinary precaution of carrying an open light and noting its behaviour will be an infallible warning of the presence of dangerous quantities of carbonic acid, but what a blessing it will be when that is the only gas which the miner need fear. Carbon monoxide, which has claimed such a heavy toll of human lives on these fields alone, and it must be the same elsewhere, seldom indicates its presence until it has its victims in its deadly grip.

We mentioned in the first paper that in no instance had nitric peroxide been observed, but we expressed the opinion that it must have been formed in small quantities. We now think that this is not the case, and that nitric peroxide was not formed at all, and is not formed so frequently as we had at first imagined. Indeed, all our experiments go to show that the margin of safety of good blasting gelatine, in so far as the production of noxious gases is concerned, is infinitely higher than is commonly supposed to be the case. What we mean is that there is a general idea that nitric peroxide is produced in every blast. That it is frequently produced there is no denying, but the amount of carbon monoxide which we have found has been so constant that we have come to the conclusion that nitric peroxide is only produced under most abnormal conditions—such as when the explosive burns instead of exploding. Now, in every series of shots arranged to go off in the neighbourhood of one another, the burdens must vary very considerably, and this we have always looked upon as tending to produce nitric peroxide, through incomplete or at least partial detonation. We have found no evidence to substantiate this at all. True, all our holes were charged and tamped with care, but with no greater care than should be ordinarily employed, and we were fortunate enough to have no "soft shots," "misfires" or untoward incidents of any kind, but the general conditions throughout were quite normal. In the one instance where the explosive was stripped of its wrapper and the holes were so badly charged that the round was hung up, an instance where one would have expected to get nitric peroxide and all sorts of complications, the result was excellent. We have already referred to the fact that when blasting gelatine burns instead of explodes, nitric peroxide is produced, indeed, it is invariably produced, and the deadly effects of this gas are well known to members. Athough we have not yet had an opportunity of thoroughly investigating the behaviour of the modified blasting gelatine under these conditions, *i.e.*, while burning, everything points to a very much diminished production of this terrible gas. The propagation of the detonation also seems to be more certain, and generally speaking, the all round margin of safety is apparently increased.

Just at the moment the thoughts of most mining men are being directed to the problem of ventilation, and the deeper they go into the bowels of the earth the more difficult does the problem become. We are free to confess that not until these experiments were well under way did we realise how much the problem was complicated, by the almost incredible amounts of carbon monoxide which we found to be present. We prescribe in our English Factory Acts a certain number of cubic feet of air space per person employed in a work room; the air not to contain more than a certain amount of carbonic acid. A margin is given when gas or oil is used as the illuminant. Even our Kafirs must have a certain amount of air space in their compounds, the underlying idea of course being the same. True, our mining regulations also prescribe a certain number of cubic feet of free air per second descending the downcast for each worker underground, but how frequently is this fresh air short circuited, and how seldom does it reach the lowest workings as fresh air. After all, the principles of ventilation are very simple, and we apply them in our own dwellings every day—we simply introduce plenty of fresh air to drive out the bad air. Now-a-days everybody is for efficiency, but we have some times wondered how terribly difficult it must have been to keep efficient when working in some of the atmospheres which we have examined. We hope that the underground conditions will be greatly improved by our new explosives. The simplest way, after all, to get rid of noxious gases is not to produce them, and this we hope we have succeeded in doing to a great extent. What savings it will effect we cannot pretend to be able to estimate, but we think these savings will be enormous—directly in the expenditure on plant, and indirectly in all round efficiency. But, after all, as we remarked in the first paper, the state of the mine air has never been realised, so it is not so wonderful that more has not been done to improve it. We trust that the result of our investigations will be the dawning of a new era, and we think that, purely from a commercial point of view, it will pay every mine, or group of mines, to do as we have done, to install our apparatus and to engage a competent person to conduct the gas analysis.

Our work has a purely scientific importance, which is already realised by those engaged in cognate work, but its practical importance can only be realised by those engaged in mining. After all, the conclusion of the whole matter may be summarised in one sentence. Carbon monoxide is produced in incredible quantities whenever a blast takes place; the modified explosive obviates its production altogether, or only produces it in negligible quantity. The less carbon monoxide we produce the more human lives do we save.

The President: I feel sure that you have been well repaid in waiting, by listening to the extra information which Mr. Cullen has given us. We should consider ourselves fortunate in that we have been able to obtain these papers, because their value is shown by their having been copied into all the metallurgical and mining papers throughout the world. I am sure Mr. Cullen's efforts in this direction will lead to good results. In listening to the remarks he has made we are reminded, by reason of his association with this work, very much of our Vice-President, Dr. Moir, who has recently undergone a very serious operation. I am pleased to be able to tell you that the operation has been highly successful, and that he is in a fair way towards recovery. I am sure he will be pleased to hear that he has been remembered amongst us to-night, and I will ask our Secretary to let him know that we are delighted to think that he will soon be amongst us again.

WHITE LABOUR IN MINING.

(*Read at January Meeting, 1909.*)

By Tom Johnson (Member).

REPLY TO DISCUSSION.

Mr. Tom Johnson (*Member*): Mr. Ralph Stokes thinks that I ought to have brought forward some arguments to show how white unskilled labour could be successfully used on shovelling.

It is a pity he copied my bad example in leaving things in obscurity, for he should have told us why this thing could not be done. I have seen white shovellers make an average of 12s. per day, *at a cost as low as natives had ever previously done the work at*, many of the men being strangers to the work; the mines were not in the best condition for white labour either.

As to a strike among white labourers, if Mr. Stokes will read my paper again, he will not find anywhere that I proposed to clear out all the natives. If he knows anything of working men, he will know that the best way to keep off the trouble of strikes, etc., is to have men of different nationalities, and therefore different ways of thinking. Too many of one kind are just as liable to go the wrong way as the right one, for they are very foolish when they get the idea into their heads that they are not being done justice.

Seeing that I am of the working class, and have been working with, and handling workmen in mining for 30 years or so; also seeing I have been in intimate contact with the workers on the Rand for 12 years, I see no reason to alter the opinion expressed in my paper that many of the so-called "skilled miners" look down on the unskilled man; the thing has come under my notice too often to mistake it.

In case anyone should think that the average of 12s. per day mentioned is for certain particularly good places, I may mention that it was for the whole mine, where, for about three months, there were no natives to shovel rock for the mill.

I have seen 80 men in a mine earning 15s. per day, at (as some of the members of this Society know) as cheap a rate as natives had ever done in the same mine. Some gangs have made over 20s. per man per day.

I do not wish it to be understood that I can instruct the managers on these fields how to handle white unskilled labour. Mr. Stokes says general opinions do not help to show how unskilled whites can be formed into a steady working force, but I can only tell you how I have seen them handled. In the mines I speak of, the men were formed into small gangs of two to four men in each, to do the shovelling, with natives doing the tramming for them; the whites being on contract at per car, the natives being paid by the company, there being no advantages possible in one case, as they did the whole of the shovelling of the rock for the mill. The orders given were that they must be shown the best way of doing their work, and that they must be treated as white men should be treated. These men were not put into a separate room at the boarding house, and fed at a cheaper rate than the skilled men, but were allowed to join, feed and pay like the skilled men. It seems strange that most of the unskilled men should prefer to pay the full board rates rather than the lower rates set for them, which was done with the intention of helping them, but so it proved. It might be called pride, but whatever it was it proved to be an important factor in handling them.

Now, regarding the machine work, Mr. Stokes thinks I took too low an average, but as it happens, I took the whole of the mines of one of our most prominent groups, and I can assure him that ·5 fathom per machine shift is as much as that group is getting on an average; I do not think that Mr. Stokes knows how little is really broken on some of our mines. Of course, I do not question his figure of ·7 fathom for the mines that he says he has the figures of, for I happen to know that on three of them they have men breaking a whole fathom per machine shift.

If the system I advocate were carried out in the mines he has in mind, he would see about 1·25 fathom per machine shift broken instead of ·7 fathom, the same argument applying to his averages as to mine.

Mr. Stokes compares the mines he has the figures for with my figures. That is not fair, as I did not write of different mines, but of different systems in the same mines, which is a vastly different thing. He must compare what he states is done with what could be done by my system, that is, taking the best rock breakers amongst the men and putting them over the others as instructors and running the machines the whole shift, and then see what the difference is. He will find that the total cost will be less, and the wages greater with the change. He has gone to the trouble of working out figures that have no bearing on the case; does he wish to throw dust into people's eyes?

Cannot Mr. Stokes criticise without wilfully misconstruing figures? There is nothing in my paper to suggest that I wish the good men to come down to the same wage as the poor men. I spoke of averages, why should he particularise? Why should he think that the blasting gang is not essential to the scheme, when the figures are based on the use of a blasting gang? It is plain enough in the paper, that I am after greater efficiency per machine by using more white labour, and having the machines running as long as possible during the shift, which the blasting gang would conduce to, so it must be essential.

As to the drillers not being in favour of this scheme, I do not see why they should not be. It is my experience that most of the machine men are not fit to be in charge of the breaking portion of the work, and that they would make a better wage drilling at per foot than breaking at per fathom, to the benefit of the company employing them.

As to whether 45s. per fathom is good enough as an average for these mines, I did not say, but if I must express any opinion I should say the companies are losing money by paying so low a price.

The "professors" of mining would not be working at the average rate, but should be on contract per fathom, so it can be seen that the scheme brings out self interest, as does contract work now, with a chance of supplying the men of different abilities with the work they are fit for. Why should working places, boys, machines, air, etc., be provided for men who do not understand how to make the best use of them, if by altering the system these men can have the chance of as good or better living, more of them employed, and at a saving to the companies. It is well known that the men who do the most work with their machines do so at a much less cost per machine in air, spares, etc., but it seems we must not take advantage of this. Why should we attempt here in mining what is not done in any other part of the world? *i.e.*, to have only one class of men—the best—cannot be done properly. Men here are only human beings as in other places, and so are of different abilities. Many men running machines, and supposed to be "miners," will never be miners as long as they live. They may be good "drillers," but not miners, so have such a system of working that the men can be fitted to the work they are most fit for. It is foolishness trying to fit square pegs in round holes; they take too much driving.

I wonder does Mr. Stokes think that the same machine men of the Rand are going to stay on for ever; if so, I agree with him that there is a probability of trouble in trying to get this scheme going for the reason he states, viz., the "instructors" would be afraid of losing their jobs, but men will come and men will go and the mines will be here and increase in number, if not for ever, at least for more years than Mr. Stokes or I expect to need them. When we get the men good enough not to need this tuition I am advocating, we can expect a blue snow on the Rand.

I am sorry that some of our managers did not join in this discussion, as we should get to know how we stand if they would come forward with their opinions.

I am glad Mr. Price can bear out what I have replied to Mr. Stokes, as regards the earning of 15s. per day by white shovellers. He says that if Kafirs can be got to do the work at 3d. per truck, why should they pay whites 1s.? Well, I guess very few have done anything of the sort, at least, I have not heard of it.

A great many people speak of how many Kafirs must be replaced by a white to enable a white to make a living wage, and the greater part of them do not stop to think what a Kafir costs, or how much work he actually does.

Mr. Price objects to putting white unskilled labour in competition with the natives. Well, it is a matter of opinion of course, but I am under the impression that the native is getting into competition with the skilled man. He says that white unskilled workmen have lowered wages, because the mines were flooded with Dutchmen and others during the strike. If I were to argue on this, I should want to know the cause of the strike and what led to it. Machinemen did not know when they were well off; the idleness of the old hands was the cause of all the trouble. When they had one machine to run, they would not run it; there were two boys for that in their opinion. Could you expect bosses to put up with that kind of thing always?

What I am advocating now, is what ought to have been from the first, *i.e.*, a white man on each machine, and the machine to be run at something like full capacity, and not at about half. If the compressors on a mine run eight hours, the machines should be running about the same time. Split up the work and put different men to the different work. As to men running three and four machines for 16s. 8d. per shift, in most cases the 16s. 8d. is as much as the man is worth, for the machineman who cannot command more than that is a loss to his employers. From what I know of contract prices, better money should be made. This three machine racket is a step in the wrong direction, and I think it is a case of the blind leading the blind. It is a fact that the machinemen were not giving value for the money, so the three machines per man was started in the hope that things would get better, but the energy ought to have been put into getting the work out of the lesser number of machines. There are so very few men who can get decent work out of even *one* machine much less out of *two* or *three.*

I do not speak of *one* man running 10 or 12 machines, there are *10 or 12* machine runners in practice. It would turn out to be a case of a shift boss—for the contractor (if one) would be practically a shift boss—for every 10 or 12 machines. That looks better than one man running 10 or 12 machines, does it not? and that is the case summed up. The man would not need to be in three or four places; two faces could easily carry the machines, and as to the holes going in some other direction than the one wanted, *that* happens now, and it certainly should be better with some good man to boss up every ten white machine runners.

THE PILGRIMS REST GOLD FIELDS AND MINING METHODS.

(Read at March Meeting, 1909.)

By J. Moyle-Phillips (Member).

DISCUSSION.

Mr. J. C. Mance (*Member, Singapore*) (*contributed*): It has given me great pleasure to read through this paper, but may I make a suggestion; that it would add considerably to the interest of the paper if the following further particulars were added:—

Total cost per ton mining by men.
Total cost per ton mining by "umfaans."
Cost of milling per ton.
Cost of cyaniding per ton.
Cost of tramming per ton per mile:
(1) For the self-acting inclines.
(2) For the electric haulage.

I do not know whether the author is in the employ of the Transvaal Gold Mining Estates, Ltd., or not, but if so, would it not be possible to ask him to give an extra paper, giving the costs of the mining in detail? I am sure the Society would greatly appreciate his work.

Referring to the paper, I should like to make a few remarks.

It would be interesting to know, if, at the contact of the reef with a dyke, the gold contents were higher, lower or regular.

Mention is made of cracks in the formation. Perhaps the following remarks about cracks might be of interest. During my stay on the Murchison Free State Property, in the Low Country, Transvaal, on the first level a very nearly horizontal crack was encountered which went right across the reef and into the country rock (talcose schist) without any dislocation of the reef. Where this crack was struck in the south end of the property it was a space of 6 in., and dipping to the West about 5°. To the West of the property, about 500 ft., the crack was about 3 in. In the crack there was a certain amount of clay, which I assayed, but could not get any gold. The values of gold were not altered by this crack, being the same above and below.

I notice Mr. W. A. Caldecott comments on the fineness of the gold in the oxidised ore. I should like to follow up his remark by stating that once, when visiting the Frankfurt Mine, to the north of Pilgrims Rest, I had occasion to take some 15—20 samples in a drive, and the highest value by panning I was able to record was 4 dwts, these samples, when assayed, gave anything from 5—18 dwts. to the ton. I was astonished at not seeing gold freely in the pan. The figures I give are from memory, but the occurrence struck me as remarkable and one not easily forgotten.

The following from my pocket book, which I happen to have by me, with reference to the gold contents in the oxidised ore, clearly shows where the gold is.

Mesh used, 900 to the square inch.

Per cent. of gold in heavy sand ...	7·18
Per cent. of gold in light sand ...	34·44
Per cent. of gold in slimes ...	58·38

I trust these remarks may be of some interest to the Society.

Mr. Chas. B. Saner (*Vice-President*) (*contributed*): This paper was to me of great interest, as it brought vividly to my mind two happy though strenuous years spent as surveyor and assayer in this beautiful part of the Transvaal, prior to the war. About the time I arrived in Pilgrims there was a "boom" on in claims, and great activity in prospecting was evident on all sides.

The district, though rich in gold, and one of the oldest in the country where mining operations have been carried on, has been consistently neglected by various governments in the matter of railway facilities. This has been the chief reason why it has not gone ahead, and even today the railway is being built to open up the wrong end of the district first, by going via Lydenburg, which is separated from Pilgrims, the gold centre, by mountainous country.

The correct idea, to my mind, is to encourage and foster an industry in the making and from its prosperity other industries, agricultural and pastoral will spring up and flourish. It used to be heartrending to be compelled to wait weeks and months for stores and materials brought up by the slow ox wagon along ill-kept tracks and through fearful and often impassable river drifts. I had to wait three months before my assay and survey outfit were delivered, though they left Johannesburg with me.

The entire district being very mountainous, the deep gorges provide ample fall for all requirements of water power and automatic ore delivery. There is a splendid supply of unlimited water all over the district, and the numerous lovely waterfalls and cascades from the Mac-Mac (about 200 ft. fall) to numbers of other smaller but no less powerful falls make an enterprising engineer's heart yearn for opportunities to harness these glorious natural forces that have for ages run to waste. Owing to the almost horizontal lie of the strata and the mountainous nature of the country, innumerable natural sections of the formation are exposed in the kloofs and gullies, and I regret that the author has not given a section with his paper. The reefs in the dolomites were the only ones payable, though there were cases of reefs or "chutes" being profitably worked in other formations on a small scale. The thin vertical quartz leaders showing much visible gold in the lower shales afforded profitable work for "diggers."

The "Sherwell Reef" was an auriferous haematite containing reef gold very rich in places, but of no extent, in a decomposed diorite.

In the coarse "berg sandstones," which run along the eastern edge of the berg, there are several "banket" beds up to 20 ft. in thickness, but where prospected and sampled by the writer, these gave poor results in gold. Below the sandstones, in the low country, is the granitic formation, this carries numerous vertical quartz reefs which have so far proved valueless except in one case, which I understand is now being worked near the Graskop farm. Correct sampling of these various reefs was extremely difficult as they were generally very patchy, the friable decomposed oxides being rich and often very thin, while barren or almost barren, hard and hungry quartz would form the greater portion of the reef.

I should here like to warn anyone who may be going to these outside districts to inspect claims or reefs to beware of having his samples "salted;" all sorts of tricks being employed to hoodwink the "new chum." Assaying these ores was also a much more complicated job than on the Rand, as several other metals and minerals were often associated together.

As I was connected with a prospecting concern all my fusions were done in a muffle furnace, the fuel used being locally made charcoal, as coke was very expensive. Charcoal, as made and purchased from the natives, was very bad, in fact was generally only half burnt wood, so as we were using a goodly quantity, both for assaying and drill sharpening, it was found better to make our own, and the following method was employed: A cord wood 3 in. to 4 in. in diameter is fixed vertically some 6 in. into the ground, the poles of wood are placed also almost vertically around this upright as close as possible, beginning with the shortest and gradually lengthening until the required height of the pit is reached, and thus forming a circular pyramid. The pole is then withdrawn leaving a hole 5 in. to 6 in. on the top for ignition and this is filled with moss and twigs. The pile is covered with moss and leaves and then earth and clay or sods is put on some 6 in. thick and patted down. Two vent holes, 3 in. in diameter, opposite each other, are made at the bottom of the pile. When the fire has been fairly started (it takes from one to two hours) the top is completely closed over; the two vent holes allowing sufficient ventilation for charring. Constant attention is necessary day and night until all charring is over, experience guiding one in the amount of ventilation required. This can be judged from the vapour escaping from the vents; no smoke must appear and no flame, after about a week's smouldering vents are closed excluding all air, and the pile allowed to cool for another week and then the charcoal taken out; to be good charcoal should be very light, have a metallic "ring" and a silky and almost metallic lustre on fracture, whole cords should be intact when removed.

Surveying of the mines on account of the adits was comparatively simple, but because of the rugged nature of the country a decent base line for any triangulation was very difficult to measure, and the farm beacons were usually only rough heaps of loose stones. In my case the true meridian was obtained by observing a fixed star at various altitudes, before and after culmination, at nights and this was frequently interfered with by the clouds hanging over the Berg for days at a time.

In summer, the surveyor is much handicapped by the long grass, thorns and thick bush. In measuring the quantity of water in the rivers or streams careful note had to be made that it was the normal quantity flowing, as at times a stream would develop into a roaring torrent because of a local downpour elsewhere while at others it might be a trickling rivulet and so quite deceive one as to the actual volume available.

As the author remarks, the native labour supply was and is very erratic, and the term of service much too short to get any degree of efficiency from the boys.

I am glad to see that now there is no loss of time through drunkenness. During my time, this was a great evil in the outside districts not so much from the Kafir beer as from bad rum, etc., sold by the local storekeepers, until the following compromise was come to that the management would wink at it and make no complaint if the storekeeper would sell only per "tot" at the counter and not per bottle; this worked well, the natives got their tots and we got our labour. The short term of service, three to four months, was a very disorganising factor; the native kraals were just below the Berg, a day's journey at the most, and yet they could not be prevailed upon to work for a longer period; the supply was best during the famine period, that is between sowing and reaping: otherwise they had their wives to till for them, their daughters to brew for them and their sons to hunt for them; while they, the overlords, basked in the sun and loafed, this was the pre-war period. I hope all is now being changed, the native has our protection and he should be made to work and be a useful citizen.

STOPE MEASUREMENTS.

(Read at May Meeting, 1909.)

By Otto S. Tonnesen (Member).

DISCUSSION.

Dr. J. Moir (*VicePresident*) (*contributed*): I have no practical knowledge of surveying, but I have to offer a little mathematical criticism on this paper.

I am sorry to find the author helping to perpetuate a certain popular error, viz., that the curve in which a stretched chain or tape hangs under gravity is a parabola. This is not the case, for the curve is a catenary, and it only resembles a parabola superficially. The equation to the catenary (origin at bottom of curve) is $y = c(\cosh\frac{x}{c} - 1)$, and the corresponding parabola is $y = \frac{x^2}{2c}$, so that they only correspond near the bottom, as will be seen on expanding $\cosh\frac{x}{c}$ in powers of x. The equation to the curve of a sloping tape (origin at lowest point) is similarly $y = c \sec\alpha\,(\cosh\frac{x}{c} - 1) + c\tan\alpha\,\sinh\frac{x}{c}$ where $c = \frac{T}{w}\cos\alpha$; α being the slope, T the bottom tension, and w the weight of the tape per unit length: if we make $\frac{T}{w} = a$ (length of tape equal to bottom tension), the curve becomes $y = a(\cosh\frac{x}{a\cos\alpha} - 1) + a\sin\alpha\,\sinh\frac{x}{a\cos\alpha}$

It is true that, when the sag is small, the expression for the latter is very nearly the same as what the author assumes, viz., $S = \frac{l^2\cos\alpha}{4a}$, where l is length of tape, *i.e.*, the sag varies as square of length of tape and also as its specific weight and inversely as its tension.

Again the calculation of area on pp. 377-378 appears to be wrong, but the author is so obscure that I am uncertain of his meaning. The area of a parabolic segment is two thirds of the *triangle* enclosed by the tangents and the chord and therefore *one-third* of the corresponding parallelogram. I hope the author will clear this up in his reply; but in fact, as the curve is not a parabola, the expression for the area is different. The area between curve and chord is, of course, $\frac{1}{2}x_1y_1 - \int_0^{x_1} y\,dx$. I have worked this out from the above curve equation, and find that (x_1 being ordinate of top of tape):—

$$A = \frac{ax_1}{2} + xc\sin\alpha + \left(\frac{ax_1}{2} - ac\sin\alpha\right)\cosh\frac{x_1}{c} + \left(\frac{ax_1}{2}\sin\alpha - ac\right)\sinh\frac{x_1}{c}$$

On expanding the hyperbolic functions as far as x_1^4, this reduces to: $A = \frac{ac + a\sin\alpha\, x_1}{12c^3}.\,x^3_1$; or since $x_1 = l\cos\alpha$, and S (sag) $= \frac{l^2\cos\alpha}{4a}$, $A = \frac{Sl}{3} + \frac{Sl^2\sin\alpha}{6a}$. Perhaps Mr. Tonnesen will look into this and see whether I have made any oversight in my criticism.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(Read at May Meeting, 1909.)

By Jas. Gray, F.I.C. (Member).

DISCUSSION.

Dr. J. Moir (*Vice-President*) (*contributed*): This is a most useful paper, and I am surprised that no one has discussed it. Shall I be accused of prejudice if I suggest that its fate might have been different had it tended towards reduction of working costs, instead of helping (as it does) those on the other side of a lime contract?

My criticism is that the author should have worked out his figures more completely. I find that the results are misleading as they stand, as the absorption of CO_2 is hidden by the fact that the increase in total weight depresses the percentages.

I will take the experiments in turn and discuss them. In No. 1, 7·28 grams CaO were present to begin with, and at the end 7·102 grams CaO (viz., $63·78 \times \frac{111·35}{1000}$); the difference 0·178 must have become $CaCO_3$, because the absorption of water does not *per se* alter the total CaO originally present. Hence, at end, $\begin{cases} CO_2 \text{ absorbed} = 0·140 \\ H_2O \text{ absorbed} = 0·995 \end{cases}$, or about $\frac{1}{8}$ of total increase of weight is CO_2 and $\frac{7}{8}$ water. Considering the insignificant proportion of CO_2 in air, the amount taken up is wonderful.

Experiment 2, continued for ten days, is however much more striking. Starting as before, at the end there was 6·077 gram, total CaO (viz., $48·44 \times \frac{125·45}{1000}$). Hence CO_2 absorbed = 0·945 and H_2O absorbed = 1·600 (a ratio of less than 2 to 1).

Hence the final (product 12·545 gram) consisted of:

Free CaO = 1·099 gram.
$Ca(OH)_2$ = 6·578 „
$CaCO_3$ = 2·148 „
Original impurities = 2·720 „

12·545 gram.

In other words the sample became fully hydrated (except for enclosed grains) probably in much less than 10 days, and thereon commenced to lose water again, just in proportion as it gained CO_2 thus, $\overline{CaOH}_2 + CO_2 = CaCO_3 + H_2O$.

There is something wrong in experiment 3, as the $CaCO_3$ found does not agree with that calculated from the assumptions; probably the 8·1% of original impurities did not consist entirely of $CaCO_3$ as assumed.

By the way, Mr. Gray's pure lime must have been pretty rotten stuff—only 91 and 88% CaO—and I suggest that he try this again before his reply comes on. There is no reason why he should not get 99·5% CaO to commence with; and two experiments with this, one for two or three days and the other for a fortnight, would clear up the points I have discussed. But, of course, his main contention—that the lime-sellers are cheating themselves—is quite clear already.

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

(Read at May Meeting, 1909.)

By the Portland Metallurgical Society (edited by Thos. B. Crowe).

DISCUSSION.

Dr. J. Moir (*Vice-President*) (*contributed*): I have to criticise some of the chemistry in this paper. In it the authors seem to have gone on the principle of *omne ignotum pro magnifico*, because whenever they "get stuck" for an explanation, they propound one depending on the supposed properties of any out-of-the-way substance that takes their fancy.

The equation on p. 398 is ridiculous, because ammonium persulphate—which, by the way, should be written NH_4SO_4—does not give rise to SO_2 under any circumstances whatever.

Again, at bottom of first column of p. 399, they assert that NH_4SO_4 liberates iodine from KI in alkaline solution. I should like to know how they know that—seeing that unless one acidifies, no iodine can be recognised; and that when one acidifies the reaction is between H_2O_2 and HI, and no hypoiodite need be assumed. The fact that CNI cannot resist alkali has been in the text-books for 20 years; also the solubility of Ag in NH_4SO_4 has been applied in photography for nearly as long. As for the statement about sodium thiosulphate at the top of p. 400, I can only say that it is not borne out by facts. The only conceivable substances producible from pyrites are SO_2 and H_2SO_4, and I strongly advise the authors to make further researches into the subject without further ado. What probably really did happen was that the already-existing oxidation of cyanide by H_2O_2 was accelerated by a trace of iron compound which had got dissolved. Have the authors tried replacing the KI by common salt?

Mr. C. H. Cropper *(Member) (contributed)*: The researches upon the telluride gold ores of Cripple Creek by the authors interest me as one having had experience in the treatment of telluride ores of Kalgoorlie, W.A. While it may be true that the history of the metallurgical treatment of the Cripple Creek ores has been attended by many failures, it can hardly be said that such has been the case in respect of the Kalgoorlie ores, and it is very evident that there is a wide difference in the solubility of the gold in telluride minerals. The ore being treated at Kalgoorlie is for the most part a sulpho-telluride, and the metallurgists on the field have met with a large measure of success in dealing with the problem before them. In the early days of the field, when the grade of ore was high and the extractions low, it was discovered that tellurium was a factor to be reckoned with in future treatment. Bromo-cyanide enabled better extractions to be obtained, although the residues contained high values. At the present day the average grade of ore has fallen considerably; the highest may be taken at 55s., the lowest at 28s. In the constant effort to reduce the value of residues the economic factor has prevailed, inasmuch as an extraction of 85% on a refractory ore may show a better cost sheet than a 95%. As to which of the two processes in vogue is the better, viz, dry crushing or wet crushing, is a matter of opinion; but the facts are that with the dry crushing and all roasting plants the residues are under 1 dwt. without the use of BrCy, while the wet crushing residues are $1\frac{1}{2}$ dwt., the raw slime only being treated with BrCy. Total treatment costs, as they are to-day, range from 8s. 8d. per ton to 10s. 8d. for wet crushing, from 10s. up to 11s. 8d. per ton for dry crushing. The capital outlay on a dry crushing plant is greater than for wet crushing, but the difference would be compensated for in the lower residues obtained; greater improvements in roasting furnace design have advanced dry crushing practice. With the wet crushing plant, concentration of the mineral is employed, but there is a proportion of very fine mineral which is carried away in the slime and which is the cause of the higher values in residues.

Telluride ores necessitate slow treatment under our present knowledge of the means for dissolving the gold, and anything which is going to shorten the time of treatment will effect a considerable saving in labour and outlay of plant. Research work in the laboratory is a useful indication as to what may happen in running the plant, but there are conditions which must not be overlooked in handling large tonnages in the practical running of the plant which will often make a commercial failure of experimental facts. In dealing with telluride ores of the present grade at Kalgoorlie, where sliming is of such importance, the quality of the slime may vary from time to time according as the nature of the ore varies in respect of making the slime more impervious, so that the physical condition of the slime may partly be the cause of making a residue assay higher by rendering the complete washing out of the dissolved gold difficult. In the actual running of a plant the condition of the solutions is the greatest factor for a good extraction, and what is required by the metallurgist to-day is a better insight into the chemical re-actions occurring in the solutions as to how and where they occur, but unfortunately the time for research work is usually not at his disposal. From a commercial point of view it seems to me that the cost of chemicals mentioned might be a hindrance to their use in practice, nevertheless it will be interesting to read the results of a practical application of the facts put forward by the Portland Metallurgical Society.

The proceedings then closed.

Contributions and Correspondence.

NOTES ON CLASSIFIERS.

For the benefit of those desirous of data in regard to classification of the mill pulp for tube milling, I send these notes on our experience of cones for that purpose. The original idea was to utilise four cones, 5 ft. 6 in. × 4 ft. 5 in., in place of the usual wooden spitzlutte, these were arranged in parallel so that each cone would have its own proportion of the pulp to classify, instead of classifying successively by running the spitzlutte in tandem.

The principal trouble which was experienced when tube milling was started was caused by the choking of the launders of the tube mill system; this continued to such an extent as to necessitate frequent stoppages and delays. The cause of this trouble I found to be the high pressure of the hydraulic head of the separators, which allowed too great a percentage of fine sands through the nozzles into the tube mill system, instead of permitting its delivery by overflow, to the cyanide works. A separator working in an ideal manner will, I take it, immediately return all the sands which are fine enough for treatment by cyanide, to the cyanide works. To return them to the tube mills is obviously waste of time and work, as well as unnecessary wear and tear.

The reduction of the size of the nozzle, which is generally resorted to, is bad practice, leading as it does to continual choking of the separators. It stands to reason that if the separators are too deep, the sands, both coarse and fine, will fall to the bottom, because if the separators were deep enough they would act as water separators. The remedy, then, is to find out the proper depth of the separators, for a pulp produced by a battery milling with a certain screen.

Another condition which militates against good classification is the conical shape of the separators. If you consider the course of a small representative portion of pulp through the separator, this portion will follow a downward course to a certain point, at which the various component particles will separate; the heavier will continue sinking to find their way through the nozzle, while the lighter and finer will start upwards with a certain initial velocity. This portion of pulp being sufficiently fine for cyanide treatment should be kept rising with its initial upward velocity kept constant; certainly not with a velocity constantly and uniformly decreasing, as obtains with a conical separator, for it is obvious that with a cone the velocity of the rising particles will gradually decrease as the sectional area of the cone increases towards the base. This is proved by the conical separators working crowded, and therefore changing their original internal shape.

To correct this, I first introduced a cylinder to fit in, half way down the conical separators, and obtained much better results, as the separators then worked quite freely; after this I had separators made of different dimensions, in order to find out the most efficient. The results of these tests proved that the factor of most importance was the height of the separator, the diameter not being quite so important. I have now concluded that a cylindrical separator, 3 ft. 6 in. in depth and 3 ft. 3 in. in diameter, is the most suitable for our pulp. The cylindrical shape is of cheaper construction, and as the sands accumulate at the bottom and form a natural and most efficient cone, the cylinder is protected from the wear. It was found that this natural cone has its sides at about 45°, and the separation occurs in the cylindrical portion above this cone.

In conclusion, I would like to state that there is absolutely no difficulty in obtaining a good overflow from the separators with practically not more than 2% or 3% of +60 material; the point is to get rid of the fine sands at once and return practically the whole of the +60, and as little of the +90 material as possible.

S. Aimetti.

New Heriot G. M. Co.,
5th May, 1909.

Obituary.

It is with deep regret that we record the death of Mr. Wager Bradford, M.I.M.M., on July 9th, from pneumonia, after about a week's illness. Mr. Bradford, who joined the Society in 1898, was one of its most active members, was elected a Member of Council in July, 1903, and Vice-President in July, 1904, to which position he was re-elected in 1905 and 1906. Owing to his leaving South Africa on holiday leave Mr. Bradford declined nomination for the Presidentship in the following year and in succeeding years, for the reason that his duties as manager of the Langlaagte Deep, Ltd., necessitated his undivided attention. In 1906 he also occupied the position of President of the Association of Mine Managers of the Witwatersrand, which he filled with dignity and success.

Mr. Bradford was born in 1863 at Stockton, Cal., U.S.A., and was a son of Judge Abram C. Bradford. Educated privately he subsequently graduated with the degree of B.A. from Hamilton College, New York, in 1885. He came to the Transvaal in 1896, and filled the position of manager of the Buffelsdoorn Estate and Buffels "A," Paarl Central G. M. Co., and Langlaagte Deep, Ltd. On the amalgamation of the latter property with the Crown Mines, Ltd., recently, he was transferred to the managership of the Robinson G. M. Co., Ltd. Mr. Bradford was held in the highest esteem by all with whom he came in contact, and the Society had the advantage of much valuable assistance and advice from him whilst a member of the Council.

A most impressive memorial service, conducted by the Right Revd. the Lord Bishop of Pretoria, was held in St. Mary's Church, Johannesburg, on the 15th July, and was very largely attended. Bishop Furze made an eloquent tribute to the many good qualities of the deceased, whom we shall all miss greatly from our mist.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Electrochemistry of Solution of Gold in Potassium Cyanide.—"In 1893 McLaurin proved definitely that gold will dissolve in potassium cyanide solution when there is oxygen present, but that, if no oxygen is contained in the cyanide solution, the gold remains unacted upon. From pure hydrocyanic acid and potassium hydroxide dissolved in water freed from air by long boiling, McLaurin prepared a solution of potassium cyanide. A Dumas bulb was partly filled with this solution, a strip of pure gold was placed in the neck of the bulb, and, while the solution was boiling, the neck was sealed. The gold was allowed to remain in the solution for 24 hours, the gold losing in weight 0·0002 gm. The gold was then allowed to remain in contact with the same solution in the presence of air for 24 hours, during which time it lost 0·0083 gm. This is the best experimental evidence we have that gold requires oxygen for its solution in potassium cyanide.

Although other investigators took up the study of the solubility of gold in potassium cyanide, little of any great value in a theoretical way was brought forward until Christy took the subject up from an electrochemical standpoint, his work being published in 1902.

Christy made use of an Au, KCN, Hg_2Cl_2, Hg couple, measuring the electromotive force and finding that the positive current flows in the cell from the gold electrode to the mercury, gold going into solution and mercury being deposited. He then suggested that, should an 'oxygen electrode'—that is, platinised platinum saturated with oxygen gas—be substituted for the calomel electrode, the gold would go into solution in the cyanide, and the platinum would receive the same electrical charge that he found in the case of the mercury.

In support of this he performed the following experiment: A strip of gold was suspended in a solution of potassium cyanide that had been saturated with oxygen. A second strip of gold was suspended in a similar solution, freed from oxygen by boiling and subsequently protected by a layer of oil. The two solutions were connected by means of a liquid bridge, and the two gold strips by means of a wire and galvanometer. Gold dissolved in the oxygen-free solution and a current was shown by the galvanometer to be flowing in the solutions from the oxygen-free solution to that containing the dissolved oxygen.

The following explanation of the phenomenon has occurred to us: Christy's cell is analogous to the simple cell constructed in the same manner by substituting zinc in sodium sulphate for gold in oxygen free cyanide, and copper in sulphuric acid for gold in oxygenated cyanide. Here the current flows in the solutions from the zinc to the copper, zinc dissolving in one solution and hydrogen being liberated on the copper. The zinc dissolves in the absence of acid just as in Christy's cell the gold dissolves in the absence of oxygen.

In the zinc-copper cell the solution pressure of the zinc drives zinc ions into the solution, the hydrogen ions in the other solution being thus forced out, giving their charges to the copper, with production of a current outside the cell from copper to zinc. In Christy's cell, the gold on account of its solution pressure sends gold ions into the solution. Hydrogen ions in the oxygenated cyanide vessel (due to the slight ionisation of water into $H^{\cdot}$ ions and OH' ions) tend to be forced out just as in the zinc-copper cell, but this does not occur. The dissolved oxygen sends, by virtue of its solution pressure, oxygen ions, O'', into the solution. These O'' ions with the $H^{\cdot}$ ions form OH', so that the cell acts like a bichromate cell or any other oxidation cell, instead of like a simple zinc-copper couple. The OH' ions thus left in the solution form with the $K^{\cdot}$ ions dissociated KOH.

In view of the above considerations, it seemed interesting to us to set up an Au/KCN/O—Pt couple. As expected gold dissolved rapidly in the cyanide, the rate varying with the concentration of the cyanide, and the concentration of the oxygen in the oxygen electrode, the current passing in the couple from gold to oxygen. Solubility determinations are needless and impossible on account of the difficulty of excluding air from the cyanide, and of the variability in the potential of the oxygen electrode.

If it were possible to supply oxygen to the platinised platinum at a high enough speed and at the same time exclude all air from the cyanide solution (other things being equal, such as physical condition of the gold, etc.), the solubility would undoubtedly be directly proportional to the current, according to Faraday's law. As a matter of fact, the physical condition of the gold varying and causing a varying occlusion of oxygen on its part, the oxygen held in solution by the cyanide, and the varying amount of oxygen occluded by the platinised platinum undoubtedly cause the gold to dissolve at a rate not proportional to the current flowing through the cell. Hence the solubilities indicated in the following table are not to be considered more than fair approximations:—

Strength of KCN.	Time.	Au Dissolved.	
0 05%	2 hr.	0·0219 gm.	Circuit closed.
0·05	2	0·0036	Circuit open.
0·1	2	0·0307	Circuit closed.
0·1	2	0·0055	Circuit open.
0·15	1	0·0193	Circuit closed.
0·15	1	0 0033	Circuit open.

We also measured the potentials of gold in varying concentrations of cyanide against the oxygen electrode, with results which agree with the measurements of Christy and others made against the normal calomel electrode:—

Strength of Solution.	Gold Potential.
0·1%	0·12 volt.
0·15	0·14
0·65	0 21
0·8	0·22
1·0	0 29

The equation for the solubility of gold in potassium cyanide, often called Elsner's equation, is

$$4Au + 8KCN + 2H_2O + O_2 = 4KAu(CN)_2 + 4KOH$$

The gold dissolves on account of the formation of innumerable short-circuited gold-oxygen cells. The gold sends 4Au ions into the solution by virtue of its solution pressure, and at the same time the dissolved oxygen sends $2O''$ ions into the solution by virtue of its solution pressure. The 4Au ions react with the $8CN'$ ions of the dissociated KCN, forming $4Au(Cn)_2'$ ions, and the $2O''$ ions react with $2H^{\cdot}$ (which are present from the dissociation of $2H_2O$ molecules into $2H^{\cdot}$ and $2OH'$), forming $2OH'$ ions. Thus the ions present before solution are

$$4Au^{\cdot} \times 8K^{\cdot} + 8CN' + 2H^{\cdot} + 2OH' + 2O''$$

and after the reaction

$$8K^{\cdot} + 4Au(CN)_2' + 4OH'$$

or written in the form of an equation,

$$4Au^{\cdot} + 8K^{\cdot} + 8CN' + 2H^{\cdot} + 2OH' + 2O'' =$$
$$8K^{\cdot} + 4Au(CN)_2' + 4OH'$$

Oxygen need not necessarily be the carrier of the four negative charges required. Any substance capable of furnishing these charges would cause gold to dissolve in potassium cyanide, or any mechanical means of producing these four charges would have the same effect. In this latter case the equation would be

$$2Au^{\cdot} + 4K^{\cdot} + 4CN' + 2H' + 2OH' =$$
$$4K^{\cdot} + 2Au(CN)_2' + 2OH' + 2H$$

and gaseous hydrogen would be liberated from the cyanide solution as the solution of the gold proceeded.

Cyanogen-bromide added to a potassium cyanide solution is a very effective gold solvent. The reaction is $Au_2 + 3KCN + BrCN = 2KAu(CN)_2 + KBr$.

Considered electrochemically the reaction is as follows:—

The solution pressure of the gold sends $2Au^{\cdot}$ ions into the solution while the solution pressure of the cyanogen bromide sends a Br' ion and a CN' ion into the solution. The $2Au^{\cdot}$ ions react with the $4CN'$ ions, forming $2Au(CN)_2'$ ions, so that the equation would be written

$$2Au^{\cdot}+3K^{\cdot}+3CN'+Br'+CN'=$$
$$2K^{\cdot}+2Au(CN)_2'+K^{\cdot}+Br'.$$

It should therefore be possible to set up a cell in which gold and solid cyanogen bromide were the electrodes and potassium cyanide the electrolyte. The current would flow in the cell from the gold to the cyanogen bromide, both being consumed, just as in the gold-oxygen cell, the current flows in the cell from the gold to the oxygen, both being consumed."—EKELEY and TATUM, *Western Chemist and Metallurgist.—Electrochemical and Metallurgical Industry*, April, 1909, pp. 156-7. (**A. McA. J.**)

CAUSE AND PREVENTION OF RUST.—"Studies have recently been made of the causes which produce, decrease, or accelerate rust. It is generally believed that carbonic acid plays an essential and necessary rôle in the oxidation of iron. Such, however, is not the case. Iron rusts in air quite free from carbonic acid, or even in water containing no trace of this gas, whilst carbonic acid alone, in the absence of oxygen, exerts no action on iron in pure water. On the other hand, we know that in dry oxygen iron does not rust. What, then, is the cause of rust? It is of electric origin, and due to the lack of homogeneity in the iron. Any impurity incorporated in the metal forms an electric couple; water is decomposed, and the nascent oxygen which accumulates at the positive pole attacks the metal. Numerous facts corroborate this theory. Thus, corrosion of iron in contact with zinc, under water, is considerably decreased at cost of the latter metal. On the contrary, if the iron is in contact with copper or nickel, corrosion is much greater. Contact between two pieces of iron of different structure suffices to produce rust. How can iron be preserved? If it is plunged into a liquid, the addition of potassium or alkaline salts will protect it; but they must be in a rather concentrated solution, otherwise the remedy is worse than the evil. Chromates, on the contrary, are perfect preservatives, either in solution or as paints, and, strange to say, when iron is removed from chromated water or the paint is washed off, it will still remain proof against rust for a certain time. Thus, these observations perfectly demonstrate the value of chromate paints recommended during the last few years."—*La Nature*, Feb. 13.—*London Mining Journal*, April 10, 1909, p. 460. (A. R.)

DE-RUSTING OF IRON IN FERRO-CONCRETE.—"The cause of the disappearance of rust from iron bars, etc., used in the erection of ferro-concrete structures, has been traced by Rohland (*Stahl und Eisen*, March 17) to the presence of acid carbonates and sulphates in the cement, these salts dissolving the iron oxide and leaving the metal bright. The cement in setting absorbs carbonic acid from the air, thus forming the necessary acid carbonates; and experience has shown that the de-rusting process is effected whilst the concrete is setting and commencing to harden. This discovery affords an additional guarantee for the safety of ferro-concrete structures, inasmuch as the metal is protected from rusting by the alkaline reaction of the cement during the mixing process, and any rust on the bars is removed by the acid carbonates at an early stage in the erection of the structures."—*London Mining Journal*, April 10, 1909, p. 460. (A. R.)

METALLURGY.

WATER REQUIRED FOR CONCENTRATING MACHINERY.—"The following approximate data are given by the General Engineering Co., of Salt Lake City for use in making preliminary estimates only.

Stamps and Pulverisers.—For each 800—1,000-lb. stamps, from 3—6 gallons per minute per stamp. On medium hard ores with 1,000-lb. stamps crushing to 20 mesh, 5 tons per stamp with 3½—4 gallons per minute is good work. This equals a pulp of from 4—5 to 1. Chilian and Huntington mills on similar ore will require a total of 5—6 tons of water per ton of ore passing the screens.

Jigs.—For each 18 in. × 36 in. compartment of Harz jigs treating:

1½— 2½ mm. particles, 2— 4 gallons per minute
3½— 5 mm. particles, 5— 7 gallons per minute
7 —10 mm. particles, 8—14 gallons per minute
15 —20 mm. particles, 21—28 gallons per minute

This is in addition to the water in the feed which can be assumed to be not less than 1—1 water and ore.

Revolving Screens (Spray Water).—Each 36 × 72 screen will require on:

1½— 3½ mm. holes, 15—25 gallons per minute
3½— 7 mm. holes, 10—15 gallons per minute
7 —15 mm. holes, 5— 8 gallons per minute

Callow T B Screens (24 in Duplex).—Water in pulp = 3½—4 tons water per ton of feed.

Shaking spray, 6—10 gallons per minute
Oversize spray, 8—12 gallons per minute

Wilfley Tables.—5—10 gallons per minute.

Six-Foot Frue Vanners.—1½—3 gallons per minute.

Hydraulic Classifiers.—A very variable quantity. Richards gives from 10—50 tons water per ton of pulp, or an average of 20 tons for all the mills visited by him."—*Mines and Minerals*, March, 1909, p. 381. (A. R.)

CYANIDING GRAPHITIC ORES.—"The ore in the bottom levels of the Great Boulder mine carries a greater percentage of graphite than that of the shallower workings. No entirely satisfactory theory as to the origin of this material in the Kalgoorlie ore bodies has yet been advanced. It is of course quite impossible that it is in any way organic, as no matter of organic origin has ever been associated with the Archæan or pre-Cambrian rocks to which these ore bodies belong. The elements, however, are all present in the rocks which could cause the formation of graphite inorganically, and it is highly probable that it was derived from some of the alkaline carbonates by a series of chemical reactions assisted by the enormous dynamical stresses and temperature obtaining in these plutonic rocks. The graphitic material in the Kalgoorlie lode bodies does not as a rule occur disseminated throughout the ore, but appears to be deposited along the cleavage planes of the rocks. This lends colour to the theory of its formation by the dissociation of a carbonate. It has been a moot point among metallurgists whether the presence of graphite does interfere with the extraction of gold by causing its precipitation prematurely from the auro-potassic cyanide solutions in the agitating vats. No doubt when unroasted ore is being treated graphite does cause this premature precipitation from the solutions, for the writer has knowledge of an interesting experiment recently made in Kalgoorlie in order to settle this point. The experiment was conducted with a gold-bearing cyanide solution to which a finely powdered raw graphitic ore, containing 3% graphite, was added and agitated. Precipitation of gold soon commenced, and after a two hours mechanical agitation, analysis of the solution showed that from 30% to 50% of the gold was precipitated. On taking another portion of solution and introducing similar, but roasted ore with the subsequent agitation, assays showed no appreciable loss of gold. The results of this experiment are manifestly important, for it goes to show that in this particular ore experimented on, in order to nullify the precipitative action of the graphite on gold-bearing cyanide solutions, it is neccessary to first roast the ore. It further goes to show that it is not actually the graphite itself that causes the precipitation, for the temperature employed in the oxidising roast of ore is not such as will in any way affect the graphite, which is, as is well known, a very refractory substance except under the most extreme temperatures. It appears then that it is something associated with the graphite that causes the precipitation, and it has been suggested that it is perhaps hydro-carbons which the moderate heat of roasting is sufficient to eliminate, and the graphite left in an inert form."—*Australian Mining Standard*, March 3, 1909, p. 223. (A. R.)

Fluxes for Silver Residues.—"In electroplating with silver, and in any other technical processes in the industrial arts, there are large quantities of silver tailings and precipitates, for the proper recovering and utilisation of which smelting is necessary. These by-products are of such different characters that if they were all melted together, there would result no pure material, but the mass would ball together in the crucible and form a half-melted lava-like material, in which, as far as industrial purposes are concerned, the silver would be in a worse condition than before. In order to smelt such a mass and get therefrom a pure silver bar, there must be used a flux, for the purpose of setting free the oxide that has been formed, and permitting the metal to form a single mass.

The flux described below by a writer in the *Deutsche Metall Industrie Zeitung* has been used for a long time in establishments that smelt silver tailings and sweepings, and has given the highest satisfaction. Originally it was used in the cyanide process for the extraction of gold from auriferous ores.

This process consists essentially in grinding the ore fine and then leaching the resulting material in wooden tubs by a weak solution of potassium cyanide. The latter substance dissolves out the gold and silver in the ore: the clear solution is drawn off, and the noble metals precipitated by the addition of zinc scrap. The deposit thus formed consists of gold, silver, copper and a greater or less quantity of zinc. When dried it is smelted with a flux, and the mass sent to the refinery.

As this flux has given good satisfaction in the process just named, experiments were made to determine whether or not it could be employed to advantage for other kinds of silver-smelting, and for other sweepings, etc., containing silver or gold. Favourable results were obtained therewith. The flux consists of 3·06 kg. of calcined soda (soda ash), 0·9 kg. of borax glass, 0·4 kg. of sand, 126 gm. of fluorspar. Borax glass is to be used, not borax itself, as the latter, when heated, swells too much, and this makes trouble in smelting the silver-bearing or gold-bearing materials. Borax glass can be obtained by melting crystallized borax and sand in a graphite crucible, until all is clear, and then pouring the liquid. Any kind of sand may be used, only it must not be coarse.

The silver-bearing material is mixed with the flux, and the mixture melted in a graphite crucible. This latter must not be quite full, as the mass swells a great deal in melting. The amount of flux necessary depends on the kind of material being melted. For most such material, the flux is employed in the proportion of one thereof to two parts of the former.

Where the tailings or the sweepings are all chlorides, more flux—even equal parts by weight—is necessary. Fluorspar is added to make the resulting mass more liquid. In a slag of thick consistance, such as is produced by soda-ash alone, the metal does not separate, but attaches itself to the slag, so that much thereof remains on the latter. Where, however, fluorspar is used, the slag is rendered so liquid that all the metal, once melted, falls at once to the bottom of the crucible.

Where there is much iron in the material to be smelted it must be removed by the aid of magnets. As iron has a very unfavourable effect on the process of smelting, it must in no case be allowed to remain in the material. Incidentally, this latter must be perfectly dry before being mixed with the flux."—*Canadian Mining Journal*, May 15, 1909, p. 305. (A. R.)

Slime Settlement.—"Lime slaked with hot water, according to Leo D. Bishop, causes slime to settle with nearly twice the rapidity of lime slaked with cold water. Mr. Bishop also calls attention to the great difference in effectiveness of different limes as coagulators for slime. The composition, as well as the perfection of burning, are important. In short the more CaO the lime contains, the more it is worth."—*Indian Engineering*, April 17, 1909, p. 253. (A. R.)

Conical Pebble Mill.—"The prominent feature of the design is that it is of conical and not of cylindrical shape. The result of this design is a sizing action, whereby the pebbles and the particles of material to be crushed arrange themselves according to their size at different sections of the cone. The illustration shows the comparative peripheral speeds in feet per minute at different sections of the mill, and a diagrammatic comparison of the size and weight of the crushing pebbles and the particles to be crushed. The great increase in the difference as the sizes decrease is comparatively equalised by the diminished fall and reduced peripheral speed.

Or, in other words, the principles involved of gradual reduction in the same apparatus through the application of a combination of the laws of gravity and centrifugal force, result in classified zones of crushing pebbles and particles to be crushed. A device is thus produced which, while revolving with the same speed in revolutions per minute, contains a multiplicity of zones with gradually changing peripheral speeds.

The novelty of the section raises the question as to the main cause of the sizing. But whatever the ultimate cause may be, the result of this sizing effect is that large pebbles pound on large particles of the material to be crushed with high momentum, and smaller pebbles on smaller particles with a less momentum. Mr. Hardinge says the principle of crushing in his mill is to use a nail-hammer upon a nail and a tack-hammer upon a tack, *i.e.*, to adjust the power and mechanism to the work desired. This results not only in great uniformity of the ground product, but in a great reduction of the power required for grinding.

The following results were obtained by the Cobalt Central Mines Co., at Cobalt, Ont., in a series of screen tests on the same ore (diabase gangue) extending over a period of several days. They show the adaptability of the Hardinge mill to fine and coarse grinding for cyaniding or concentration.

Mesh.	% on 8.	% on 14.	% on 20	% on 40.	% on 60.	% on 80.	% on 100.	% thr 100.
Heads ...	16·7	45·1	6	13·9	6·8	4·4	·3	7
Tails ..	—	—	4·8	9·8	22·8	22·3	·9	39
First adjustment.								
Tails ...	—	—	—	1·3	8	27·5	2·5	60
Second adjustment.								
Tails ...	—	—	—	—	1·4	10	2	87
Third adjustment.								

The average capacity was above 2 tons per hour; the power required, 12 h.p. to 15 h.p.; charge of pebbles, 2,000 lb.; consumption of pebbles, ⅓ lb. (¼ e.) per ton of ore.

The total weight of a conical mill (except the lining), having a capacity of 2 tons to 4 tons of material per hour, is 8,500 lb. and the floor space occupied is 7 ft. × 10 ft. The charge of pebbles is about 2,000 lb., and the power required for operation is from 12 h.p. to 15 h.p. These mills can be arranged singly, in series or in tandem, and the large amount of aeration which takes place during the crushing suggests promising results in cyaniding ores during crushing, which, if attained, will greatly simplify present milling practice."—H. W. HARDINGE.—*Electrochemical and Metallurgical Industry,* Jan., 1909, p. 47. (A. McA. J.)

THERMAL ANALYSIS AND INTERMETALLIC COMPOUNDS.—"Of the various methods of determining the quality of metals used in engineering, the process known as 'thermal analysis' as introduced by Prof. Tamman is beginning to assume an important phase. This method is based upon an examination of the rate of cooling of the molten metal, each metal having apparently its own characteristic cooling curve. The Saladin double-mirror galvanometer is an instrument designed to record the manner in which cooling takes place in a molten metal, and is being shown by Messrs. Siemens Brothers & Co. (Limited), of Woolwich. The apparatus consists of a pair of standard pillar galvanometers mounted on the same base, and so arranged that their mirrors are optically connected by means of a prism. This prism is the essential part of the apparatus, its function being to convert the horizontal movement of the beam of light from the mirror of one galvanometer into a vertical movement on the second mirror, so that the movement of the second mirror results in a curvilinear motion which can either be directly observed or recorded at will on a stationary photographic plate. The beam of light is reflected from an external source in the usual way, and on passing through the prism from the first to the second mirror, records the variation if any, in the fall of the temperatures under test. To use the instrument, one of the galvanometers is placed in circuit with two pyrometers in series. The first of these pyrometers records a cooling of a metal which is known to have no recalescence points, while the second pyrometer records the cooling of the metal under test. The second pyrometer is also connected up with the second galvanometer. By this arrangement the difference in temperature between the two metals is recorded from instant to instant, and the photographed curve indicates any changes in the uniformity of the rate of cooling of the metal undergoing analysis. The instrument is of particular value in the study of inter-metallic compounds, the analyses of which are based upon the assumption that heat is liberated when a metallic compound is formed in a cooling metal. In such a case the photographic curve rises for a short period and then resumes the declining curve normally associated with the cooling metal. The cooling curves of iron and steel are almost invariably of this character, and already sufficient is known of these cooling and combining processes to enable the quality of iron to be determined by observing the nature of the recorded curve. Mr. Saladin is employed in the steel works of Messrs. Sneider & Co., at Creusot, France."—*Times Engineering Supplement,* Feb. 3, 1909. (J. A. W.)

FILTERING GOLD SLIME.—"In small cyanide plants where a clean-up press is not provided, the filtering and washing of the gold slime after acid treatment of zinc is generally found to be a slow and tedious process, but the following arrangement can be very cheaply installed. It will save much time in the clean-up, and is, in fact, with small plants, a good substitute for a filter-press. The whole operation of washing and filtering can be done in the acid tub, and consists simply of applying the principle of vacuum filtration to the clean-up. The appliances necessary are a filter frame, a vacuum chamber and a vacuum pump.

The frame is constructed of ¾ in. piping, screwed together in the form of a square, with a T-piece let into one of the sides, to which pipe and hose can be fitted; the sides of the pipes towards the centre have ¼ in. holes drilled every 6 in. A filter cloth to fit the frame is made of two pieces of good canvas or cotton duck, between which are two layers of coarsely woven cocoanut matting. The four layers are all held together by a few rows of stitchings, about 3 in. apart. The cocoanut-mat layers are made so as to fit easily inside the frames, the duck being left large enough so that the top and bottom layers can overlap the frame on opposite sides and can be well sewn together all round. This then forms an efficient cell for filtering solution. The frame is made of such size as to fit the bottom of the acid tub with about 2 in. clearance all round.

The vacuum chamber can be conveniently made from a sulphuric acid drum, fitted with pipe connections consisting of a pipe with cock to vacuum pump; a pipe with cock and hose to filter frame; a cock for running out clear acid solution; this should be placed 6 in. above the bottom of the drum; a cock in centre of bottom for cleaning out drum. The pipe connecting filter frame to drum is carried down 1 ft. inside the drum, so as to obviate the carrying over of any of the acid solution to the vacuum pump. Drums with convex ends should be chosen as they can be thoroughly drained when cleaning out.

After the acid treatment is finished, the tub is filled with water, and the contents well stirred, then after the precipitate has settled, the clear solution is decanted off. The remaining gold sludge is well stirred and the frame is put into the tub, and the cocks to the vacuum chamber and pump are opened. The frame should lie horizontally on the bottom of the tub and remain there until all the liquid is drawn off; it can then be lifted from the bottom and placed upright against the side of the tub while sufficient water is run in for washing. The slime adhering to the frame is scraped off, and, together with the rest of the slime in the bottom of the tub, is thoroughly stirred up with the water. The frame is again laid horizontally on the bottom of the tub and the vacuum cocks are opened as before. One washing in most cases will be ample, but can easily be repeated if insufficient. After the liquid is all drawn off, the vacuum is maintained for a time so as to air-dry the slime as much as possible. The slime is then collected from the frame and from the bottom of the tub; with careful scraping very little need be left on either. What is left can, however, be collected by washing and sponging the tub and frame with about half a bucketful of water. The washings can then either be filtered through a small calico filter, or use may be made of the filter frame, by laying it down flat and, with the vacuum on, slowly pouring the washings upon one side of it; when dry the slime is carefully scraped off. If the filtering is slow with only one frame, there is no reason why two or more should not be used."—EILER JENSEN, *Journal Chamber of Mines, W.A.—Engineering and Mining Journal*, May 1, 1909, p. 902. (W. R. D.)

ELECTRO-CHEMICAL AMALGAMATION.—"After a cursory examination of mining periodicals during the last two years I have been struck with the absence of any considerable discussion regarding improved methods of gold extraction by amalgamation. So far as I can learn, the mill plate of to-day is essentially the same as the mill plate of forty years ago. I believe in a very few cases some daring souls had the temerity to lengthen the plates beyond the ordinary, but as this innovation is in such direct conflict with precedent it is no wonder that the great mass of mill men refused to consider it. Many of the mining processes of to-day seem to consist in doing something that somebody else has done; in other words, instead of originality and adaptation, we have imitation.

Yet authorities all admit that the mill plate of today is a very imperfect device. It is satisfactory only in a few cases where the values are found under the most favourable conditions. The average mill plate is losing from 5% to 25% of the assay value, and there are many free gold veins that cannot be worked by ordinary plate amalgamation. The large amount of gold that exists in tailings after passing plates calls forth a host of supplementary devices for the extraction of values therefrom. As high as $500,000·00 has been expended in one auxiliary plant to treat tailings, and there are millions of dollars now locked up in tailings dumps that have been abandoned.

These considerations have for years caused progressive metallurgists to carefully consider the whole question of gold extraction in an attempt to find the better way which must always exist. Some sixteen years since, Mr. C. G. Warnford Lock read a paper on amalgamation, which may be found in vol. i., p. 205, 'Proceedings of London Institution of Mining and Metallurgy'; in this paper Mr. Lock discusses the various methods of amalgamation and notes their shortcomings. He then proceeds to tell in the following paragraph what he considers a perfect amalgamation. 'The ideal amalgamator should provide for a constantly clean and bright mercurial surface, coupled with mechanical means for forcing each gold particle into contact with that surface.' The author then described a machine which, he considered, met the requirements of a perfect amalgamating device. The amalgamator consisted of a circular pan, 40 in. in diameter, holding 270 lb. of mercury; briefly, a cylinder of slightly less diameter projected an inch or so into the mercury; a much smaller cylinder in the centre of the large one, also projected under the mercury; this small cylinder contained a solution of carbonate of soda, into which projected an anode from a 5 volt 15 ampere generator, the negative lead being connected with the mercury. In operation, pulp and water were delivered into the larger cylinder, the static head being sufficient to force the gangue under the mercury, after which it flowed over the edge of the pan. The electrical current passed into the soda solution, then into the mercury, and then back to the generator.

As the current passed through the solution some of the carbonate of soda was separated into carbonic acid and metallic sodium; hydrogen was also liberated. Mr. Lock stated that the sodium and hydrogen were 'occluded' and spread through the whole body of the mercury. On coming to the surface the sodium combined with the oxygen of the water in the pulp and liberated nascent hydrogen in considerable quantities. He held that in the presence of nascent hydrogen, mercury cannot 'sicken,' cannot oxidise, or become sulphated. It appears that Mr. Lock was not aware that mercury under ordinary circumstances does not oxidise; the scum or slag that sometimes appears on mercury is composed of various impurities and the oxide of foreign minerals contained in the mercury oxides.

But to resume. Mr. Lock also contended that the presence of hydrogen acts as a cleanser on particles of gold passing over the mercurial surface. He also added that free sodium tended to remove impurities from metallic substances, destroying any grease or organic substances that may be present in pulp or water, or attached to the particles of gold. This machine would handle 15 tons in 24 hours, according to Mr. Lock, and he stated that the cost of passing pulp over the device was 6 cents per ton. He closed the article by giving several cases where the above-described device greatly added to the output of gold. In one case 2½ dwt. of gold were taken from the pyrites containing rusty gold that yielded nothing to other amalgamators; in another case the output was increased 62%; in another case the yield of gold was increased ninefold. In the Providence mine in Nevada this machine took out 1 dwt. from slimes and tailings assaying 1½ dwt. per ton, after passing plates, blankets, and vanners; one mine having ores containing arsenic, antimony, and sulphur, yielding 30% of assay values, gave 77% with the amalgamator above mentioned.

Beyond question, the amalgamator described will do all that Mr. Lock claimed for it. Why, then, it may be asked, is it not in general use, since Mr. Lock described the device some sixteen years ago? It is quite possible in protracted use that the mercury would 'flour' and soon pass away with the gangue, as this is almost invariably the case when pulp is forced through mercury. Again, the capacity of the machine is small, probably not more than 8 or 10 tons daily, in actual practice. In the next place it is an electrochemical device, and few mill men or superintendents understand electro-chemics; and sixteen years ago few mill men knew much about electricity. If the machine were installed someone with at least an elementary knowledge of chemistry and electricity would be necessary to look after it. Then there is the fierce opposition to all mill men when a freak machine is in question; it is human nature to resist change of any kind. The mill foreman knows how to handle plates; he does not know anything about 'new-fangled contraptions'; if a new form of amalgamation is installed he might lose his position, as a new man would be needed to run it, etc.

Nearly every dredge master, mine foreman, mill superintendent, etc., hugs this delusion closely to his breast—that he is saving 99% of the values; and the installation of any kind of a supplementary gold-saving machine is opposed, diplomatically or otherwise, as such installation would be an admission that present methods were defective, and that there was something yet to be learned. The average expert is not willing to admit that there is anything in his line that he does not know, and he prides himself on knowing just a little more than the other fellow. In many cases he considers it treason to even discuss the subject of values in the tails.

But to resume the main subject, there is no doubt that the above-described machine will do all claimed for it by Mr. Lock. Experimentation in electro-chemical amalgamation has been continued by Californian investigators, and great improvements have been made in the art. Mr. Lock would hardly recognise the electrolytic amalgamators as they are now constituted. Not only have they been greatly simplified, the cost of operation reduced, and their capacity increased to several hundred tons daily, but their effectiveness has been increased to such a point that all values in pulp not encased are quickly amalgamated, regardless of those conditions which usually prevent amalgamation; platinum in placer sands is amalgamated as easily as gold.

Several forms of electrolytic amalgamators have been constructed or planned, but the simplest device consists of the usual mill plate with an arrangement of graphite anodes, a few inches apart, suspended about $\frac{3}{8}$ in. above the plate. The anodes are connected with a low voltage current, and the plate proper is used as a cathode. More water is used than in ordinary amalgamation, so that the water touches the anodes and completes the circuit. A low amperage is generally sufficient. In some types of machines no free mercury is used, but a solution of bi-chloride of mercury in suitable quantity to supply mercury according to values in pulp is fed into the water supplying the device. It may be stated here that the action of these machines is entirely different from the usual mill plate amalgamation, and deductions drawn from the ordinary practice will not apply in the electrochemical process. In the first place, nascent mercury is deposited on the copper plate by the usual electroplating action; in fact, the apparatus is simply an electroplating device applied to a mill plate. The mercury is plated beautifully over the copper, and in brightness it rivals a polished mirror; the larger particles of gold naturally come in contact with the mercurial coating; the microscopic particles of gold, being practically in solution, are coated with mercury while passing in the water, and the electroplating action of the current deposits them on the copper plate. Nascent mercury has a powerful affinity for gold, so that amalgamation is effected even when pyritic, rusty, coated, or greasy conditions are present.

The deposition of nascent mercury and the formation of amalgam are simultaneous, automatic, and constant; a sodium solution can be also used, from which the electrical energy deposits nascent sodium; nascent hydrogen is also constantly forming. Sodium and hydrogen form an amalgam with mercury, which has a powerful affinity for all metals, and the free chlorine and hydrogen destroy any grease or organic matter that may be attached to the particles of gold. Platinum is readily amalgamated when sodium amalgam is formed If necessary, a special cleansing solution can be used, which will liberate larger quantities of nascent hydrogen and chlorine.

As the amalgam is formed by the electro-plating action of the device, it adheres very tenaciously to the copper plate; large volumes of water have no injurious effects, and there is no scouring effect from pulp or sand. This permits of an increased capacity with all the water that may be necessary.

It is the contention of those who have tested the electrolytic systems that with fine grinding the values in the tailings will fall to such a point that no supplementary treatment will be necessary; also that the immense black sand, beach, river, and desert deposits, together with hydraulic, placer, and dredge tailings, and all low grade material can now be profitably worked, as the electro-chemical systems can be quickly installed at trifling cost."—ELMER ELLSWORTH CAREY.—*London Mining Journal*, May 15, 1909, p. 617. (A. R.)

THE HARDINGE CONICAL PEBBLE MILL.—"The following comparison of actual results, obtained by crushing equal quantities of material in a conical mill and in an ordinary tube mill, is of interest. The charge for the conical mill was of 4 mesh size, while that for the tube mill had already been crushed and passed through a 25 mesh screen. A 0·25-in. gr. contains more than 200 particles of 25 mesh size.

		On 40 mesh Screen. %	On 60 mesh Screen. %	On 80 mesh Screen. %	On 100 mesh Screen.	Thro' 100 mesh Screen.
Conical mill; with 2,000 lb. pebbles; consuming 15 h.p.	Charge	75·5	6 6	5·5	2·3	9·5
	Product		1	5·4	2·5	90·8
Tube mill; with 11,000 lb. pebbles; consuming 48 h.p.	Charge	1·5	13·5	17·0	17·0	53·0
	Product		·5	1·5	·2	96·5

Mines and Minerals, Nov., 1908, p. 160. (K. L. G.)

METHOD OF ESTIMATING WEIGHT OF GOLD IN A QUARTZ SAMPLE.—"The weight of gold in a quartz specimen can be closely approximated, if there be no admixture of pyrite or other foreign material, by a specific gravity method. Let

A = specific gravity of the gold.
B = specific gravity of the quartz.
C = specific gravity of the specimen,
x = weight of the gold.
y = weight of the quartz.
W = weight of the specimen.

Then $$\frac{x}{A} + \frac{y}{B} = \frac{W}{C}$$

and $$xA + yB = WC$$

If now the values for the specific gravities of gold (19·25) and quartz (2·65) be substituted, these equations reduce to

$$x = \frac{W}{18{\cdot}9}\left(C - \frac{7}{C}\right)$$

which gives in a simple expression the weight of pure gold in the specimen. It must be remembered that the specific gravity of both gold and quartz may be different from the values given, owing to silver alloy in the gold, and to variations in the density of the quartz, so they should be determined independently when it is intended to accurately estimate the quantity of gold. In a similar way the ratio between the weights of two minerals, such as galena and sphalerite, can be found, when they contain no other substance intermixed."—*Mining and Scientific Press*, March 6, 1909, p. 345. (K. L. G.)

Concentration of Ores by Wet Methods.—"The discovery by Everson in 1866, that metals and metallic substances in a finely divided state will combine with compounds of fats or oils and acids, and that divided quartz or other rocky gangue will not combine with such compounds, resulted in a concentration process in which the gangue could be separated from the treated pulverised ore by washing; and since that time methods have been devised for effecting the separation by mixing and washing out with oils alone, or by the use of water alone, the object being to entangle the metallic particles in surface layers of oil or air, so that they shall rise to the top of the water by ordinary flotation. In the Murex process the separation is effected by the action of a selective oily substance, with which is incorporated magnetite in a finely divided state. This mixture has the appearance of a black varnish, but it is heavier than water, and when added in a small quantity it rapidly forms a magnetisable film over the metalliferous particles of the crushed ore in water, whether those particles were previously magnetic or whether they were not. A magnet dipped into the water attracts the treated metalliferous particles at the bottom of the containing vessel of water to the rejection of the gangue. Flotation is thus replaced by magnetic separation; and the surface tension at the free surface, which is made use of in flotation processes, is here replaced by magnetic forces. To render the process continuous, the crushed ore, after agitation with the selective substance, in practice, is fed forward on a sloping table by a current of water, until it reaches the pole-face of a vertical magnet. An endless belt traverses the pole-face, preventing direct contact between the magnetised particles and the magnet, and carrying them beyond the field where they drop off, or are washed off, the belt."—*Times Engineering Supplement*, May 26, 1909. (J. A. W.)

Saving Fine Gold by Adhesion.—"There is perhaps no line of metallurgical work which has called out more effort in the improvement of methods than that of recovering flour or flake gold. In most cases it has been along the line of amalgamators, riffles, or the mechanical submersion of the containing sand in baths of mercury. The old copper plate has always held its place, and, although many devices have been brought out which promised a higher recovery, none has been able to supplant it in the general operation of gold milling. One of the great objections to the ordinary copper plate has been the loss of quicksilver resulting from scouring and fouling, for, while the mercury has a very decided affinity for copper, the attachment which has no side support is so delicate that it takes but little to disturb and detach it. The great advantages of copper plates are the ease with which they are kept in condition when once working normally, and also the fact that they are always open to close observation, and attention, in order to keep them in condition.

A new line of experiments (says *Mining Science*), has been recently taken up by J. D. Luttrell, of Benton Harbour, Mich. (U.S.A.), based on the idea of securing some medium or material which would more securely retain the mercury, and that, in such a form as not to entirely envelop the gold particle, but still hold it securely until it may be released as an individual particle and not as amalgam. Following this idea, he has produced a compound incorporating disseminated particles of mercury which are independent of one another. This compound, when made, is plastic, and can be used as a coating either on a plate, flat surface, or on the inner surface of a cylinder. On standing, the material hardens to any desired degree of density, according to the proportion of the ingredients, and as the mercury is evenly distributed through it, there are thousands of particles presenting bright faces at the surface, as attaching points for the gold. In that the materials of the compound, the nature of which has not been disclosed to the public, have a very high affinity for the mercury, it is quite difficult, if not impossible, to break them loose from their seating, as they are supported on all sides, thus reducing loss to a minimum. The surface of the compound, when ready for operation, has the texture and 'feel' of a moderately rough cement surface, and is in a most admirable condition for deterring the rapid passage of pulp over it. This roughness tends to roll the particles and keep them constantly climbing over obstacles. It has been demonstrated that when particles of gold become attached to the mercury particles they can be readily brushed off from the surface by passing a tuft of cotton batting over it, and the gold is recovered as dust. Based on this fact, the inventor proposes to use a large cylinder having a coating of the compound on the inside, as probably the best form of machine. This will be mounted on a slight pitch, similar to a trommel, and the sand, which will be fed at the upper end, will be gradually progressed to the lower end by the revolving of the cylinder. In this way every particle will be brought in contact with the surface. A brush of cotton, or similar material, is placed so as to be in contact with the surface at the highest point, and act as a detacher of the gold particles. This brush can then be burned and the gold recovered from it.

In a small demonstration, all the points mentioned above were clearly demonstrated, and the very finest of scale or flaked particles were seen to attach themselves to the mercury, but retained their individuality. A tuft of cotton then passed over the surface, cleaned it entirely, and entangled the gold completely in the fibre. A pan which was coated inside with the compound, entirely freed the sand of all the gold particles which were readily seen firmly held by the mercury, and the same results were obtained with dry or wet pulp. Water did not affect the action, appearance or condition of the surface further than to brighten the mercury particles, nor was there any change after hours of immersion. One feature of the method is the ease of renewal. If from any cause or accident any portion of the surface is injured, all that is necessary, in the way of repairs, is to apply a new coating of the compound, either to fill the break, or to cover the entire surface, if necessary, at but slight expense, and the apparatus is as good as ever.

The machines are designed to meet all requirements in the way of capacity, from 1,000 to 5,000 cub. yd. per day, and are made in such manner as to allow of transportation over the most difficult trails or roads."—*Australian Mining Standard*, May 19, 1909, p. 530. (A. R.)

New Nickel-Copper Alloy.—"Monel metal consists primarily of nickel and copper in the proportion of three parts of nickel to one part of copper, this being the natural proportion in which these metals exist in the largest deposits of nickel ore in the Sudbury district of Canada. Throughout its entire manufacture this alloy acts as a single metal, and while the proof is still incomplete, evidence seems to show that in these proportions nickel and copper have a remarkable affinity for each other.

Like steel, this alloy absorbs carbon, and, like steel, its physical properties are profoundly influenced not only by the percentages of carbon, silicon, etc., but by the heat treatment it receives. A standard analysis of Monel metal is as follows:—Nickel, 68 to 72%; iron, 0·5 to 1·5%: sulphur, 0·014%; carbon, 0·073 to 0·15%; copper to balance.

This alloy is silver-white and takes a brilliant finish, which it retains indefinitely. On prolonged exposure the surface assumes a greyish cast, which may be easily removed with a polishing cloth. In the rolled sheets, the surface assumes on heating a coating of oxide which has great resistance to acids. Samples of the rolled metal showed no loss in 56 days' test in pumping 40° sulphuric acid. A special pickle has to be used to remove the oxide coat for articles destined to be polished, but for purposes where resistance to acid is required, the oxide coating is allowed to remain.

Monel metal melts at 1,350° C. It can be rolled perfectly from 900° to 1,200° C., and its annealing temperature is above 875° C. It can be finished hard or soft, like sheet copper. Its specific gravity, as cast, is from 8·86 to 8·87, and when rolled from 8·94 to 8·95.

We have here a metal possessing about 25% greater tensile strength and 50% greater elastic limit than the best rolled steel, with the additional quality of incorrodibility. The mechanical possibilities open to such an alloy are endless.

During the last year some 300,000 sq. ft. of Monel metal sheets were used to roof the Pennsylvania terminal station in New York City. The manufacturers, who have tested this material for roofing, state that they find it as easily worked as copper. The sheets are silvery-white, exceedingly flexible and malleable, and are handled in exactly the same way as sheet copper.

The rolled metal has been drawn into wire of all sizes down to 0·004 in. This latter is as soft and pliable as a silk thread. In the form of wire exhaustive tests have been made of the metal for purposes where its incorrodibility makes it especially effective. In small articles, such as tacks, bolts, screws, angle braces, and such stamped work as door and window trimmings, hinges, gas and electric fittings, and other small articles, it is said to be rapidly finding its way into the market.

What is claimed to be one of the most satisfactory uses of Monel metal is in seamless tubes for condensers and boilers for automobiles and motor boats. The metal draws perfectly into a smooth tube, and its high elastic limit and tensile strength, combined with its incorrodibility, are of especial value in light machinery. The largest casting so far made was a solid hub and propeller for a steamer. This casting was 9 ft. 6 in. in diameter and weighed 6,500 lb. It has given perfect satisfaction. Larger castings can be made, and as the material is much stronger than manganese bronze or Tobin bronze, it is eminently suitable for any purpose to which these have heretofore been applied."—DAVID H. BROWNE, *Electrochemical and Metallurgical Industry.—Iron and Coal Trades Review*, March, 1909, p. 418. (A. R.)

ELECTROLYTIC PRECIPITATION.—"This method of recovering metals from cyanide solutions was proposed early in the history of gold cyaniding in Africa. Solutions from ores containing ½ oz. gold per ton will contain about 2½ dwt. per ton where 4 tons of solution are treated for every ton of ore crushed. This ratio of solution to ore is not an average, since Merrill precipitates from only 0·6 tons of solution per ton of slime treated, while at places in Nevada, with very high-grade ores, the ratio is eight to one. Taking the ratio of four to one as a basis for calculation, the lowest-grade solution may carry 0·25 dwt. per ton, and the precipitation from this grade is quite as important as the treatment of the higher grade solutions.

Recently Mr. Lay has given results of his tests and daily working results on precipitation, and places the ratio for practically complete precipitation of clear, gold solutions at 0·03 amp. per sq. ft. of cathode surface. The quantity of metal in such solutions is so small that the area of the cathode is by far the most important factor of the precipitation. He estimates the required cathode area as 1,750 sq. ft. per ton of solution per hour, and is using thin, lead-foil cathodes which remain in the boxes a month before being melted and cupelled.

The usual ratio of solution to ore in silver cyaniding is from 4 to 6 tons solution to a ton of ore. Then with an ore worth $6 per ton, such solutions would contain about 2½ oz. per ton, or 40 times the weight of metal in gold solutions of the same value. It will be seen at once that the problem of electrolytic precipitation in dealing with solutions from ores containing from 500 gm. to 1,000 gm. of silver per ton is much easier to solve, and is altogether different. By comparison, the solutions dealt with in this last case resemble the electroplater's bath.

The managers and foremen of the plants operated by Charles Butters & Co., Ltd., have improved on electrolytic precipitation until they are are now getting good results. Their object has always been to secure as complete a precipitation as is consistent with reasonable first cost and maintenance expense. Ordinary tinned iron plates are used as cathodes and peroxidised lead plates as anodes. Acheson graphite will probably replace lead anodes finally, since the latter are expensive and short-lived. A current density of 0·50 amp. is employed, which is sixteen times the density recommended by Mr. Lay. This heavy current deposits the metal loosely on the cathodes whence it is rubbed, and dropped to the bottom of the boxes.

The variable conditions of the boxes used by Mr. Hamilton at Minas Prietas in making his tests were: current density, rate of flow of solution, besides the constant slight fluctuations of alkali, cyanide and metal contents of the solution. The number of and distance separating the anodes remained constant. The accompanying table shows the conditions and results of the experiments made with solutions carrying 1·625 oz. gold and silver. With solutions containing more metal, the box will have greater capacity for the same per cent. precipitation. Experi-

Electrolytic Precipitation at Minas Prietas.

Tons Solution Passed Through Boxes per 24 Hours.	Cathode Area of Box, sq. ft.	Current Density amp.	Percentage Precipitated.
120	3,500	0·3	90·0
240	3,500	0·7	84·0
120	3,500	0·8	88·0
240	3,500	0·8	84·5

ments prove that with constant rate of flow of solution and current, the residue remains nearly the same, with varying grades of solution. Doubling cathode area increases percentage of precipitation more than does doubling current strength, but it also doubles attendance and first cost of plant.

My object in writing this is to urge that a judicious combination of electrolytic and zinc precipitation will have an advantage over either alone. At one plant with which I am familiar, 80 tons of 50 oz. silver ore will be treated daily, producing approximately 4,000 oz. of metal. At the lowest figure for zinc consumption this would require 270 lb. of zinc shavings per day. With an electrolytic plant precipitating 85% of the metal the zinc consumption will be reduced to 40 lb. with a very slight charge for power. In any case, the power cost is low. In this particular plant the labour required will be that of one man having extra help only at clean-up, who will attend to precipitation during the month. Then, too, in a closed-circuit plant, that is, one milling in solution and filtering, perfect precipitation is not required, since no loss is sustained if solution which is pumped to the battery supply contains more or less metal. Also in this class of plant the tonnage of solution to be precipitated is less, particularly where the entire tonnage is agitated.

The one advantage of purifying the solutions, which become so foul in treating the average silver-sulphide ore, warrants the use of a high-duty electrolytic-precipitation plant. Further, a recovery or regeneration of as much as ⅓ lb. of cyanide per ton of ore treated has been observed. The product of the electrolytic boxes, when compared to zinc precipitate, is of lower grade, if copper or other base metal is in solution, and is of higher grade with clean ores and rich solutions. Such zinc as remains in solution after passing through the batteries, concentrators, and treatment plant is found in the electrolytic precipitate, but is of negligible quantity."—MARK R. LAMB.—*Engineering and Mining Journal*, April 3, 1909, p. 705. (A. R.)

MINING.

STRAINS ON WINDING ROPES.—"Acceleration and retardation strains on winding-ropes were factors requiring very careful consideration, and, where the engine powers and speeds were very high, the danger might soon become a very serious one. Although the special arrangements for helping the acceleration and increasing natural retardation by the form of drum adopted might be very useful in regular work, he considered that their practical disadvantages, in adjustments or emergency working, often overbalanced their theoretical advantages; and, for his own part, he preferred the practically parallel drum with balance-rope arrangement, especially if the latter were so arranged as to overbalance some 10% to 20%.

What surprised him was that the idea that winding engines should have steam admitted for practically the full stroke was still so prevalent with both engine-builders and colliery engineers, when the results of such a practice had so often been shown to be both wasteful and inefficient; but a better mode of working was being surely, even if slowly, developed, and with the increasing depth of mines the problem would have to be fully considered, if successful work was to follow. Compression in the cylinder was not a loss of power, although it slightly reduced the useful effect; but it was necessary for good working with all engines running at moderate or quick speeds.

Clearance spaces were considered by many to be a total waste, but that was not the case except where no compression took place, and, with non-condensing engines working under good conditions, the loss was very considerably neutralised, although that was no reason for making them larger than was actually necessary for proper design."—B. WOODWORTH.—*Transactions of the Institution of Mining Engineers*, vol. xxxvii., part 1. (A. R.)

SHRINKAGE STOPING IN WESTERN AUSTRALIA.—"At the meeting of the Institution of Mining and Metallurgy held recently, a paper by Mr. F. Percy Rolfe described the method of stoping, called 'shrinkage' stoping, used at the Lake View Consols Gold Mine. This system consists in allowing the ore broken in a stope to lie there and accumulate, thus acting as filling material. On account of ore in the broken condition occupying, approximately, 50% more space than when *in situ*, it is necessary to draw off about one-third of the ore stoped so that the level of the broken rock shall always be at a workable distance from the back of the stope. After the whole block has been worked out, the mass of broken ore is completely withdrawn, and the resulting empty stope is either abandoned or filled up with mill residues or waste rock. It goes without saying that these depleted workings are simply abandoned in the Lake View Consols. In starting to work a block of ore on the 'shrinkage' system it is usual to take off a first, or 'leading' stope of about 12 ft. in height. In order that the rock drill shall have access to all parts of this high face it is necessary to effect two horizontal riggings of the machine bar, the latter position vertically below the former. A leading stope of this size thus gives ample room for putting in the timbers, which are placed higher than is the general custom on account of the space demanded underneath by the 'chinaman' chutes. Two stylse of timbering have been employed to support shrinkage stopes on the Lake View, viz., 'saddleback' and the ordinary single stull. There is, however, only one instance of the former variety, it having been entirely superseded by the more economical stull-timbering. The 'saddleback' is made use of in a big ore-body, where the drive is stripped to the full width of the lode, meaning that the level is thus too wide to be spanned with a single stull. This kind of timbering may be described as analogous to the construction of the ordinary house roof, in which the rafters correspond to the timbers of the 'saddleback.'"—*The Science and Art of Mining*, March 6, 1909, p. 337. (A. R.)

DEEP MINING.—"There is not so much fear of a vein or its values playing out in depth as of the lodes passing out of the property into another man's ground. Increase in heat is not an obstruction in going to great depths. The increase has been in the past somewhat overrated, except in such mines as the Comstock, which is following down a retreating geyser. In the Mysore mine of Kolar, India, 4,000 ft. deep, the increase is said to be only 1° to every 1,000 ft. Several of the Rand mines are working at depths of 4,000 ft., and it is proposed to go even to 5,000 ft. To go to great depths in a mine does not involve a consideration of increasing heat, but matters of ventilation and hauling, which, with modern appliances, are not very serious difficulties."—*Mining Science*, April 8, 1909, p. 272. (A. R.)

GOLD IN REEFS.—"Let us take up the history of a lode of gold at the moment when, after its original deposit, its highest part is for the first time brought by the progress of erosion into contact with the superficial stream of water. The property of such streams, which are constantly recharged by their very circulation with oxygen, carbonic acid and other mineral principles common to surface waters, such as traces of chlorides, nitrates, etc., is to exert an oxidizing and dissolvent action upon the sulphuretted ores. Applied to auriferous pyrite this reaction produces sulphate of iron and soluble gold salts. The sulphate of iron in the topmost parts of the deposit is immediately precipitated under an oxidized form and constitutes what is called the ferruginous 'cap' of the lodes. The dissolved gold may be partially drawn out and lost (thus going to enrich the placers, of which I shall speak further on); but a large part descends with the waters along the lode and is precipitated on its deepest parts, enriching them under the form of free or native gold deposited in all their fissures.[*]

Let us now imagine such a reaction continuing during entire geological periods upon deeper and deeper parts of the lode, which are successively brought to light by the progress of erosion.[†] It is easy to conceive how, besides a portion of the gold drawn chemically or mechanically away and destined to form alluvia, growing quantities of gold have been accumulated in the remaining altered region, especially in its lowest part, near the hydrostatic level, where the reprecipitation chiefly occurs. The conclusion is, therefore, that an altered deposit presents when completely filled, going from top to bottom: (1) a cap of hydrated sesquioxide of iron often rich in native gold; this is called the region of peroxidation; after which comes (2) what is called the region of cementation, or of 'bonanzas,' that is to say, exceptional productivity, in the language of American miners. There with a residue of untouched sulphides, ores of native gold exist which may be very rich locally, following the chimneys, or 'shoots,' along which the circulation of water chiefly went, but which are always very irregular and certain to disappear at some depth. Finally (3) when we penetrate sufficiently deep, we almost always find the original pyritous type, in which the gold is in a state of mixture or more or less complex combination with various metallic sulphides, and which keeps the same aspect to the end of the deposit.

It is the enriched ores of the two first regions, but chiefly the region of cementation, which furnish the beautiful fragments of milky-white quartz through which gold seems to ooze or grow like a plant out of all the fissures, and which the English compare to a 'jeweller's shop.' Most extraordinary proportions of gold may be found, up to blocks of native gold; but this is not a type of deposit on which we can count for the future, and as it is a superficial class of ores, without deep-seated continuation and easy to recognise, the deposits of this kind decrease every day in importance, and are doomed in a not far-off period to disappear completely. Despite the fascination which they exercise, it is not these, but the poor and regular ores of the deeper strata which give the large returns. Roughly speaking, the lower limit of these 'bonanzas' would seem to be at the hydrostatic level, which may be complex and compatible with the reappearance of oxidized parts below unaltered parts[*]; but, as the American geologists, Mr. Weed, Mr. Emmons, etc., have shown by numerous and interesting observations the reprecipitation of gold, the cementation, and therefore the bonanza, can descend far below the hydrostatic level, so long as the circulation of water is continued there. These geologists have also demonstrated that other metallic sulphides, galena for example, could not have taken the place of pyrite for the reprecipitation of gold, so that in deposits of galena without pyrite there is no bonanza.

Most mines have seen this phase of superficial enrichment in a more or less marked form, according as the alteration of the pyrite has been more or less complete. On the Rand the miners began by finding disaggregated conglomerates at a depth of forty-five to sixty feet, the pebbles of which, fixed in their beds, were often covered with an auriferous coating. High proportions of gold were then obtained which have disappeared very rapidly, and, if at the pyschological moment the process of cyaniding had not been discovered, which has been applied with marvellous success to the deep-seated pyritous ores, the industry of the Transvaal, on the disappearance of the ores, of native gold, might have undergone a grave crisis. In the famous Californian lodes, at depths of 120 to 180 feet, proportions of free gold have been found in the 'iron cap' which have frequently reached 4 to 5 ozs. Lower down the proportion fell, and quartz with auriferous pyrites and sulphides of all kinds was found; the proportion of gold in the ores fell to 225 to 300 grains, while the proportion of gold in mechanically prepared sulphides was from $3\frac{1}{2}$ to $4\frac{1}{2}$ ozs. Finally in Australia, in the equally celebrated Mount Morgan (Victoria), altered and irregular ores, with silicified peroxide of iron were found as far as 270 feet down; the proportion of gold increased from the surface towards the region of cementation where it reached $3\frac{1}{2}$ ozs.; lower down, quartz with auriferous pyrite was found, the proportion of gold in which soon descended to 1 oz. and lower."—L. DE LAUNAY (1908), p. 57 *et seq.*—*The World's Gold*, June 21, 1909. (W. A. C.)

VALUATION OF MINING AREAS ON THE RAND.—"During recent years it has become the practice on the Rand, when forming new mining companies, to include a much larger area of ground than was the custom formerly, while several mines already formed into companies have secured larger areas by amalgamation with neighbouring properties.

In laying out the first row of mines below the outcrop mines, the area selected used to be about 200 claims (294 acres) for each mine with a milling plant of 200 stamps, the depth of the workings being from 1,000 to 3,000 ft. The standard adopted was a stamp a claim, the stamp being equivalent to an output of five tons per day. This area is now considered too small, and companies with 1,000 claims and upwards have been formed.

These changes in the areas selected make it of interest to consider the factors that decide the size of a mine and the principles upon which valuations should be founded.

As regards the question of size, it may be stated generally that the larger the area the less will be the capital expenditure on equipment and the less the working costs per ton. It is obvious that a large area lends itself to more economical development than a small one. Administration and general expenses, being spread over a larger tonnage, are proportionately reduced, while a great saving in capital expenditure is obtained by concentrating the reduction and power plant instead of having a number of small units.

---

[*] The dissolution of gold is, contrary to the general idea, a very ordinary phenomenon, and examples are known (Taupo in New Zealand and Steamboat Springs in California) of hot springs depositing gold and silica in the neighbourhood of the surface.

[†] Faults play a very important part in these introductions of superficial waters. On the occasion of a recent work, La Metallogénie Italienne (The Geological Congress of Mexico), I analysed several cases of this kind. We must take account also of ascending or thermal springs.

[*] I do not here speak of movements of the soil. I have long ago called attention to the probable part played by them in certain alterations which go very low, and Mr. Weed has given examples at Butte City (Genesis of Ore Deposits, 2nd Edition, p. 497).

The statistics of the Rand mining companies illustrate very forcibly the advantages of large units. The writer has compiled from the August, 1908, sheet of the Transvaal Chamber of Mines the following table, which shows the effect of large outputs as compared with small. The figures relate to 53 out of 63 mines in the Witwatersrand district. The 10 mines omitted are either small producers or mines which do not report working costs.

Results of 53 Milling Companies on the Rand, August, 1908.—

Tonnage treated.	Companies.	Yield per ton milled.	Costs per ton milled.	Profit per ton milled.
Tons.		s.	s.	s.
Under 10,000 ...	10	33·4	23·4	10·0
10,000 to 20,000 ...	13	29·6	20·0	9 6
20,000 to 30,000 ...	13	28·8	17·0	11·8
30,000 to 40,000 ...	10	31·0	17·0	14·0
40,000 and upwards	7*	32 6	14·8	17·8

With regard to this table, it is advisable to mention, in order to prevent any misunderstanding, that the costs and profits shown are what are known as costs and profits on working account. Profit tax, depreciation and sundry expenditure are not included. There is generally a difference of about 30% between the working profits shown and the profits distributed as dividends, that is, the dividends are 70% of the working profits.

The effect, therefore, of working on a large scale is to reduce working costs, and it is clearly an advantage to have a large area and to make the output as large as possible, paying due regard to capital expenditure and life.

In calculating the most suitable rate of working for any given area it is necessary, in the first place, to make an estimate of the probable tonnage and of the value of the ore. The Rand ore deposits are of such a character that fairly reliable estimates can be made, the reliability depending on the situation, size, amount of development and records of neighbouring mines. Granting that calculations of this sort have been made, and that definite figures have been arrived at as to the amount of gold that the mine contains, the problem that then needs solution is the rate at which it may be most profitably extracted. The elements of the calculation are:

1. Capital expenditure required for a given production.
2. The available tonnage and its value.
3. The cost of working.
4. The rate of interest required.

In mine valuations of this class it is 'time' that is the important factor. As long as the gold is left in the ground, there is a loss of interest going on and, after a certain period, this loss of interest will exceed the cost of new shafts and equipment. An example will make this point clear. Suppose a property contains 1,000 claims and a tonnage of 20,000,000 tons estimated to give a net profit of 10s. per ton, and suppose the equipment is equivalent to an output of 400,000 tons per annum; the life would be 50 years, and the annual income £200,000.

The present value of an income or annuity of £200,000 for 50 years, assuming that 6% interest is demanded and that a sinking fund to redeem capital was invested at 3%, would be worth 14·5 years' purchase, or £2,900,000. If, however, the ground had been divided into two mines, or if the output had been doubled by increasing the reduction plant, the annual profits would have been £400,000 for 25 years. The present value of this income is worth 11·5 years' purchase, or £4,600,000. There is, therefore, an increased value by doubling the rate of production of £1,700,000, which would far exceed the capital expenditure necessary to secure this increased rate.

This example indicates that in order to obtain the greatest value from any mining area containing definite values, as on the Rand, it is possible to calculate with some precision the most profitable rate of working.

The formula necessary for calculating the most profitable life from a theoretical point of view have been give by Hellman, Browne, Kotzé and others.

According to Mr. Kotzé, the most suitable life for a mine giving a 20s. profit per ton is about 11 years, and that for a mine with ore yielding a profit of 10s. a ton, 16 years. The definition that he gives of the most profitable life is that at which the ratio of increase of present value of profits to increase of working capital becomes unity. This would be the most profitable life theoretically. In practice, however, the life should be extended somewhat, and the ratio taken higher than unity. Mr. Kotzé also comes to the conclusion that the larger the profit per ton, the shorter would be the more profitable life if other conditions remained the same.

No doubt the calculations given in his paper would have to be modified to suit present conditions, the capital expenditure required for a given production being now less than at the time the paper was written. But as the effect of a decrease in the capital expenditure for a given output would be to decrease the theoretically most profitable life, the ideal life, on a 10s. per ton profit basis, may be taken to be under 16 years.

Mr. Browne, who has also made calculations as to what is the most profitable life, arrives at the conclusion that 10 years is the ideal life.

These calculations go to show that the life must be kept within bounds to obtain the greatest value out of a given area. The author wishes to emphasise this point, because in recent amalgamations a long length of life has been put forward as a favourable feature.

It is no doubt a good thing for a mine to be worked on a large scale, but it does not follow that its life should extend beyond certain limits, which, as shown above, can be calculated.

The author does not wish it to be understood that he advocates the working of a mine strictly on the theoretical standard. In practice, it would, in his opinion, be unwise to go to the extreme limit. Even on the Rand there must be uncertainty as to the total tonnage and values available, and the scale of operations will also be affected by financial and labour considerations, factors which are variable and uncertain.

The safest course in practice would be to lay out the mine at the start for a production below the ideal production, but the aim should be to work ultimately to the scale of operations that calculations of total profits and total expenditure indicate as the most profitable.

---

*Included in this class is the East Rand Proprietary Co., which is rather a group of mines than one unit. If the figures of this company are omitted the averages for mines milling over 40,000 tons per month would be six companies, yield 33·4s., costs 13·8s., profit 19·6s.

In coming to the decision as to what is the best scale of operations, an important point to be considered is that of the most profitable grade to work to. The ore deposits of the Rand are such that the engineer can vary his grade within wide limits. He can increase his tonnage and lower his costs by including low-grade ore. Low costs and large tonnages do not, however, necessarily mean that the mine is being worked to the greatest profit.

To determine the most profitable grade it is necessary to calculate the present value of the total profits expected.

To illustrate this, the author will suppose that a mine has 1,000,000 tons which will give a profit of 20s. per ton, and that, by leaving some of the poorer ore in the mine, a production of 800,000 tons, giving a profit of 25s. per ton, can be obtained.

The question arises which is the more profitable grade to work, the high-grade and the smaller tonnage, or the low-grade and the larger tonnage. Assuming that the reduction plant will treat 200,000 tons per annum, the life will be either five or four years and the annual profits either £200,000 or £250,000. Calculating the present value of these profits on the annuity principle and assuming that 6% interest is required and that a sinking fund for the redemption of capital is invested at 3%, the low-grade policy will show a present value of £806,000 and the high-grade £835.000. Consequently the high-grade is the more profitable to work.

It will be observed that in all calculations as regards life or grade one of the most important factors is time; and it is this factor that carries great weight in valuations of deep level claims, which may be proposed for amalgamation with an adjoining property.

Some examples of the sort of valuation required may now be considered. Taking a simple case, that of the amalgamation of two undeveloped areas, each too small to be worked separately, but which together would give an area of suitable size for a mine, the valuation of the ground would depend on the present value of the profits to be derived on an assumed rate of working. It is assumed that the profit to be derived from the deepest ground would be less than that from the more shallow, partly on account of the profit being deferred and partly due to the assumption that working costs increase and consequently profits decrease with the depth of working. This is what is known on the Rand as the zone basis of valuation.

For convenience the ground is divided up into belts or zones parallel with the strike, and a value is calculated for each zone. The author believes in this system of valuation provided it is not carried to extremes. If carried too far, the value of the deeper claims is unduly depreciated. The profits are discounted over too long a period, and a fairer way to value such claims would be to assume that they could be made productive earlier—which might be the case if a new shaft were put down—than if they had to wait for the exhaustion or all the ground above them. The relational value of claims has received considerable attention in South Africa, and those who wish to study this subject are recommended to read the paper by G. A. Denny, read before the S. A. Association of Engineers, and the discussion that followed.

Another case that may be considered is that of the amalgamation of deep level claims with those of a producing and profit earning mine. In this case the value of the deep claims to the producing mine will depend on the amount of tonnage still available in its own property—in other words on its life—unless the development of the deep ground can be taken in hand before the exhaustion of the upper ground.

The value of the deep level claims to the producing mine is the present value of the estimated profits at the time when the upper claims are exhausted. If the producing mine has still a long life before it, the value of the deep claims will be small, because the profits to be derived from the deep claims will be so long deferred.

It must also be borne in mind, when valuing claims, that there is a liability attached to them in the way of claim licence, amounting to £3 per annum. The capitalised value of the claim licences must be deducted from the capitalised value of the profits expected in the future. The factor, time, here again exercises an important influence, and if the realisation of the profits is to be long postponed, the value of the claims will be small and may in fact, be a minus quantity.

A third and somewhat more complicated case may now be considered.

As a rule, when an amalgamation of a producing mine with deep level claims is effected, an increase in production is arranged for, and the owners of the deep claims frequently contribute capital for this purpose. In making a valuation of the different interests in such a case, it would be necessary to calculate the present values of the estimated profits to be derived from different claims put in, and also to assess separately the value of the other assets, such as equipment, development and cash. Both sides benefit by amalgamation, owing to the speedier realisation of the profits, and the case differs from that previously given in that the producing mine has here to consider the value of the extra profit that will accrue owing to the increased scale of operations which the capital of the deep block allows of, whereas, where no increase of production takes place, the producing mine has merely to decide whether the price put upon the present profits of the deep ground is sufficiently attractive.

There is one other point of great importance relating to mine valution to which the author wishes to draw attention. It is well known that on the Rand there are rich reefs and poor reefs, as well as great variations in values in the reefs themselves. It is usual to work out the ore at an average grade, taking each reef out in its proper proportion. This system is financially unsound.

Theoretically, to get the most value out of a mine, the grade in the early part of the mine's life should be higher than in the later years. In practice it is of course not possible to mine all the rich ore first and then return to the poor. Stopes or levels cannot be kept open indefinitely without great expense. At the same time the author thinks that some selection is possible, and that if the richest reefs and the richest put up sections of the poorer reefs were attacked first, a higher grade than the average could be obtained during the early years of the mine's life. A considerable appreciation in value would result from the adoption of this policy."—W. Fischer Wilkinson, *Institution of Mining and Metallurgy.—The Mining World*, May 15, 1909, p. 927. (J. Y.)

Miners' Electric Lamps.—"In a paper read before the Midland Counties Institution of Engineers, Mr. W. H. Mills summoned up the advantages of miners' electric lamps over oil lamps as follows:—Dr. Court found that of 500 men working with oil safety lamps, over 100 were suffering from nystagmus, and that only three men out of 500 were affected where naked lights were used, showing that constant straining of the eyes owing to deficiency of light is the cause of the disease. The use of electric lamps, with the increased illuminating power, will undoubtedly reduce the number suffering from this serious malady. For examining the roof in the working places and roadways, the electric lamp has the advantage of the light not being affected by being tilted in any direction, and there is no doubt that with this advantage and the better light accidents from falls of stones will be reduced when electric lamps can be used. The electric lamp is a necessity for use with life-saving apparatus, but the lamps now in use are not altogether satisfactory. At the present time there is a serious loss of working time caused by the oil lamp being so easily extinguished, and with the shortened hours of labour this loss of time will be still more serious. Electric lamps have not this disadvantage. It must also be admitted that even with the ingenuity expended on the manufacture of the safety lamps, and with the great care exercised in cleaning, locking, and in the use of the lamp, there is still an uncomfortable feeling that a defective lamp, or the wrong use of a lamp, may be the cause of a serious explosion of fire-damp and coal dust; and now that such high velocities of air are usual, and most of our deep mines are dry and dusty, a load of anxiety will be removed from the over-taxed colliery manager if a still safer lamp than the oil lamp can be found. It, therefore, behoves everyone who is interested in coal mining to do his utmost to improve the miners' safety lamp."—*Science and Art of Mining*, May 15, 1909, p. 458. (A. R.)

HAULAGE IN COLLIERIES.—"Notwithstanding the increased popularity of electric haulage, the endless rope system remains in favour at many collieries where special conditions exist. The best features of this latter system of haulage are: (1) It is capable of dealing with large outputs in an easy form; (2) there is less wear and tear of rolling stock than with other systems; (3) it travels at a much lower speed than other systems for a given output; and (4) it is, as a result of the foregoing advantages, less liable to breakage and accidents to persons and animals. In determining the size and weight of the rope in an endless-rope system of haulage, an ample margin should be allowed for safety, considering at the same time unusual strains which may come upon the rope by overloading. The net strain on the rope at the driving pulley rim may be taken as the net load or resistance to be overcome by the engine. In practice, it is usual to allow from 25% to 50% for friction in engine gear and margin for undue requirements."—*Science and Art of Mining*, May 29, 1909, p. 481. (A. R.)

MINER'S LAMP TESTS.—"Recent tests show that an ordinary miner's lamp (of the type generally worn on the miner's cap) when burning tallow, consumes 2·49 cub. ft. of oxygen and produces 1·74 cub. ft. of carbonic acid per hour. The same lamp when burning oil consumes 1·13 cub. ft. of oxygen and produces 0·78 cub. ft. of carbonic acid per hour. The same investigators determined the candlepower of various lamps by photometry as follows:—Wolf safety-lamp (having a wider flame than most safety lamps and a white enamelled reflector), burning with a flame 9 in. high, gave 8 candlepower. Scottish miners' oil lamp (consuming 13·8 gm. of oil per hour), average during 15 minutes with uniform flame, 1 candlepower. The kind of oil used affects the candlepower considerably. Scottish miners' tallow lamp (consuming 17·4 gm. tallow per hour), average during 15 minutes with uniform flame, 2·3 candlepower. Scottish miners' lamp (consuming 13 gm. paraffin wax per hour), gave 1·6 candlepower. Scottish miners' lamp (consuming 19·3 gm. paraffin wax per hour) gave 2·6 candlepower."—*Science and Art of Mining*, May 29, 1909, p. 482. (A. R.)

ELECTRIC HAULAGE AND WINDING PLANTS.—"A very great deal has been done in recent years in the application of electricity to haulage underground, and the plants now being installed are of much greater power than hitherto. A few years ago plants of 25 h.p. to 50 h.p. were considered important, whereas, at the present time, plants of from 200 h.p. to 300 h.p. are by no means uncommon, and some are at work up to 500 h.p., or even larger. In considering the adoption of electric haulage one has to be guided by local circumstances in selecting the most suitable of the various systems, which may be divided as follows:—(1) Main rope haulage, consisting of a single drum, and hauling up gradients, exceeding, say, 3 in. per yd., so that the empty tubs are able to draw the rope back by gravity; (2) main and tail haulage, in which the main rope usually draws the full tubs up an incline or along a road with a gradient in favour of the load of less than 3 in. per yd., the tail rope being used for taking back the empties or the lighter load; (3) endless rope haulage, which can be used on roads with a gradient in favour of the load or against the load, but this system can only be used where a road is wide enough to allow the full tubs to pass the empty tubs, unless a special by-pass arrangement is used, which is not generally satisfactory; (4) by electric locomotives, these are generally used on nearly level road, and for surface work. There are various modifications of these methods of working and combinations for special cases, but the above generally represent the main four systems on which haulage in collieries is effected. Main rope haulage requires only one line of rails and is adopted on single roads where there is not sufficient width to use an endless rope, and where the tubs are of sufficient weight to run back by gravity. It will be readily understood that it is impossible to fix any definite gradient upon which the haulage will self-act, *i.e.*, run back by gravity, as this depends so very greatly upon the condition of road, namely, upon whether the rails are of sufficient weight and are well laid, or whether—as is often the case—they are light and badly laid. In addition to this, the class of tub wheel has a very great deal to do with easy working. With heavy tubs fitted with large wheels, such as are used in South Wales, the train will self-act, *i.e.*, run back, on a gradient of even less than 1 in. per yd. On a fairly good road, and with the tub wheels of reasonable size, it is found that 2 in. to the yd. is sufficient for self-acting, but this is working rather close to the line, and therefore, a gradient exceeding 2 in. or 3 in. per yd. is one upon which a main rope haulage is usually applied. Main and tail haulage is largely used in the north of England and also in South Wales, and in districts where, owing to the possible rising of the ground or the difficulty in maintaining a good roof, it is found impossible to make the roads of sufficient width to allow endless rope haulage to be used. This system is being gradually discarded for endless rope, but there are still a good many mining engineers who believe in it; this is generally in older collieries or mines where the roads are of considerable length, and where it is necessary to get the men inbye, *i.e.*, to their work, in the shortest space of time.

It will be easily recognised that in any system of intermittent haulage, such as main rope, in which the full load is usually drawn up and the empty load allowed to self-act back, or in main and tail haulage, where the full tubs are drawn out-bye and then the empty tubs taken in-bye, a very considerable amount of time is wasted in making up the train of tubs, changing from the full to the empty in-bye, and also going through the same operation at the pit bottom, or to whatever point the haulage gear may draw the coal or minerals. Approximately, a gear of this type is worked at full power for only one-third of the time; it then stands for about one-third, and runs at a much smaller power for the other third. Consequently, the amount of power necessary for dealing with a given output with either main or main and tail haulage is very much greater than with the endless rope.

Perhaps the most important application for which electricity should be employed, or in which there is the greatest use for large power electrical machines, is that of main winding. It is a subject which has had a great deal of consideration. There are many opinions as to whether the use of electricity is an advantage over steam winding or otherwise. There is much to be said in favour of both, and conditions and circumstances are such important factors in each individual case, that it becomes impossible to make a general statement, or for anyone to express a decided opinion applicable to all requirements. It would seem that where it is necessary to lay down a complete electrical generating plant for the electric supply and also the motors and other machines for conveying power to the winder, it is questionable, taking the cost of the plant and the losses in conversion into consideration, whether the advantages are sufficient to make the scheme practical and economical; but, on the other hand, where there exists a suitable electric supply, and there is a margin or provision to prevent entire stoppage in the event of the failure of supply, winding by electricity would appear to be worthy of most careful consideration, especially as the apparatus needful, as now manufactured and and supplied by those firms who have given the matter attention, is in every way suitable and reliable. In this country there are comparatively few instances of electric winding plants at work. It seems to have been left to our Continental friends to take the initiative. They have some very fine examples at work, amongst which the most important plants are those at Preussen II. Colliery, near Dortmund, Germany, and Grand Hornu Colliery, Belgium—both by the Allgemeine Elektricitats Gesellschaft, and Zollern II. Colliery, Merklinde, Germany; the electrical machinery by Messrs. Siemens and Halske, Berlin. A very important and interesting electric winding plant on a large scale, which compares very favourably with any of the Continental examples, is that erected during last year at the Maritime Pit of the Great Western Colliery Company. This installation, as well as being a pioneer equipment, has many special features, described in papers read by Mr. Hugh Bramwell and Mr. Gerald Hooghwinkel before the South Wales Institution of Engineers. At the Brook pit, Heckmondwike, an electric winding plant was installed for special purposes. A large fault or throw occurs about 3,000 ft. away from the pit bottom. This fault throws down the coal seam a vertical depth of 180 ft. below its original level. To win the coal-field on the dip side of the fault, it was necessary to drive either a large drift or sink a staple. If it had been decided to drive a drift it would have been necessary to put down a powerful haulage plant, at least equal to the size of the present winding plant. The driving of such a drift with a gradient of 1 in 6 would have involved 1,080 ft. of drifting, and the cost would have been about £1,960. The sinking of the staple cost about £3 10s. per ft. for 180 ft., or a net saving of £1,360 against the drift scheme. It also took nine months less time to win the coal by the staple than it would have done by the drift. The shaft is 180 ft. deep, and with the present arrangements the winding gear is capable of raising from 150 tons to 200 tons per day of 10 hours. The winding drum is 3 ft. in diameter, and a wind is made in 40 secs. to 45 secs., equal to a speed of, say, 300 ft. per min. The electric motor driving the drum is of the series wound continuous-current type, capable of working up to 40 effective h.p. when running at a speed of 500 revolutions per minute. The motor is fitted with slotted drum armature with specially large commutator, fitted with carbon block brushes, self-oiling bearings, and is of the usual type that the makers have designed for this class of work. A machine-cut pinion of forged steel is fitted on the end of the spindle and gears into a spur wheel on the counter-shaft. The first motion gearing is machine-cut forged steel, and the second motion gearing is machine-moulded cast-iron. A powerful foot-brake is provided for the drum operated by a foot lever. as a further protection, this gear is fitted with an electric brake very similar to those on the German gears already described. This brake is fitted on the motor shaft and comes into operation as soon as the current is switched off, and thereby sustains the load. An indicator is also fitted on the winding gear to show the position of the cage in the shaft. This winder is operated by means of a liquid controlling switch arranged for reversing and speed regulation. The plates are arranged to dip into the liquid in an opposite manner to those of the gear at Preussen II. Colliery, in which the liquid rises on the plates. This winder could have been fitted with a three-phase motor of the same power, in which case a similar type of liquid starter would have been employed."—M. B. Mountain, *Manchester Geological and Mining Society.*—*The Science and Art of Mining*, May 29, 1909, p. 488. (A. R.)

Coal Dust Explosions.—"Recent experiments show that ⅓ oz. of coal dust per 35 cub. ft. of air will suffice for the production of an explosion. It was also found that it is more a question of fineness of the division of the dust, than the weight, that causes a condition of great danger."—*Mining Science*, March 25, 1909, p. 227. (A. R.)

Silicified Timber.—"The extraordinary power of hot silica bearing water to penetrate and microscopically replace certain substances with the utmost minuteness and faithfulness is shown in some of the deep and old abandoned workings of the Comstock mine, where not only have the timbers been partially silicified by the hot silicious waters, but even the fine, soft, impalpable fungus growing on those timbers, well known to miners in the timbering of damp portions of mines, is replaced by opaline silica down to the minutest fibre and showing the delicate fernlike patterns of this parasitic growth."—*Mining Science*, March 25, 1909, p. 223. (A. R.)

MISCELLANEOUS.

The Use of Concrete for Mine Support.—"The author said that up to comparatively recent times timber had been the principal means of support for underground working; but its use as an economical material for support was rapidly passing. The falling off in the use of timber had been augmented by the application of concrete, which was now recognised as a reliable and satisfactory substitute for timber. Concrete and reinforced concrete were now extensively employed in mining shafts, mine galleries, tunnels, gangways, in the support of rooms and stations, in building dams and stoppings underground, and in fact, in all mining work both within and without the mine. Concrete had the advantage that it could be applied equally well in shafts of any shape and size. Several rectangular concrete shaft-linings had been successfully constructed in the United States, but practice favoured the oval and elliptical forms. Only about half the thickness of lining was necessary when curvilinear forms were employed. Many of the deep-level shafts of the Lake Superior copper region were lined to the bed-rock with concrete. Instances could be quoted of the use of a circular concrete drop-shaft in sinking through loose materials. The relining of timbered shafts with reinforced concrete had been successfully accomplished in a number of instances in Pennsylvania, one of the best illustrations of such work being that of Manville shaft, in the vicinity of Scranton. Stations for underground machinery such as pumps, engines, stables, offices, etc., were now, in many large mines, being lined or re-lined with concrete. Concrete in coal-mine workings had an advantage over timber and stone in that with a little care in surfacing, the resistance to air-currents was materially reduced, which circumstance, together with the considerations of greater strength and durability, was rapidly extending its use."—Prof. W. R. Crane.—*Times Engineering Supplement*, June 2, 1909. (J. A. W.)

New Concrete will not retain oil-paint because of dampness, and the presence of caustic lime. Time and nature will remove these objections, the carbonic acid of the air neutralising the lime, but we shall hardly be allowed to wait for it. Acid will certainly do away with the caustic lime, but the remedy is about as bad as the disease, the salts of lime which are formed being either highly hygroscopic, like calcium chloride ($CaCl_2$), or the action being too superficial. After exhaustive trials Fred. J. Bosse has worked out the following method as the only one which will satisfy the needs for a paint on concrete: Dissolve 10 lb. carbonate of ammonia in 45 gallons water and apply with a brush; or make the solution weaker and give several coats, or apply as a spray. The following reaction takes place:

$$CaO + (NH_4)_2CO_3 = CaCO_3 + H_2O + 2NH_3.$$

The ammonia which is liberated will evaporate, and the calcium carbonate which has been formed is perfectly insoluble and will become dry and hard in a short time. On the surface thus prepared ordinary oil paint may be applied. This process will be found convenient, cheap, and efficient."—*Mining and Scientific Press*, March 6, 1909, p. 359. (K. L. G.)

Electricity in Coal Mines.—"The main object of this paper is to consider the practical application of electricity to coal mining, regarded purely from a safety point of view. No system of power transmission was altogether free from danger, but even under the severe conditions which obtained underground, mine-owners might confidently avail themselves, over a very wide field, of the advantages which electricity offered as a power agent, provided that due care were given, first to the choice, and secondly to the installation of the apparatus. The dangers of electricity in its application to coal-mining work underground were two in number—namely, danger of shock to persons owing to contact with 'live' metal, and danger to persons and property through the ignition of explosive gas, coal-dust, or other combustible material by an electric spark. The opinion was frequently expressed during the departmental committee proceedings on the use of electricity in mines, that the chief danger to be apprehended in working would arise from the cables in a mine rather than from the motors and apparatus. There was much to be said in support of this view. The transmission system in shafts and roadways should be well insulated, and wherever there was reasonable ground, following a chemical investigation, for assuming that the water in the pit would not attack lead, a lead-sheathed and armoured cable was to be recommended. He would draw particular attention to the advantages which should follow the employment in mines of the Merz-Price protective gear. There was one special use of electricity underground which really deserved much more than passing consideration, and that was its use in driving coal-cutting machinery. In this connection attention might be directed to a gate-end switch for coal-cutters, designed by Mr. H. J. Fisher of the Lambton collieries. From the point of view of safety, the switch represented a very considerable advance on older methods, and it was applicable either to a three-phase or to a direct-current system. In connection with switch-gear the three-phase system enabled oil-immersed switches, controllers, and even resistances to be used when otherwise the circuit would require to be broken in air—an advantage almost sufficient, if none other existed, to justify the preference. In recent joint experiments of the German mine-owners and electrical engineers, the conclusion reached was that "plate" protection afforded sound mechanical protection together with what was lacking in the case of totally enclosed motors, namely, ventilation. Electric lighting underground was practically confined to places near the shaft-bottom and to motor-rooms separately ventilated by intake air, and there could be no doubt that it should be so confined."—Robert Nelson, Annual Meeting, Institution of Mining Engineers, May 27.—*Times Engineering Supplement*, June 2, 1909. (J. A. W.)

Cyanide Poisoning.—"Risk arises, in the first place, from the fact that cyanide solutions look exactly like water. The addition of a small amount of a suitable colouring matter would be the best precaution, were it always practicable, but it could be insisted on in a laboratory. Failing such obvious methods of prevention, it is surely desirable that means of cure should be kept at hand wherever an accident is possible; and, as this can be very simply done, it appears to me that proprietors of works should be compelled by law to do it. Seven years ago the Victorian Government evidently contemplated some action of the sort, for they commissioned Dr. C. J. Martin, then professor of physiology in the Melbourne University, to investigate and report upon the antidotes for cyanide poisoning. Their experiments led them to reject certain reputed antidotes on the score of inefficiency or danger, and to strongly recommend the following mixture: 1 oz. of 23% solution of ferrous sulphate, 1 oz. of 5% solution of caustic potash, 30 gr. of powdered magnesium oxide. There is every reason to believe that this mixture will prove an effective antidote if administered at once, but a very few minutes' delay must prove fatal. Drs. Martin and O'Brien therefore recommended that the materials be kept in sealed tubes which can be broken at a moment's notice, the tubes and a suitable mixing vessel to be conspicuously placed wherever they may possibly be wanted."—*Science and Art of Mining*, May 15, 1909, p. 459. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE DRESSING OF MINERALS. By HENRY LOUIS, M.A., D.Sc., A.R.S.M. (London: Edward Arnold.)

"The author deals first with the subject of volumetric sizing, the operation of dividing the broken mineral into parcels according to the sizes of the particles. This is always performed by passing the material over a screen, or a series of screens, and the book before us contains a full description of the various screening appliances in use. Prof. Louis then proceeds to discuss the subject of sorting—separating, that is, the barren or worthless material from that which is of value, and sometimes dividing minerals of different kinds or qualities. In Cornwall copper ores are picked out from tin ores, and set aside for separate treatment. In many lead mining districts pure galena is picked out from lead ores mixed with gangue, or from mixed lead and zinc ores, which have to undergo more elaborate processes of dressing. On the Witwatersrand the gold-bearing banket is picked out from the barren quartzite which is broken down with it. In most coal mining districts shale is picked out from the good coal; at Rio Tinto various grades are made by sorting, such as quartzose ore, smelting ore, leaching ore, and export ore. Sorting is performed either upon floors, fixed tables, or movable surfaces. The author describes various revolving picking tables, including one in use at the Crown Reef mine, where eight natives at work on the table (only four of whom are, strictly speaking, engaged in sorting) go over from 650 to 700 tons per day, and pick out about 14 per cent. The operating cost appears to be about 4d. per ton on crude ore for working expenses, and about 3d. per ton for maintenance. Next to sorting the question of comminution, or the breaking down of mineral to smaller sizes, is dealt with. The author divides this portion of the subject into:—(a) Coarse breaking—often a preliminary operation; (b) Medium crushing; (c) Fine crushing (pulverising). Various types of mechanical rock-breakers and crushing machines are described and profusely illustrated by diagrams, including the "Blake," the "Gates," the "Dodge," and the "Bartsch" rock-breakers, Marsden's Pulveriser, the Sturtevant Roll Jaw crusher, Krom Rolls, tube mills, and every form of stamp mill in existence. Separation by specific gravity is an interesting branch of the subject which the author does not neglect, nor does he overlook the important questions of pneumatic, magnetic, or electrostatic separation. Finally, Prof. Louis offers some general considerations and suggestions with regard to the design or construction of dressing works, which mining engineers will find extremely useful, although, as the author himself points out, it is impossible to lay down hard and fast rules where a number of different appliances may be used for attaining a given object. The choice of any particular one may, therefore, depend not only on the nature of the materials to be treated, but also upon other conditions, and is frequently influenced by local custom, or even by the idiosyncracies of the engineer in charge. The book contains over 400 illustrations, and a vast amount of care and labour must have been involved in its production. As a contribution to mining literature we consider it of great value."—*Mining World* (London), May 1, 1909, p. 539. (A. R.)

Allsop, F. C. Telephones: Their Construction and Fitting. 8th edition; with Agenda. Cr. 8vo. pp. 230. *Spon.* Net 3s. 6d.

Antimony, its History, Chemistry, Mineralogy, etc., etc. Chung Yu Wang. Illustrated. 8vo. pp. 228. *C. Griffin.* Net 12s. 6d.

Beck, Dr. Richard and Walter Harvey Weed. The Nature of Ore Deposits. 257 illustrations. 8vo. pp. 700. *Hill Publishing Co.* Net 25s.

Blyth, Alxander W. and Meredith W. Foods: Their Composition and Analysis. 6th edition; thoroughly revised, enlarged and re-written. 8vo. pp. 646. *C. Griffin.* Net 21s.

Collins, A. F. The Design and Construction of Induction Coils. 8vo *Spon.* Net 12s. 6d.

Davidge, H. T. and Hutchinson, R. W. Technical Electricity. 2nd edition. (Organised Science Series). Cr. 8vo. pp. xl-539. *W. B. Clive.* 4s. 6d.

Encyclopaedia of Practical Engineering and Allied Trades. Ed. by J. G. Horner. Vol. IX. Spe.-Tim. Imp. 8vo. pp. 240. *Virtue.*

Evans, C. E. Hints to Coal Buyers. With maps and diagrams. Fol. pp. 64. *Business Statistics Publishing Co.* (Cardiff). Net 21s.

Garrett, A. (If) Hydraulic Tables and Diagrams for Practical Engineers. Fol. *Longmans.* Net 15s.

Groth, Dr. L. A. Welding and Cutting Metals by aid of Gases or Electricity. 8vo. pp. xvi-281. *Constable.* Net 10s. 6d.

Gunther, Godfrey. Electro-Magnetic Ore Separation. 81 Illustrations and Diagrams. 4to. pp. 200. *Hill Publishing Co.* Net 12s. 6d.

Harker, A. The Natural History of Igneous Rocks. 8vo. pp. 400. *Methuen.* Net 12s. 6d.

Kleiser. G. How to develop Power and Personality in Speaking. Cr. 8vo. *Funk and W.* Net 5s.

Lea, F. C. Hydraulics for Engineers and Engineering Students. 8vo. pp. 548. *E. Arnold.* Net 15s.

Lewkowitsch, Dr. J. Chemical Technology and Analysis of Oils, Fats and Waxes. 4th edition, entirely re-written and enlarged. 3 vols., 8vo. pp. xx-542, 813, 406. *Macmillan.*

Lorentz, H. A. The Theory of Electrons. 8vo. *Nutt.* Net 9s.

Louis, Henry. The Dressing of Minerals. Illustrated. Roy. 8vo. pp. 556. *E. Arnold.* Net 30s.

Mumby, Allan E. Introduction to the Chemistry and Physics of Building Materials. 8vo. pp. 366. *Constable.* Net 6s

Newell, L. C. A Course of Inorganic Chemistry. Cr. 8vo. *D. C. Heath.* 6s.

Ostwald, Wilhelm. The Fundamental Principles of Chemistry. Authorised Translation by H. W. Morse. 8vo. *Longmans.* Net 7s. 6d.

O'Toole, I. Tables of Seamless Copper Tubes. 12mo. *Spon.* Net 3s. 6d.

Parkes, Louis C. House Drainage: Sewerage, Sewage Disposal in relation to health. Cr. 8vo. pp. 150. *H. K. Lewis.* bds. Net 2s. 6d.

Proceedings of the International Association for Testing Materials. May, 1909. No. 5. 8vo. *Spon.* Net 1s. 3d.

Renwick, W. G. Marble and Marble Working. Illustrated. 8vo. pp. 240. *Lockwood.* 15s.

Rosenberg, E. Electrical Engineering. An elementary text book. 4th edition. 8vo. pp. 282. *Harper.* 6s.

Self-Instruction for Students in Gas Supply, Advanced. By Mentor. Cr. 8vo. pp. 120. *J. Allan.* Net 3s. 6d.

Seymour, Charles, Speaking in Public. How to produce ideas and how to acquire Fluency. Cr. 8vo. pp. xii-208. *G. Routledge.* Net 3s.

Stitt, E. R. Practical Bacteriology, Blood Work and Animal Parasitology. Cr. 8vo. *H. K. Lewis.* Net 6s. 6d.

The National Physical Laboratory, Report for the year 1908. Roy. 8vo. pp. 115. *Parrott* (Teddington).

Ville, M. George. Artificial Manures: Their Chemical Selection and Scientific Application to Agriculture. New edition; revised by Sir Wm. Crookes and John Percival. 8vo. pp. 388. *Longmans.* Net 10s. 6d.

Winzar, O. G. The Instanter Decimal Tables, giving at sight the Decimal Equivalents of the Monies, Weights, Measures, Angles, Fractions, etc., in general use in the British Empire and the United States. Cr. 8vo. pp. 32. *Simpkin.* Limp net 1s. 6d.

Xydis, C. Handbook on Tacheometrical Surveying. 8vo. *Spon.* Net 6s.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 213/09. Fraser & Chalmers, Ltd. (1), Frank Leonard Whitmore (2). Improvements in safety devices for hoisting apparatus. 30.4.09.

(C.) 214/09. The Hon. Charles Algernon Parsons, C.B. Improvements in and relating to commutators for dynamo electric machines. 30.4.09.

(C.) 215/09. Edwin Lette Oliver. Improvements in continuous vacuum filters. 30.4.09.

(P.) 217/09. John Hudson Hughes. Improvements in the operation of the stamps of stamp batteries. 1.5.09.

(P.) 218/09. Thomas Millar Smith. Improvements in means for obtaining better combustion of the fuel in the furnaces of steam generators and the like. 1.5.09.

(C.) 219/09. Paul Gaultier. Instrument for topographical surveying. 3.5.09.

(P.) 221/09 William Bowie Stevenson. Improvements in or relating to amalgamating tables. 4.5 09.

(P.) 222/09. George Ogilvie Strachan. Improved automatic lever safety catch for mine cages, skips, lifts and the like. 4.5.09.

(P.) 223/09. Angus Fraser. Improvements in driving mechanism for ore feeders and the like. 5.5.09.

(P.) 224/09. Angus Fraser. Improvements in ball and roller bearings. 5.5.09.

(C.) 225/09. Alfred Smith. Automatic concentrating machine. 5.5.09.

(P.) 226/09. John Roche. A new and improved box for fixing up flush electric switches. 6.5.09.

(C.) 228/09. Wilfley Mining Machinery Company Limited. Improvements in and relating to apparatus for the separation of solid material from liquids. 7.5.09.

(C.) 230/09. George Gates. Railway rail support and fastener. 7.5.09.

(P.) 231/09. Charles Reeves. Improved method for extracting pulp from mortar boxes. 13.5.09.

(P.) 232/09. Frederick William Slack. Improvements in explosive compositions. 14.5.09.

(P.) 233/09. George Francis Gaydon. An improvement in candle holders for use in the mines. 14.5.09.

(P.) 234/09. Otto Tonnesen. Improved fitting or fixture for support of temporary survey-fix-points or for other purposes. 14.5.09.

(C.) 235/09. Thomas Parker. Improvements relating to the destructive distillation of bituminous coal or similar carbonaceous substances. 14.5.09.

(C.) 237/09. Ludwig Horst. Process and apparatus for utilising carbonic acid for power purposes. 14 5.09.

(P.) 238/09. Henry Samuel Potter. Improvements in means for rotating the tool and feeding water in connection with fluid actuated tool engines. 14.5.09.

(P.) 239/09. Hans Charles Behr. Improvements in fluid meters. 15.5.09.

(P.) 240/09. Ernest St. George Alderton Hughes. Non-refillable bottle. 17.5.09.

(P.) 241/09. John Ernest Roux Adendorff. An improved process for preventing the formation of the "white precipitate" (hydrated zinc oxide) in the precipitation boxes employed for extracting the precious metals from their solutions. 15.5.09.

(P.) 242/09. Alexander John Arbuckle. Improvements in apparatus or plant for treating crushed ore products for the recovery of the metal contents thereof. 21.5.09.

(P.) 243/09. Frederick George Morel. Improvements in centrifugal filters for slimes and the like. 21.5.09.

(P.) 244/09. James Garvie. Improvements in valves applicable for rock drills and other engines or machines. 21.5 09.

(P.) 245/09. Lawrence Scott (1), Charles John Arundel (2). Improved yielding or cushioning guide roller or pulley for use in inclined and other mining shafts or other engineering works where travelling ropes are used. 21.5.09.

(P.) 246/09. Arthur Cyrus Whittome. Improvements relating to the fine grinding of ores and the like. 21.5.09.

(C.) 247/09. William Campbell Paterson. Improvements in pulp agitators. 21.5.09.

(C.) 248/09. George Lawrence Smith. Improvements in or relating to fire and temperature alarms or indicators. 21.5.09.

(C.) 249/09. Stephen Evans Alley. Improvements in gate or sluice valves. 21.5.09.

(C.) 250/09. Guy James Stock. Improvements in connection with bessemer convertors. 21.5.09.

(C.) 251/09. Alexander Albert Holle. Improvements in steam and other fluid engines. 25.5.09.

(P.) 252/09. Hans Charles Behr. Improvements in and relating to pneumatic stamping plant. 25.5.09.

(P.) 253/09. Frank Simon. Improvements in rope jockeys. 25.5.09.

(P.) 255/09. William D'Arcy Lloyd (1), Ernest Thomas Rand (2). An improved medium of precipitating the precious metals from their solutions. 28.5.09

(P.) 256/09. James Stephen McNamara. An improved connection for the drill rods of boring apparatus for compensating for the weight of the rods. 28.5.09.

(C.) 257/09. William Forbes Leslie. Improvements in pulverising mills. 28.5.09.

(C.) 259/09. Robert Harrington (1), Edwin William Sydney Young (2). The self adjustment of brakes on railway vehicles whether worked by vacuum steam or compressed air. 1.6.09.

(P.) 260/09. Frederick Victor William Swanton (1), Thomas Arnott Fletcher (2). Improved apparatus to produce power by the reciprocating motion of two dissimilar fluids acting upon same. 2.6.09.

(P.) 261/09. Alfred Walker. Improvements in apparatus for stopping runaway skips, cages and the like. 2.6.09.

(P.) 264/09. Thomas Parker. Improvements relating to the distillation of coal and other carbonaceous substances. 3.6.09.

(C.) 265/09. Joseph Knight. Improvements in electrically operated rock drills or the like, specially applicable for stoping. 4.6.09.

(C.) 266/09. Thomas Robertson Laing. Improvements in devices for drawing keys, pins, spindles and the like out of pulleys, wheels or other stock. 4.6.09.

(C.) 267/09. William Spiers Simpson (1), Howard Oviatt (2). Improvements in connection with the metallurgy of iron and steel. 4.6.09.

(C.) 268/09. William Ernest Bennett. Improvements in apparatus for distilling mercury from gold amalgam. 4.6.09.

(P.) 272/09. Robert Craig. Improvements in sands distributors for use on tailings and the like. 5.6.09.

(P.) 273/09. Frank Robinson (1), Jacob Andreas Frerichs (2). A new and improved device for working power stamps, hammer rock drills, pile drivers and the like, by means of explosive gases. 7.6.09.

(P.) 274/09. William Cullen (1), Gilbert Frederick Ayres (2). Method of precipitating zinc from solutions. 8 6.09.

(P.) 275/09. Archibald Frank Stuart. Tank strainer. 8.6.09.

(P.) 276/09. Francis Ima Matthews (1), Charles Blagburn (2), Francis Michael Wright (3). Stamp mill. 11.6.09.

(C.) 277/09. James Nicolson Bailey (1), The British Westinghouse Electric and Manufacturing Company, Ltd. (2). Improvements relating to fluid pressure engines. 11.6.09.

(C.) 278/09. Edgar Arthur Ashcroft. Improved process and apparatus for the production of metals of the alkali group by electrolysis. 11.6.09.

(C.) 279/09. Albert Hibbs Taylor. Improvements in hammer drills. 11.6.09.

(P.) 282/09. Hugo Per Wilhelm Esping. Improvements in apparatus for raising liquids. 11.6.09.

(P.) 283/09. Walter Daniel Douglas. Improvements in stamp mills. 12.6.09.

(P.) 284/09. James Irving Williams. An improved furnace for smelting gold and other ores. 12.6.09.

(P.) 286/09. Ernest Slatter Day. Improvements in the continuous treatment of slimes and apparatus therefor. 16 6.09.

(P.) 287/09. Horatio Dunn. Improvements in reciprocating engines. 17.6.09.

(P.) 288/09. Malcolm Canmore Inglis. Improvements in crucible tongs. 17.6.09.

(P.) 289/09. Karl Sophus Sturup (1), Theodore Wilfred Rill (2). Improvements in means for distributing sand on mine dumps, also applicable for other similar purposes. 18.6.09.

(C.) 290/09. Andrew Swan. Improvements in the lixiviation of ores and the like. 18.6.09.

(P.) 291/09. Frank Ondra (1), Alfred Adair (2). Improvements in zinc recovery processes. 18.6.09.

(C.) 292/09. Walter George Kent (1), John Lawrence Hodgson (2). Improvements in fluid meters. 18.6.09.

(C.) 293/09. Eugen Assar Alexis Gronwall (1), Axel Rudolf Lindblad (2), Otto Stalhane (3). Improvements in electric furnaces. 18.6 09.

(C.) 294/09. Axel Rudolf Lindblad. Method of electric smelting of iron steel and other metals and means for carrying out the same. 18.6.09.

(P.) 296/09. Lars Pedersen. Improvements in respiratory devices for use in mining and the like. 18.6.09.

(P.) 298/00. John Bartle (1), John Miller Anderson (2). Block and tile liners for launders and tube mills. 19/6/09.

(P.) 299/09. Samuel Pond Ruthven (1), Thomas Williamson Donlan (2). Improvements in the manufacture of wire screening. 19.6.09.

(P.) 300/09. Frank James Chalmer Carruthers. An improved apparatus for carburetting air. 19.6 09.

(P.) 301/09. Junius Ford Cook (1), Frank Alfred Roberts (2). Improvements relating to the decantation process of ore treatment. 22.6.09.

(P.) 302/09. Harry Davis. A new or improved joint for the tools of percussion or jumper drills. 23.6.09.

(P.) 303/09. Arthur Reginald Angus. Improvements in and relating to devices for preventing collisions between railway trains. "A." 24.6.09.

(P.) 304/09. Philip Herd. Improvements in means for regulating the sensitiveness of tripping devices employed in connection with the automatic operation of electrical switches. 25.6.09.

(C.) 305/09. Augustus Rosenberg. Process of and means for depositing metals upon metallic surfaces. 25.6.09.

(P.) 306/09. Philip Herd. Improvements in means for regulating the sensitiveness of the tripping devices employed in connection with the automatic operation of electrical switches. 28.6 09.

(C.) 307/09. William James Wright (1), Robert Lincoln Armstrong (2). Two-cycle compound engine. 29.6.09.

(C.) 308/09. Ernest Wilhelm Koster. Improvements in or relating to installation of engines working with exhaust steam, and particularly applicable for use in compressing air and other gases. 29.6.09.

(P.) 309/09. Henry Alfred Walker. Improvements in drills or bits for rock drilling and like operations. 1.7.09.

(P.) 310/09. James Henry Ruthven. Automatic safety clutch or device applicable to mine skips, cages, hoists, elevators, and the like. 2.7.09.

(C.) 311/09. John Hutchings. Improvements in and relating to direct acting pumping machinery. 2.7.09.

(P.) 312/09. John Whitehead Wilson Drysdale. Improvements in or connected with centrifugal and rotary pumps. 2 7.09.

(P.) 313/09. Percy Theodore Dunton. Improved flexible steam joint. 3.7.09.

(P.) 314/09. Bertram Richard Avery. A new or improved composition for preventing rain and mois.

ture from penetrating of brick, stone, and cement plastered walls of buildings. 5.7.09.

(P.) 315/09. H. C. Behr. Improvements in clutch mechanism for winding or hauling engines. 5.7.09.

(C.) 316/09. J. C. Snelling. An improved machine for producing slabs for building purposes. 6.7.09.

(P.) 317/09. H. Bone (1), W. H. Shaw (2), J. Mitchell (3). The "B.S.M." pick. 7.7.09.

(P.) 318/09. J. Oliver. Improvements in the lining of tube mills and the like. 7.7.09.

(P.) 319/09. G. H. Kirsch. Improvements in tube mill lining. 7.7.09.

(P.) 320/09. J. H. Anton. Improvements in means for arresting the movement of trucks or other vehicles in the event of the reversal of their direction of travel. 9.7.09.

(C.) 321/09. G. Lezinsky. Improvements in explosive compounds and manufacture of the same. 9.7.09.

(C.) 322/09. A. A. Lockwood. Treatment of ores and carboniferous earths and to the treatment of oils for this purpose. 9.7.09.

(P.) 323/09. H. Schwarz. Improvements in rock drill valves. 9.7.09.

(C.) 324/09. W. A. F. Black. Improvements in primary batteries. 10.7.09.

(P.) 325/09. A. Y Niven. Improvements in tube mill lining means. 10.7.09.

(C.) 326/09. B. E. D. Kilburn. Improvements in or relating to explosives. 12.7.09.

(C.) 327/09. B. E. D. Kilburn. Improvements in or relating to explosives. 12.7.09.

(C. 328/09. H. Perrins. Improvements in wheels for motor and other road vehicles. 12.7.09.

(P.) 329/09. P. T. Dunton. Tube mill feeders. 12.7.09.

(P.) 330/09. Hans Charles Behr. Improvements in pipes for conveying corrosive or abrasive matter. 15.7.09.

(P.) 331/09. William George Manners. An improved centrifugal filtering machine. 15.7.09.

(P.) 332/09. John Roberts. Wind cooling tower. 16.7.09.

(C.) 333/09. William Miller. Improvements in crucible furnaces. 16.7.09.

(C.) 334/09. Hon. Charles Algernon Parsons. Improvements in and relating to dynamo Electric machinery. 16.7.09.

(C.) 335/09. George Harrison Sheffield (1), James Denis Twinberrow (2). Improvements in and connected with bogies for rolling stock. 16.7.09.

(P.) 337/09. Hans Charles Behr. Improvements in means for storing and transporting ore in connection with reduction mills. 16.7.09.

(P.) 338/09. Hans Charles Behr. Improvements in internal combustion percussive apparatus. 16.7.09.

(P.) 339/09. Edward Jacob Laschinger. Improvements in stamp batteries. 20.7.09.

(P.) 341/09. C. Bosch (1), H. Keller (2). Improvements in the absorption of oxides of nitrogen. 22.7.09.

(P.) 342/09. G. A. Mower (1), F. W. R. Williams (2). Improvements in ore feeders for stamp mills. 23.7.09.

(P.) 343/09. T. Bubola. Process of treating crushed ore with liquid. 23.7.09.

(C.) 344/09. T. Bubola. Apparatus for treating crushed ore with liquid. 23.7.09.

(P.) 345/09. A. J. Arbuckle. Improvements in straightway valves. 24.7.09.

(P.) 346/09. Improvements relating to vats, tanks or vessels for separating solids from liquids. 24.7.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Allen, J. A., *l/o* Buenos Aires; Castaño Nuevo, prov. San Juan, Argentine Republic.

Anderson, P. von M., *l/o* Johannesburg; New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni.

Avesson, R., *l/o* Austria; 17c, Fabrikstrasse, Reval, Estonia, Russia.

Bell, S. W., *l/o* Newcastle; Ashanti Goldfields Corporation, Ltd., Obuassi, via Sekondi, Gold Coast Colony.

Bennett, R. W. *l/o* Maraisburg; Ferreira G. M. Co, Ltd., P. O. Box 1021, Johannesburg.

Blewett, E., *l/o* Selukwe; West Nicholson, Rhodesia.

Clennell, J. E., *l/o* France; 6, Great James Street, Bedford Row, London, W.C.

Corstorphine, Dr. G. S., *to* 14 and 15, National Bank Buildings, Simmonds Street, P. O. Box 1758, Johannesburg.

Crocker, C. J., *l/o* Barberton; Knights Deep, Ltd., P. O. Box 143, Germiston.

Davies, John, *l/o* Cleveland; St. Kilda, Muizenberg, C.C.

Denny, H. S., *l/o* London; 50 51, Quirk Edificio, No. 1, Gante, City of Mexico, Mexico.

Douglas, Ross E., *l/o* Mexico; 2,330, La Mirada Avenue, Los Angelos, Cal., U.S.A.

Evans, Lewis, *to* Rand Club, P. O. Box 1032, Johannesburg.

Hamilton, E. M., *l/o* Middlesex; c/o Messrs. Chas. Butters & Co, Ltd., 333, Kearney Street, San Francisco, Cal., U.S.A.

Haugh, H., *l/o* Sydney; Pipeclay Creek, via Fambula, New South Wales.

Hilditch, C. H., *l/o* Knights; Poste Restante, Boksburg North.

Hilliard, C. B., *to* Crown Mines, Ltd., "C" Section Reduction Works, Fordsburg.

Hogenraad, G. B., *to* Gold Exploration Co., Lebong Kandi-, via Post Ketahoen, Benkoelen, Sumatra.

Kyle, J. T., *l/o* Denver; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

Lichtenstein, A., *l/o* Germiston; Van Ryn G. M. Estates, Ltd., P. O. Box 22, Benoni.

Simpson, F. L., *l/o* Germiston; Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg.

Thomas, D. L., *l/o* Johannesburg; East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand.

Tonnesen, O. S., *l/o* East Rand; East Rand Proprietary Mines, Ltd., 4, London Wall Buildings, London, E.C.

Trevor, T. G., *l/o* Barberton; P. O. Box 401, Pretoria.

Warden, P. S., *l/o* Battlefields; Ardpatrick Mine, Queque, Rhodesia.

Whytock, P. R., *l/o* So. Dak; Box 19, Rawhide, Nevada, U.S.A.

SUPPLEMENT TO THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA.

JULY, 1909.

PLATES ILLUSTRATING

THE

INAUGURAL ADDRESS

ON

THE CRYSTALLISATION OF IRON AND STEEL

BY

A. McARTHUR JOHNSTON, M.A., M.I.M.M., F.C.S.

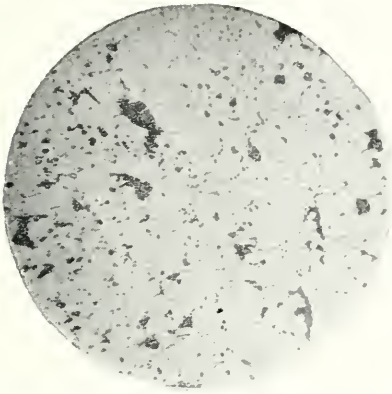


FIG. 1.—Wrought iron, × 100 diameters. Cross section.

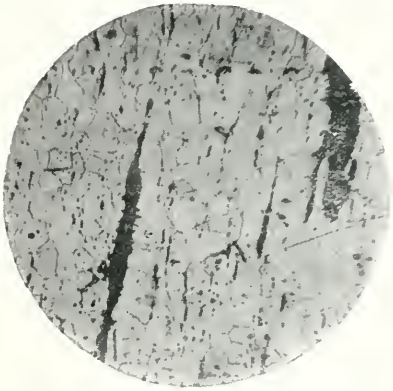


FIG. 2.—Wrought iron, × 100 diameters. Longitudinal section.

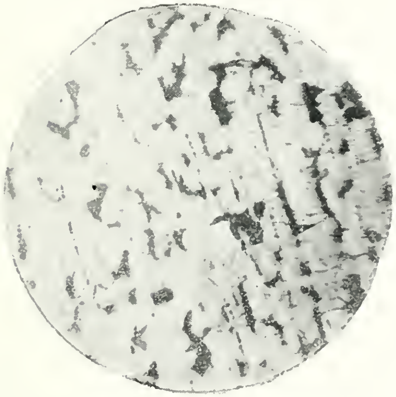


FIG. 3.—Mild steel, × 100 diameter. 0·2% carbon.

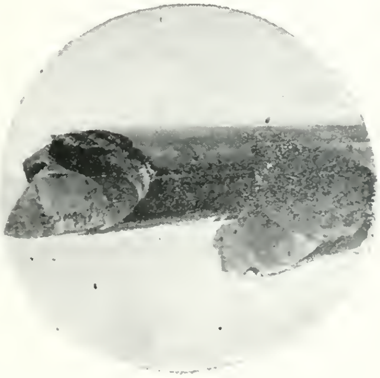


FIG. 4.—Broken cam shaft.

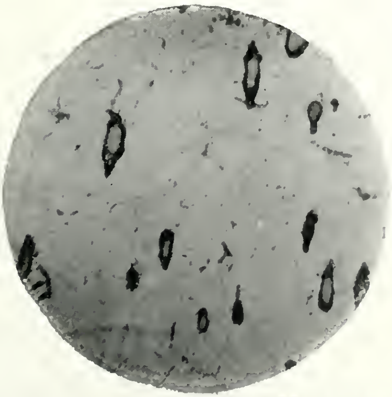


FIG. 5.—Part of cam shaft, × 100 diameter. Wrought iron, Cross section.

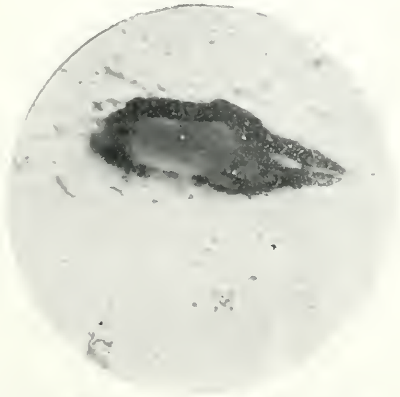


FIG. 6.—Same as Fig. 5, × 400 diameter.

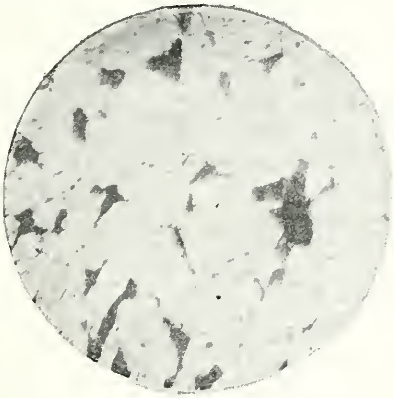


FIG. 7.—Part of cam shaft, × 100 diameter. Mild steel. Cross section.

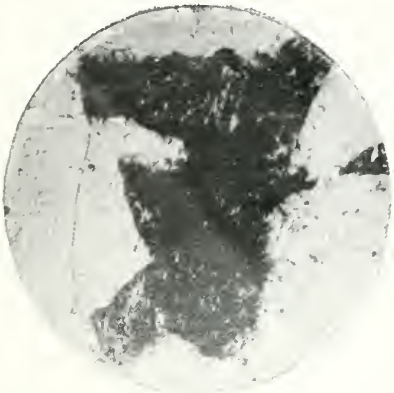


FIG. 8.—Same as Fig. 7, × 400 diameter.

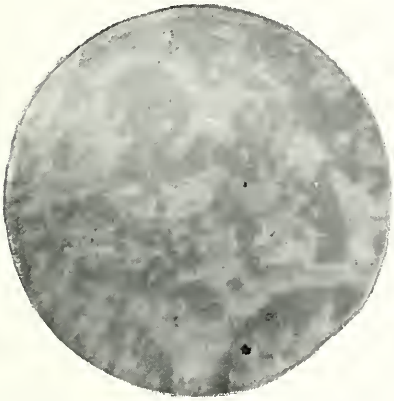


FIG. 9.—Low carbon steel, × 100 diameter. 0·30% carbon.

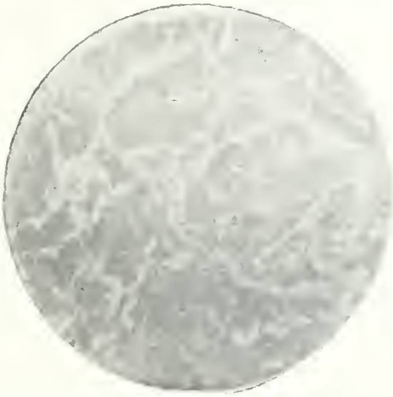


FIG. 10.—Medium carbon steel, × 100 diameter. 0·50% Carbon.

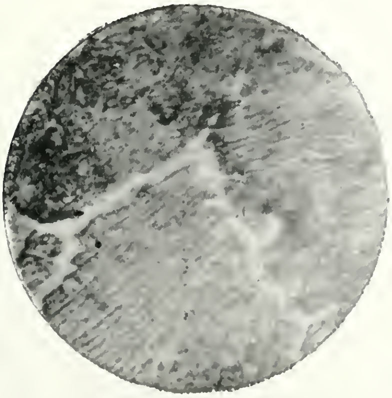


FIG. 11.—High carbon steel, × 1000 diameter. 0·75% carbon.

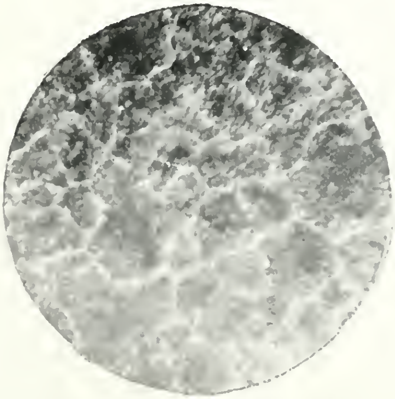


FIG. 12.—High carbon steel, × 100 diameter. 1·25% carbon.

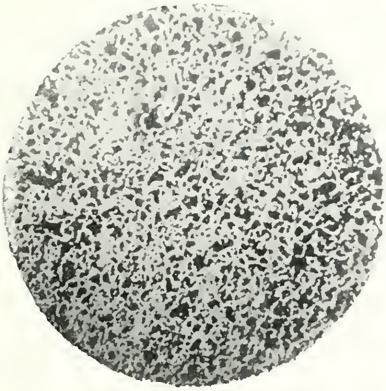


FIG. 13.—Steel.
Grain size at 900°C (Campbell).

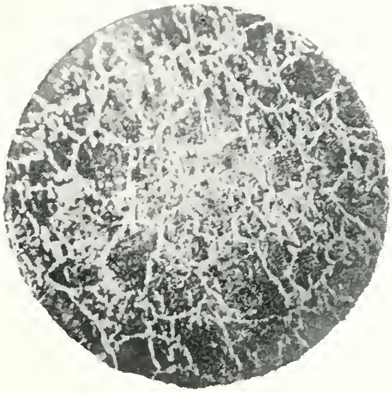


FIG. 14.—Steel.
Grain size at 1200°C (Campbell).

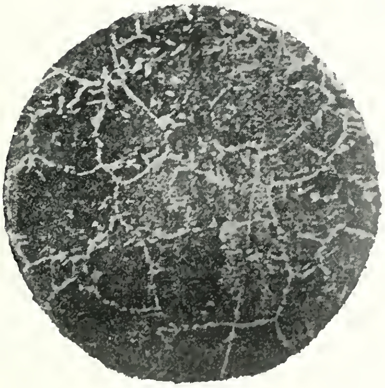


FIG. 15.—Steel.
Grain size at 1300°C (Campbell).

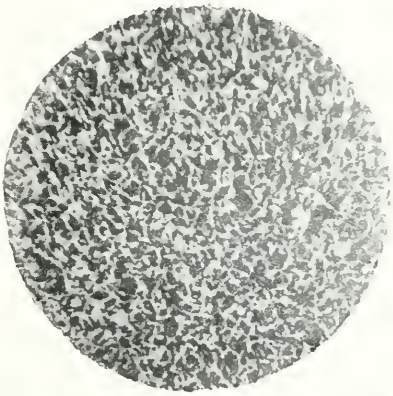


FIG. 16.—Steel (forged). Heated to 800°C and cooled in air. ×100 diameters.

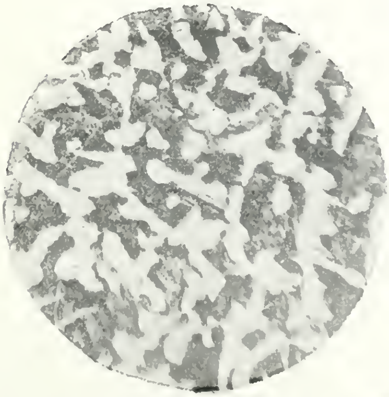


FIG. 17.—Steel (forged). Heated to 1000°C and cooled in air. ×100 diameter.

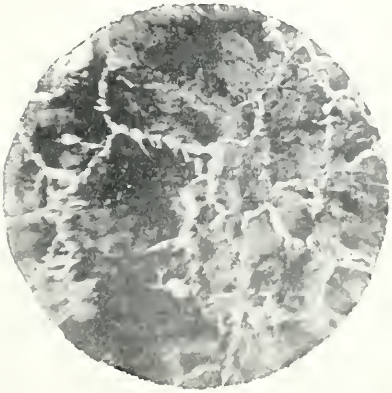


FIG. 18.—Steel (forged). Heated to 1000°C and cooled in furnace. ×100 diameter.

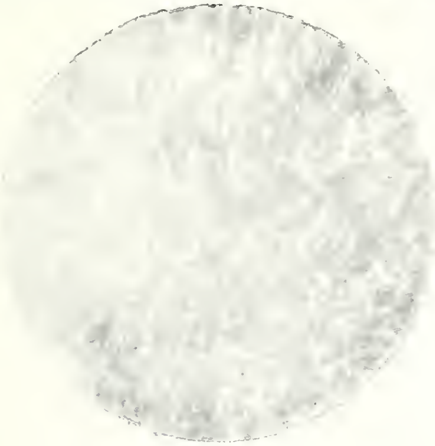


FIG. 19.—Wire, × 100 diameter.
Regular structure.

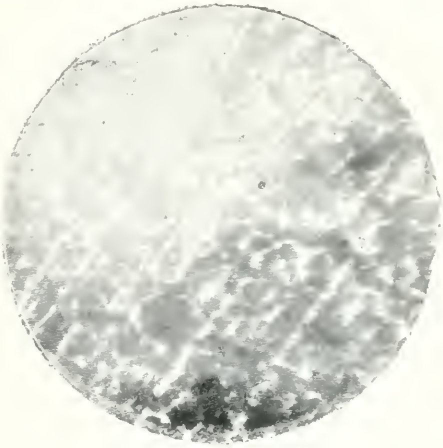


FIG. 20.—Wire, × 100 diameter.
Longitudinal section, showing cold work.

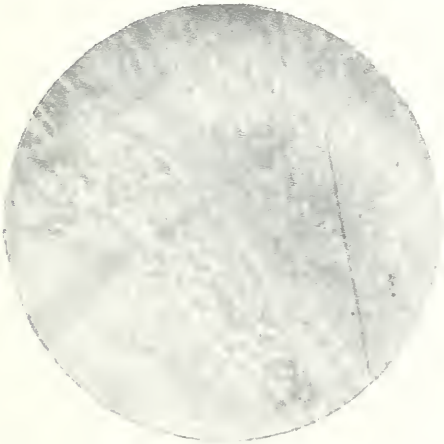


FIG. 21.—Wire, × 100 diameter.
Longitudinal section. Distorted grains and cracks.

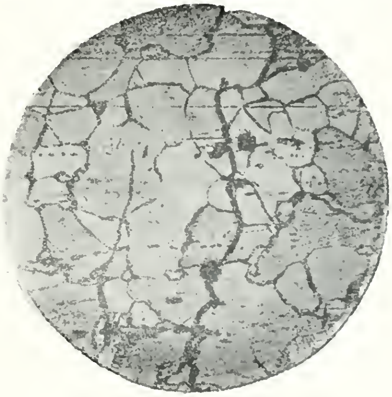


FIG. 22.—Polished surface with large crack (Ewing & Humphrey).

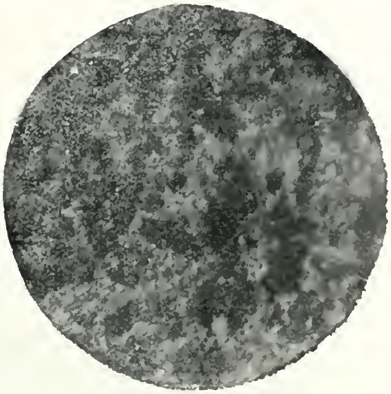


FIG. 23.—Outside section, × 400 diameter
Before use

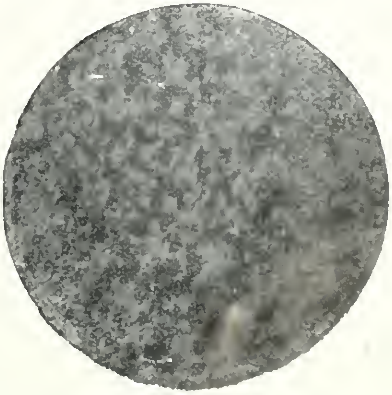


FIG. 24.—Middle section, × 400 diameter,
Before use.

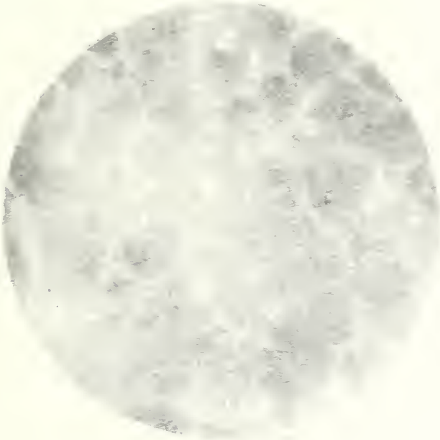


FIG. 25.—Inner section, × 400 diameter.
Before use.

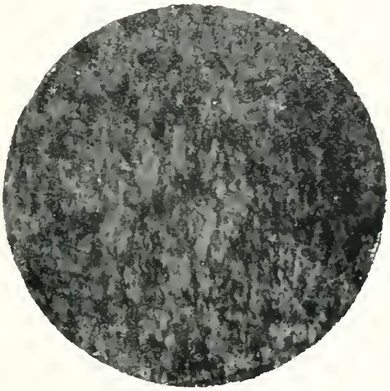


FIG. 26.—Outer section, × 400 diameter.
After breaking.

FIG. 27.—Middle section, × 400 diameter.
After breaking.

FIG. 28.—Inner section, × 400 diameter.
After breaking.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. X. AUGUST, 1909. NO. 2.

Proceedings AT Ordinary General Meeting, August 21, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, August 21st, Mr. McArthur Johnston (President) in the chair. There were also present:—

64 Members: Messrs. W. R. Dowling, Fraser Alexander, E. H. Croghan, K. L. Graham, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, R. G. Bevington, W. A. Caldecott, W. Cullen, S. H. Pearce, J. R. Williams, Prof. J. Yates, E. L. Adams, H. A. Adams, A. Avent, E. Blume, A. J. Bowness, W. Broom, S. Caris, J. Carruthers, F. F. Chapman, F. W. Cindel, M. H. Coombe, J. F. Ferguson, A. Gillies, G. Goodwin, J. H. Harris, A. H. Hartley, C. B. Hilliard, J. H. Johnson, J. Acheson Jones, T. W. Jordan, J. Kennedy, J. F. Mitchell, P. T. Morrisby, M. T. Murray, S. Newton, C. F. Parry, F. D. Phillips, W. S. V. Price, S. C. Quin, L. J. Robinson, N. Rodgers, A. Salkinson, H. A. Scarf, W. Sharp, A. M. Sim, A. L. Spoor, R. S. Stokes, C. T. Sydenham, J. A. Taylor, A. Thomas, K. Tonnesen, J. F. Walker, J. P. Ward, F. W. Watson, P. H. M. Whyte and L. J. Wilmoth.

29 Associates and Students: Messrs. A. R. Adams, A. G. Anderson, C. J. Crocker, J. Cronin, C. N. Davies, C. L. Dewar, R. Fiander, J. Gibson, H. W. Graham, P. H. Grumitt, C. van Hasselt, W. J. R. Hunter, A. King, H. G. Kirkland, H. Lomberg, W. H. Odgers, W. Parfitt, J. G. Peebles, H. B. Powter, A. Radley, W. G. Reeler, H. Stadler, W. A. C. Tayler, G. G. Thomas, W. E. Thorpe, C. Toombs and A. L. Wright.

22 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the July *Journal*, were confirmed.

NEW MEMBERS.

Mr. J. R. Williams and Prof. J. Yates were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

ADAMS, ALFRED CHARLES, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

BUUREN, WILLIAM VAN, Crown Mines, Ltd., P. O. Box 102, Fordsburg. Mill Foreman.

CHILTON, JAMES, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Miner. *(Transfer from Associate Roll.)*

COLL, DANIEL, Jumpers G. M. Co., Ltd., Cleveland. Cyanider.

DONALDSON, THOMAS, Dynamite Factory, Modderfontein. Technical Chemist.

HURLBATT, CHARLES, Eldorado Mine, Lomagundi. S Rhodesia. Mill Manager.

MORRIS, JAMES WILLIAM HENDERSON, East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Assayer.

MURNING, JOHN, Luipaardsylei Estate & G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Tubemiller.

OSTERLOH, ERNEST ALBERT, Simmer Deep, Ltd., P. O. Box 178, Germiston. Slimes Foreman. *(Transfer from Associate Roll.)*

RASCHER, LOUIS HUGO, Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Assistant Mine Surveyor.

REYNOLDS, JAMES, Consolidated Langlaagte Mines, Ltd., P. O. Box 25, Langlaagte. Mill Foreman.

SCOTLAND, ROBERT DUNCAN, East Rand Proprietary Mines, Ltd., P. O. Box 37, East Rand. Assayer.

TAGGART, THOMAS, Porges Randfontein G. M. Co., Ltd., Randfontein. Mill Manager.

TAYLER, WALTER ALFRED CHARLES, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Foreman. *(Transfer from Associate Roll.)*

THORBURN, JOHN MACKAY, P. O. Box 3427, Johannesburg. Analytical Chemist.

TOOMBS, CHRISTOPHER, East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand. Metallurgical Chemist. (*Transfer from Associate Roll*).

WELLS, FRANK, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Amalgamator.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

A: Associates.—

CHOMLEY, WILLIAM BURGH, Day Dawn, Western Australia. Cyanider.

COOK, BASIL, Vogelstruis Consolidated Deep, Ltd., P. O. Box 3, Florida. Assayer.

Crea, John M., Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tube-miller.

Dixon, John McKay, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assistant Mill Foreman.

Donald, Henry Stone Hutcheon, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Mining Engineer.

Fox, George William, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

Hunter, William John Russell, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

Johnson, Cecil Robert, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

Jupp, Herbert Lauder, French Rand G. M. Co., Ltd., P. O. Box 30, Luipaardsvlei. Cyanider.

King, Andrew, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

Lewis, Arthur, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Millman.

Lomberg, Henry, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Assistant Smelter.

Mackilligan, Hector Rannie, South Kalgurli G. M. Co., Ltd., Fimiston, Western Australia.

Martin, William Alexander, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.

Meredith, Vaughan Wilson, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Mill Manager.

McAlister, Alister Gordon, Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Assistant Surveyor.

New, William Stanley, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.

Packman, Augustus Frederick, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanider.

Schaefer, Benjamin, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

Taylor, Hamish Ernest Oliff-, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanider.

Ward, Harry, Vogelstruis Estates and G. M. Co., Ltd., P. O. Box 116, Roodepoort. Amalgamator.

Whittuck, William Henry Giffard, Troitzk Goldfields, Ltd., Kotchkar, Orenburg Government, Russia. Metallurgist.

Willcox, Alexander, New Rietfontein Estates G. M. Co., Ltd., P. O. Rietfontein Mines. Tube Mill Foreman.

As Students.—

Bolt, Walter Carl, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.

Botha, Fritz Alexander, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tube Mill Learner.

Chadwick, Herbert Hamilton, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.

Hains, Arthur Rutherfoord Bolt, Robinson Deep G. M. Co., Ltd., 34, Becker Street, Yeoville, Johannesburg. Mill Learner.

Medlin, Edward James George, Rose Deep, Ltd., P. O. Box 6, Germiston. Tube Mill Learner.

Mortimer, Norman, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.

Musson, Gordon, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube Mill Learner.

O'Brien, Horace Vincent, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.

Radley, Andrew, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Reduction Works Learner.

Reeler, William George, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Extractor House Learner.

Rowe, John Lofflin, Nigel Deep, Ltd., P. O. Box 50, Nigel. Assay Learner.

Stewart, Victor Mallet James, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.

Smith, Harold James, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.

The President : I think it is a very good sign of the prosperity of the Society that at this meeting we have 53 new members, associates, and students brought forward. It is also satisfactory to note that we have no less than six Past-Presidents of the Society present this evening. We have not always that pleasure.

General Business.

The Secretary : I have received the following note from Capetown :—

Mr. A. A. Coaton (*Member*) : I would like the Council to consider the advisability or otherwise of having the agenda portion of the journal printed only on one side of the paper. The other side could either be left blank, or could be let out for advertisements. In the former case, the blank space would prove very useful for notes, remarks, etc., and in the latter case the additional revenue should more than cover any extra postage entailed by the increased bulk. In either case members could then dissect their journals for filing purposes, without any fear of destroying valuable matter on the other side.

Of course, no alteration need be made in the present form of the bound volumes.

The President : This matter will be brought before the Council at its next meeting.

RAPID ECONOMICAL METHOD OF MAKING SILVER AMALGAM.

Mr. E. H. Croghan (*Member of Council*) : In the November *Journal* of last year Mr. Caldecott gives a method as described by Louis. Recently a large quantity was required at very short notice, and Mr. Gillies and I were requested to make the amount within a few hours, and as cheaply as possible. In view of this, we naturally could not use silver nitrate ready made, so we obtained some standard silver. Obviously it was quite out of question adopting the method of Louis, so the following process was evolved.

The crude silver is dissolved in dilute HNO_3 (1 : 1), avoiding unnecessary excess, in a porcelain basin with the aid of gentle heat. About an equal volume of ordinary water is added, and the silver precipitated by a strong solution of commercial NaCl. (This is conveniently done in a large wide-necked stoppered bottle and well shaken after each addition of NaCl solution.) The precipitate of AgCl is allowed to subside, the supernatent liquid poured off, and the precipitate thoroughly washed by decantation. Transfer precipitate to a double cloth filter (an ordinary linen handkerchief is very convenient) and hand squeeze AgCl into a lump, which is then easily removed to a porcelain basin. Add HCl (1 : 1) and a good number of 3 in. wire iron nails, and gently heat producing a vigorous action until all the AgCl is reduced to amorphous silver, stirring with a glass rod from time to time. The whole reaction takes a very short period. Thoroughly wash by decantation, picking out any undissolved iron nails. Transfer amorphous silver to a porcelain mortar, add a slight excess of mercury and rub well with pestle. The amalgam thus formed is washed several times with ordinary water, keeping the amalgam worked with the hand. When bright and clean, it is squeezed in a cloth to remove excess of mercury, leaving a clean silver amalgam. By this process 100 oz. of silver amalgam can easily be made in three hours.

The process is exceedingly cheap, the only so-called expense being the initial crude silver. As a suggestion, instead of possibly wasting AgCl residues on the mines, these could easily be worked up at any time into silver amalgam by the assayer or other competent person. All materials are *in situ*, viz., AgCl, HCl, wire iron nails or pieces of scrap iron and mercury, etc. If the amalgam is always squeezed under similar conditions, it is a simple matter to state what quantity of mercury should be added to have the consistency required for practical use, or it can be left to the judgment of the amalgamators themselves.

In conclusion, we would state that a satisfactory product is only obtained with freshly made amorphous silver.

The President: I am sure we must all thank Mr. Croghan for bringing forward this note, showing a quick and practical method for obtaining silver amalgam.

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

Mr. W. Cullen (*Past-President*): My attention has been drawn to a slight omission on my part in connection with the paper I read before the Society. I find, on going through the bye-laws, that I have not fully conformed to paragraph 14 which says, among other things, that the composition of the body with which I am dealing should be disclosed. This I avoided intentionally, because at the time the paper was written, the explosive had not been specially authorised by the Inspector of Explosives. I propose to remedy that omission at the next monthly meeting, and at the same time to give a few interesting results obtained since writing the paper, with regard to gases given off in the burning of explosives.

The President: Do I understand that you are going to tell us the composition of this explosive?

Mr. W. Cullen: Yes.

THE CONTINUOUS COLLECTION OF SAND FOR CYANIDING.

By W. A. Caldecott (Past-President).

The advance since the war in metallurgical practice on the Rand has left little to be desired in the percentage recovery of gold content of the ore now economically practicable. Given adequate plant, a total residue of only one-third of a pennyweight of fine gold per ton of ore,—equivalent to seventeen pence in value, or to 93% extraction on 4·8 dwt. ore or 96% on 8·3 dwt. ore—is usually obtainable with profit. The increased scale of operations, together with many minor improvements, has likewise reduced working costs so that the total cost of ore treatment—including surface transport, breaking, sorting, stamp-milling, tube-milling and cyaniding—has, in some cases, fallen below 4s. per ton, or less than the value of one dwt. of gold. Our knowledge, however, of what may be profitably done is frequently in advance of what it is actually possible to do with the existing equipment of many reduction plants. The enormous scale of operations prevents any improvement being considered insignificant in the aggregate, whether in the percentage recovery of the gold or in more economic work or in reduction of interest and redemption of capital sunk in plant.

The capital expenditure on plant still remains absolutely high for a huge modern plant handling several thousand tons of ore daily. Much has been saved by the use of large units—trucks, stamps, tube-mills, vats, pumps, piping and so forth—and much effort is at present being devoted to still further reducing the number of crushing units needed and their aggregate cost.

As regards the secondary treatment, however, the writer was authorised in the beginning of

1907 to endeavour to effect some saving in the cost of the usual cyanide plant. On consideration it appeared that the most likely prospect of success lay in eliminating the sand collecting vats, or rather in utilising them for treatment purposes instead of merely as sand storages, which take the place of the old tailing dams. The upper vats of the superimposed type of sand plant were, on their first introduction before the war, utilised as treatment vats as well as sand collectors, but the results of contamination of mill water with cyanide caused this practice to fall into disuse, and in any case a considerable time elapsed between the filling of a vat and its charge yielding gold-bearing solution to the boxes. Hence, although it reduced cost of transfer to a minimum, this type of plant fell into disuse as being expensive, through having only one-half its capacity available for extraction purposes, and was replaced by separate blocks of collectors and leachers (with belt or truck transfer) in the ratio of one to two or two to five, which is now the common practice.

On considering how to utilise sand collecting vats for treatment purposes and to retain their advantages, including minimum ground space required, whilst avoiding the difficulties mentioned, the problem became essentially that of continuously removing the water and slime from the tailing pulp so as to leave the sand in proper condition for cyanide treatment. The use of conical classifiers appeared the most ready means of removing the great bulk of the slime and water, and in the early part of 1907 the writer tried at the Knights Deep Simmer East joint plant various combinations of such classifiers with other devices for removing the surplus moisture from the thick sand pulp underflow of the classifiers. The ordinary type of slime filter with vertical filter leaves was obviously inapplicable for the purpose, and at the same time a continuous non-intermittent action was desirable, whilst the question of washing the sand cake on the filter did not arise at this stage. Simplicity as well as economy of operation were needed, and obtaining the necessary feature of large capacity was facilitated by the more porous nature of sand than slime. All things considered a horizontal type of vacuum filter finally appeared the most feasible device. The first filter constructed on this principle was a horizontal launder with false filter-cloth bottom, below which a vacuum was maintained. The sand pulp was continuously fed into the filter launder at one end and scraped slowly forward by a belt scraper conveyor to the other. Whilst a certain measure of success was thus attained, the continuous movement and disturbance of the layer of sand pulp undergoing filtration prevented efficient removal of moisture, and the trial of another filter of the same type but with a worm-scraper conveyor yielded no better results.

Finally there was installed a slowly rotating horizontal filter, which proved so satisfactory that little modification in its design, except increase in dimensions and variation in details of driving mechanism, has since been made. (See Photograph A.) This sand filter was 10 ft. in

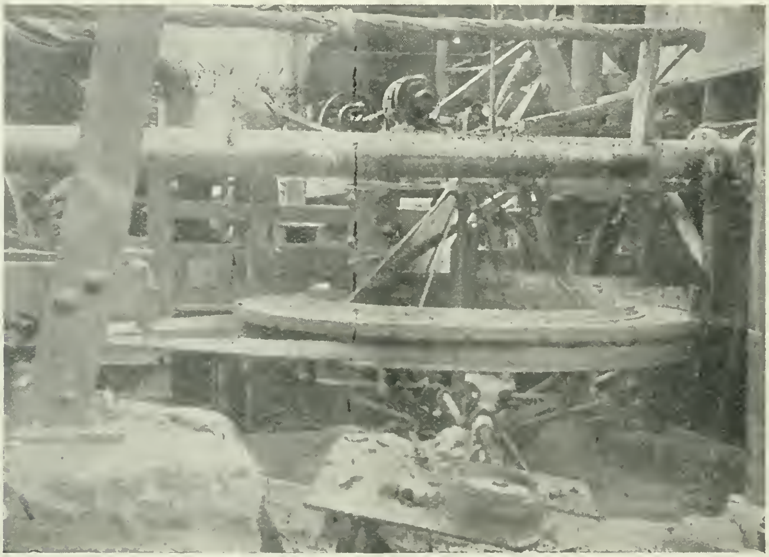


Photo A.—10 ft. sand filter table erected at Knights Deep-Simmer East joint plant (1907).

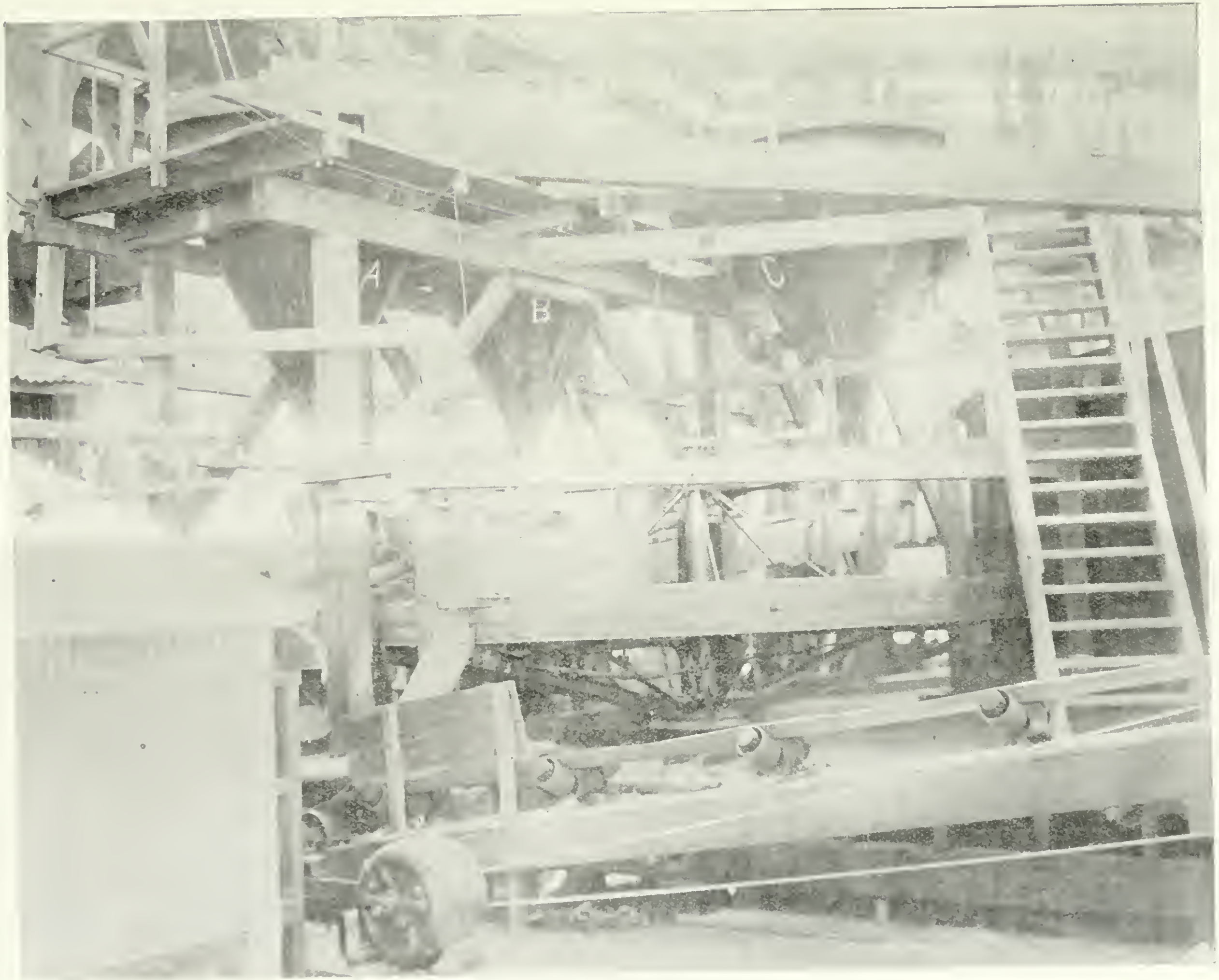


Photo B.—15 ft. sand filter table erected at Knights Deep-Simmer East joint plant (1907), with belt transfer of sand to cyanide treatment vat.

external diameter and consisted of an annular launder 12 ins. wide containing a filter cloth upon a grating as a false bottom. The filtering area was thus 28·3 sq. ft. The space under the filter cloth was connected by means of radial pipes to a central hollow spindle, in which a vacuum was maintained from a receiver. To the lower portion of the receiver was connected a water pump and to the upper a vacuum air pump.

This filter gave such promising results, by handling sand in unwashed sand pulp at the rate of up to 200 tons per day, that another 15 ft. in external diameter and with 18 ins. filtering breadth was constructed at the Knights Deep. Among other results over 1,600 tons of sand were collected by this filter and transferred by belt to the leaching vats, and when treated yielded somewhat better extraction than usual. This arrangement is seen in Photograph B. (see p. 45) as well as in Fig. 1., except that no washing cone was installed, although shown in the latter. This system is hence quite practicable, and has the advantage of completing the sand treatment in one vat without subsequent transfer, but the belt installation may be somewhat expensive and it is required to run as continuously as the mill, whilst the vat capacity available is not fully utilised since cyanide treatment does not begin immediately the sand leaves the filter table.

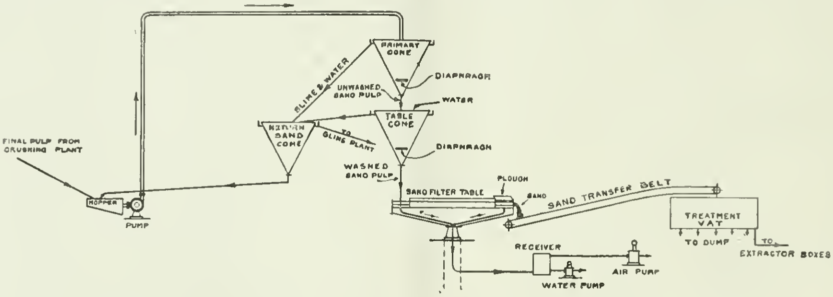


Fig. 1.—Diagram of sand plant with belt transfer of sand to vat from sand filter table.

In the early part of 1908, owing to lack of space for such work on the Knights Deep, this 15 ft. sand filter was transferred to the Simmer and Jack and a long series of systematic trials carried out; this filter was subsequently taken into regular use by the Simmer and Jack Proprietary Mines and is still working. Whilst in the experimental stage an attempt was made to deliver to it a very thick pulp by separating the sand in the underflow of the primary cones from excess of slimes and water by means of a belt scraper conveyor,* working in a wide box with one sloping side (Photograph C.) (see p. 47). Whilst the product was excellent and fall was saved, the capacity of this device of about 140 tons of sand per day was far below that of the filter table, and hence inadequate as a unit for a large plant.

At the present time the underflow of the primary cones at this installation, being too low to gravitate upon the table, is mixed with slimy water from the tailing pulp to form a fluid pulp and pumped into a secondary cone, placed above the table and delivering upon it a thick flow of pulp containing about 30% of moisture To ensure a thick steady underflow of pulp in large amount, a disc, which I have termed a diaphragm, is placed near the bottom of the cones, which are run nearly filled with settled sand. This device has been adapted for tube-milling classification, and its use for this purpose has been more fully described elsewhere.* The sand from which the surplus moisture is removed during the almost complete circuit of the filter, is continuously scraped off by a fixed inclined plough some three feet behind the point of onflow. It falls into a hopper where it mixes with a stream of cyanide solution and is pumped to the distributor of the collecting and treatment vat, while the solution overflows to a solution storage and is returned by a pump to the hopper, thus completing the circuit. In this way the transfer of the sand from the filter to the vat is effected as a pulp, and an excellent start of the dissolving operation is made by the agitation and aeration the sand undergoes in the pump, and during its travel along the delivery pipe or launder to the collecting vat. The system of solution transfer has the advantage of great flexibility and relatively small cost of installation, whilst the operating cost is mainly that of power and maintenance of centrifugal pump liners and propellers. The dissolving of the gold in the sand hence begins on falling into the hopper, within say half an hour after the ore is crushed, and by the time the vat is filled and drained half the gold in the sand is dissolved.

An important feature of the Simmer and Jack trials was the determination of the relative

---

*See McDermott & Duffield's "Losses in Gold Amalgamation" (1890) p. 32; and J. C. Scobey in Ingalls' "Metallurgical Mill Construction" (1906) p. 216.

* See this *Journal*, March, 1909 p. 312.

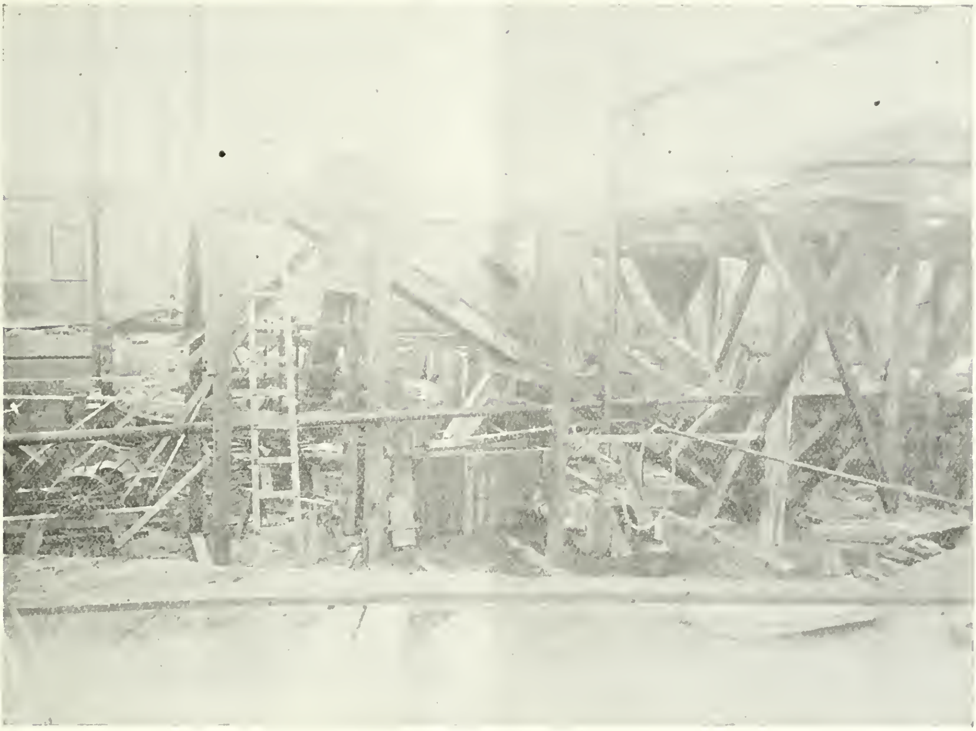


PHOTO C.—15 ft. sand filter table, as shown in (B), but after transfer to Simmer & Jack Proprietary Mines (1908), and with inclined scraper conveyor delivering sand from underflow of cones to filter.

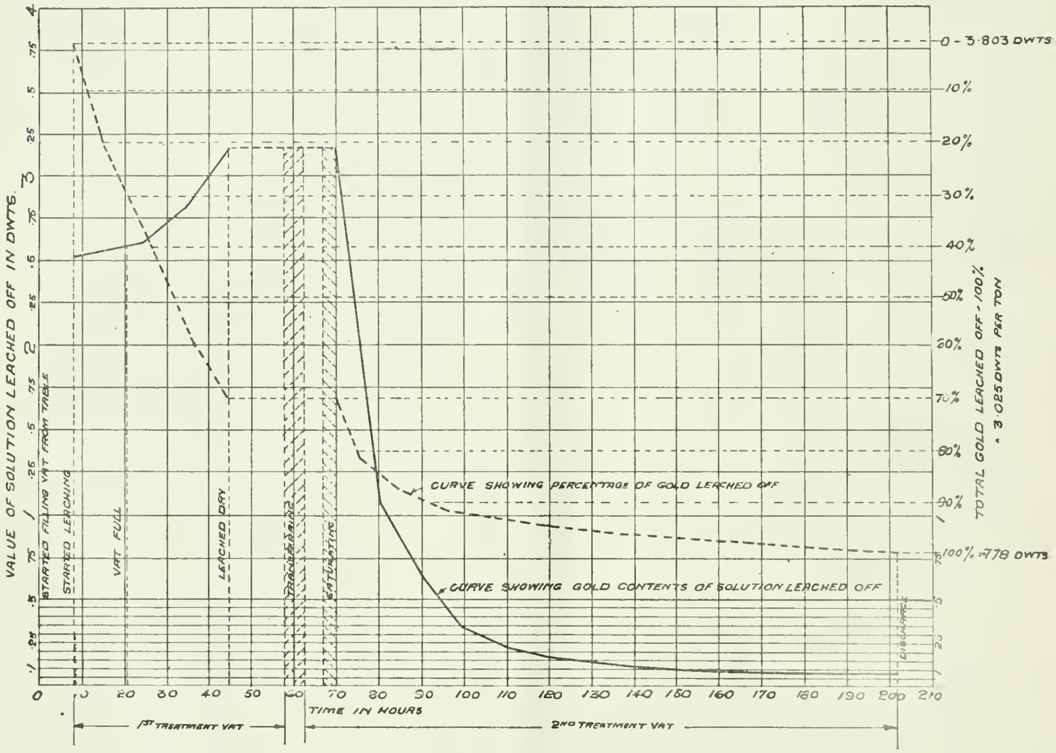
extraction yielded by completing the treatment of the sand in the vat where collected, as compared with the result of transferring the charge by truck after partial treatment to another vat. The curves obtained (Figs. 11*a.* & *b.*) (see pp. 48-49) show that whilst the final result was much the same yet the transfer, and consequent aeration of the particles whilst moistened with cyanide solution, ensured a quicker dissolution of gold, so that whilst six days actual treatment sufficed with transfer, nine days were required to complete the single vat treatment.

The satisfactory results at the Simmer and Jack since taking the sand filter into regular use, are compared with the results on the remainder of the sand plant in the table below. The Simmer and Jack sand plant consists of separate blocks of collecting vats, fitted with Butters & Mein distributors, and leaching vats to which the charges after collection and draining are transferred by truck. Some of the collectors were arranged to receive the sand-solution pulp pumped from the hopper beside the sand filter table, and these charges after draining to the extractor boxes were transferred to the corresponding leaching or second treatment vats. The remainder of the plant was run as usual. The 15 ft. table has been in regular use from the end of last year and has handled up to 10,000 tons of sand per month. The following comparative results are averages from January to July, inclusive of this year of ordinary and continuously collected sand :—

	Continuous Collecting.	Ordinary System.
Tons of sand treated monthly	7,937 tons	37,009 tons
Grading analysis—		
+ 60 (0·010 in.)	12·9%	10·9%
− 60 + 90 ...	32·7%	24·1%
− 90 (0·006 in.)	54·4%	65·0%
Ratio of solution to sand applied as washes to charge	1·54 : 1	2·02 : 1
Number of days treatment in second vat after truck transfer	4·09 days	6·50 days
Assay value of last drainings	0·083 dwts.	0·200 dwts.
Assay value of sand before treatment ...	2·591 dwts.	2·510 dwts.
Assay value of sand after treatment ...	0·454 dwts.	0·462 dwts.
Percentage Extraction	82·478%	81·594%

It will be observed that owing to the more perfect elimination of slime the sand filter product is coarser and richer than the ordinary sand, and that the solution precipitated and the time of treatment in the secondary vats are much less with filter-table sand than ordinary sand. The assay values of the last drainings and of the residues are in favour of filter-table sand, whilst the percentage extraction likewise compares favourably.

Early in the present year two 20 ft. diameter sand filters, each with filtering launder 30 in. wide and 137·5 sq. ft. in area, were installed at the Simmer Deep-Jupiter joint plant, and after some preliminary runs have handled all the sand

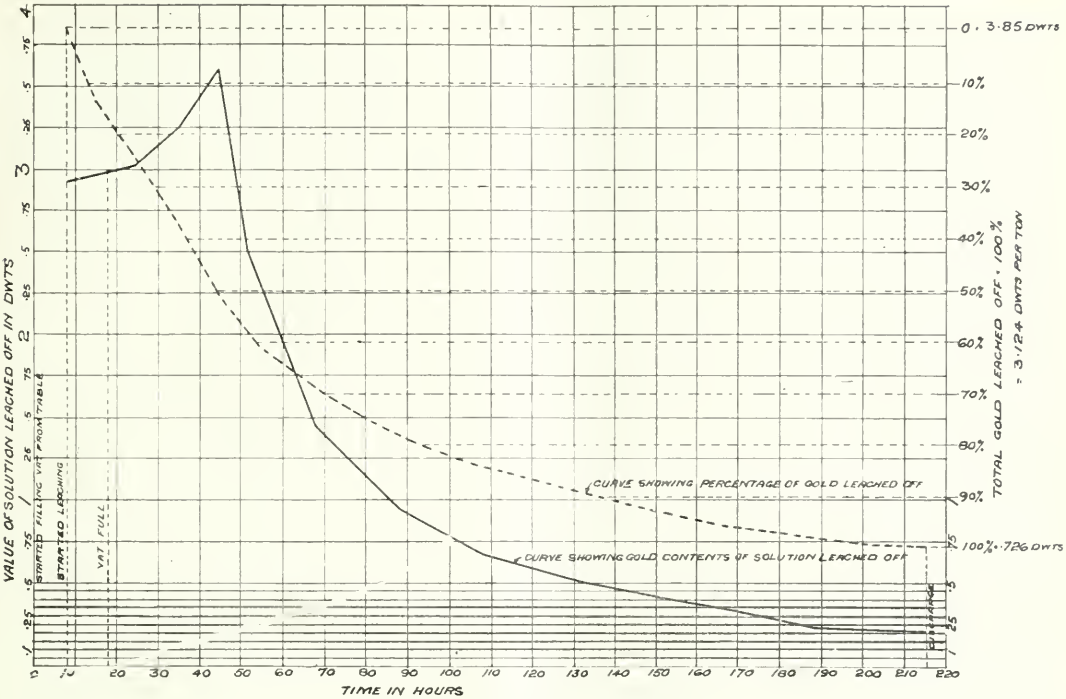


CURVE FOR DOUBLE TREATMENT.

Average of Vats, D_2, D_4, D_5, D_7, E_2, E_4 and E_7.

Weight of Charge	247·14 tons
Assay Value of Sand leaving Table	3·803 dwts.
Percentage of Moisture in Sand leaving Table	18·8
Ratio of Solution to Solid in Pulp entering 1st Treatment Vat	5·43 to 1
Undissolved Gold in Sand before reaching 2nd Treatment Vat	1·45 dwts.
Percentage of Gold dissolved in Sand before reaching 2nd Treatment Vat	61·8
Total Residue	·778 dwts.
Undissolved Gold in Total Residue	·714 dwts.
Percentage Extraction	79·59
Tons of Solution applied per ton of Sand...	1·987 tons
Grading Analysis of Sand from Table :—	
+60 (·010 in.)	28·24
−60 (·010 in.) + 90 (·006 in.)	32·69
−90 (·006 in.)	39·07%
	100·00%
Percentage of Slime	0·70%

FIG. II*a*.—Diagram showing rate at which gold is leached from sand with double treatment.



CURVE FOR SINGLE TREATMENT.

Average of Vats, D_1, D_3. D_6, E_5, E_1, E_3, E_6 and F_8.

Weight of Charge	245·00 tons
Assay Value of Sand leaving Table	3·85 dwts.
Percentage of Moisture in Sand leaving Table	18·00%
Ratio of Solution to Solid in Pulp entering 1st Treatment Vat...	4·38 to 1
Undissolved Gold in Sand after Charge has been collected and drained	1·695 dwts.
Percentage of Gold dissolved in Sand after Charge has been collected and drained	55·96%
Total Residue	·726 dwts.
Undissolved Gold in Total Residue	·661 dwts.
Percentage Extraction	80·120%
Tons of Solution applied per ton of Sand	1·99 tons
Grading Analysis of Sand from Table :—	
+60 (·010 in.)	28·32%
−60 (·010 in.)+90(·006 in.)	33·31%
−90 (·006 in.)	38·37%
	100·00%
Percentage of Slime	0·95%

FIG. IIb.—Diagram showing rate at which gold is leached from sand with single treatment.

produced during the last five months, up to 2,600 tons of ore being milled at times per 24 working hours. The general arrangement of this installation is shown schematically in Fig. III. (see p. 50) The capacity per square foot of filtering area of the Simmer Deep sand filters is about 50% greater than that of the Simmer and Jack filter, owing to the slime being so thoroughly washed out that less than 1% remains in the collected sand. Indeed the whole of the sand product of the mill up to over one ton per minute is regularly handled by one table whilst renewal is taking place in the other of the "permanent bed." This is the term applied to the inch or so of sand on the filter cloth below the plough, which becomes in time compacted and slimy, thus causing the vacuum to rise and reducing the efficiency of the filter. As a rule the vacuum varies between 3 in. and 10 in., and the vacuum air pump and tables each require about 5 h.p. to

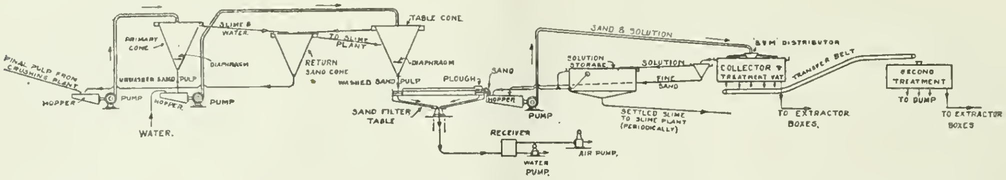


Fig. III.—Diagram of sand plant with pulp elevation between primary and table cones, and belt transfer of sand from collector to treatment vat.

run. The pump drawing water from the receiver in which the vacuum is maintained requires about 3 h.p., as does also that supplying the glands of the centrifugal pump which elevates the sand-solution pulp to the collecting vats. The last pump mentioned is the main source of power consumption and requires 40 h.p. The plough is merely a thin steel sheet with a renewable wearing edge, and is capable of being raised and lowered so as to periodically remove the top compacted layer of the permanent bed. The maintenance on the slow moving tables (about one revolution in three minutes) is practically nil, though wear on the centrifugal pump in handling even somewhat fine sand takes place.

The recent remarkable development on the Rand of centrifugal pump elevation of pulp has greatly assisted in developing the method of solution-pulp transfer, and obviously the same plan could be adopted for transferring sand from the collectors to separately placed second treatment vats, but for new plants the superimposed vat system (Fig. V.) (see p. 52), when only shovelling down is needed, is still simpler and cheaper. The solution required for pumping the sand as pulp is about four or five by weight to one of sand; the ratio is kept as low as is compatible with the grade of the delivery launder or pipe. Whilst collection is still proceeding, and before the vat is filled, the leaching off of solution from the vat is begun, and after the vat is filled as much more wash solution is applied as there is time for before transfer, with the result that half to two-thirds of the gold content of the sand is in the zinc boxes before transfer begins, and before treatment would commence at all under ordinary practice. Before delivering the sand-solution pulp to a collector the latter has three or four feet of precipitated sand solution pumped into it, the exact amount being regulated by that which is regularly withdrawn from the solution transfer circuit, in the shape of solution leached from the collectors for precipitation: in this way an equilibrium of the stock of transfer solution is maintained, and at the same time its gold value is kept low, being usually under 0·2 dwts. per ton. It is found that the best work is done and driest sand obtained with the lowest vacuum; owing to the greater volume of air being drawn through with a more porous permanent bed; when this has lately been renewed, the moisture in the sand removed by the plough is about 13%, and this percentage slowly rises up to say 19·5% immediately before the next renewal. In the Simmer Deep installation only one air pump is at present installed for the two tables, with the result that that filter table which happens to have the more porous bed has the bulk of the air drawn through its sand, to the detriment of the other filter. A separate air pump will now be erected for each filter table, and by this means it is anticipated that a still more perfect removal of moisture from the sand on the filter will be effected. In a sand filter installation a complete spare unit comprising cones, filter, receiver, vacuum and water pumps is advisable, so that the renewal of permanent bed and periodical overhauling of pumps can take place at leisure without taxing the capacity of the remaining units.

The slime overflowing the sand collector carries with it a small amount of fine sand, as is usual with a distributor, especially when the vat is nearly filled with sand. This sand may be intercepted on its way to the solution storage by means of one or more cone classifiers, whose underflow gravitates back to the sand-solution pump. As, however, the accumulated slime in the bottom of solution storage is periodically pumped to the slime plant with solution, the fine sand may be removed at this stage by cones if preferred.

The following are the costs of operation per ton of ore milled of the Simmer Deep-Jupiter sand filters for the last two months (June and July), with an average monthly tonnage of 70,883 tons of ore:—

White Wages ...	... 0·369 pence.
Native Wages ...	... 0·046 ,,
Workshops ...	... 0·014 ,,
Stores	.. 0·123 ,,
Power ...	... 0·387 ,,
Compound ...	... 0·035 ,,
	0·974 pence.

Many members present are acquainted with the features of the Simmer Deep-Jupiter installation* (Photograph D.) and its operation, so I need

---

* See this *Journal*, May, 1909, p. 402 et seq.

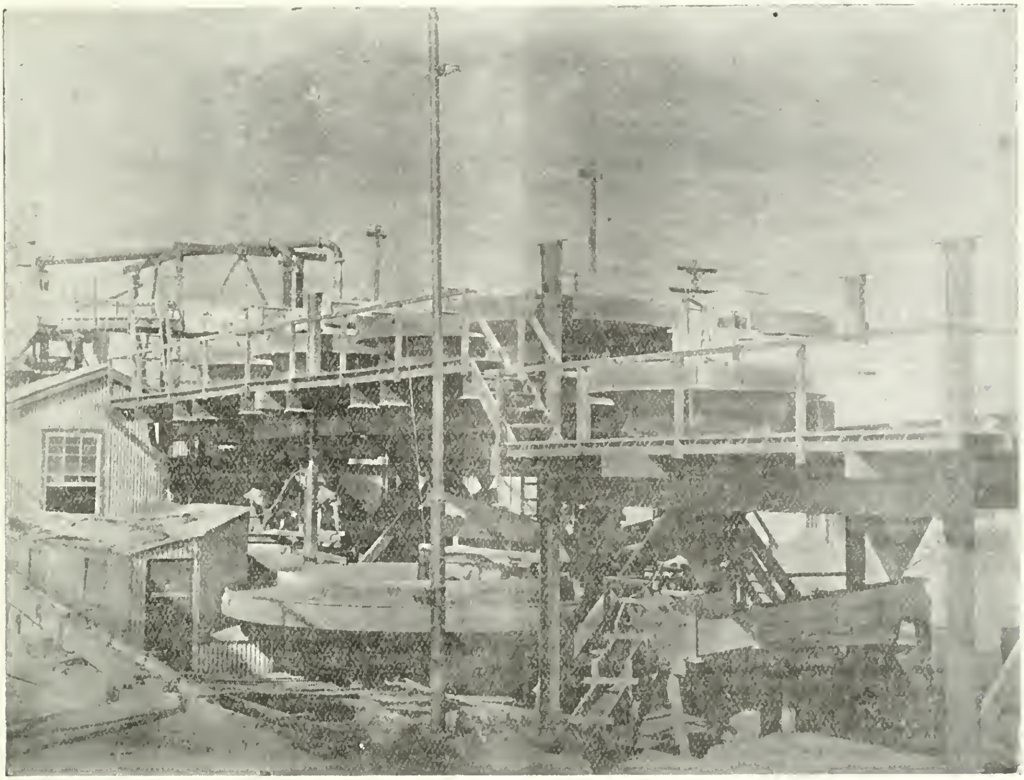


PHOTO D.—20 ft. sand filter tables erected at Simmer Deep-Jupiter joint plant (1909).

hence only recall the uniform granular sand product free from slime or lumps, which is so readily leached and aerated in the vats. The difference between a washed and unwashed sand residue or the gold content of washings from a residue sample is hardly appreciable, thus evidencing the satisfactory dissolving and leaching out of the accessible gold. The Simmer Deep-Jupiter sand plant was originally designed on the ordinary lines with separate collectors and leachers and belt transfer. As erection was begun in the early stages of the sand filter trials the only change made during erection was omission of half the leaching vats, so that the plant consists of ten collectors and ten leachers of 740 tons each. Even these are not all required for treatment with the sand filters, two being used for solution storages and one as an emergency vat in case of breakdown of the sand filters. Such an event has not yet occurred, and by adding two more tables, six more sand vats, and two solution storages it will be possible without difficulty to handle the sand from 120,000 tons of ore monthly in 26 sand vats in place of the 48 otherwise required under ordinary conditions.

One somewhat unexpected result of the cone classification comprised in the sand filter installation is the appreciable increase in the slime tonnage, which may be taken at about 40% of the weight of the ore in place of say 32% otherwise to be expected. This is mainly due to the separation as slime in the conical classifiers of – 200-mesh sand.

The combination shown in Fig. IV. (see next page), though simple and saving height in both elevations of the pulp, cannot be considered a desirable one because the unwashed sand pulp delivered upon the filter table reduces the capacity of the latter considerably, and the corresponding less perfect elimination of slime does not yield so porous a sand product in the single vat, in which both collection and treatment are carried out without further transfer of sand. Unwashed sand pulp is handled by the Simmer and Jack table with subsequent truck transfer of sand from collector to another vat, and yields the results shown elsewhere. The increased time for treatment in a single vat required for even washed sand is likewise referred to in another portion of this paper.

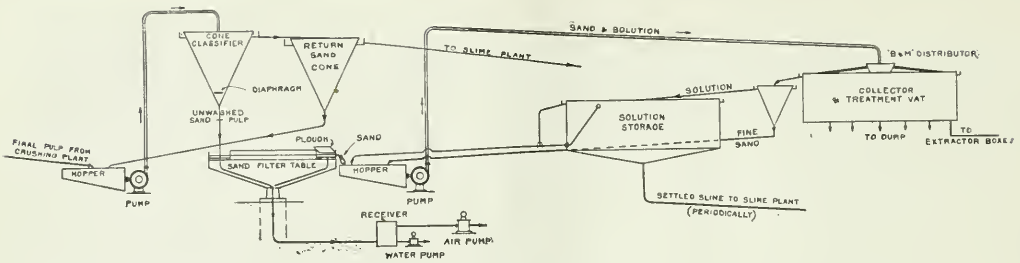


Fig. IV.—Diagram of sand plant with delivery of unwashed sand pulp to filter table, and with collection and treatment of sand in same vat.

The schematic arrangement in Fig. V. shows what is in the writer's opinion at present the most suitable type of plant with sand filter installation. The superimposed vat system under these conditions is no longer open to the objection of contaminating mill service water, whilst all the vats both upper and lower are utilized for treatment, and the cost of installing and operating transfer belts or trucks is done away with.

The collecting vats used to receive the almost slime-free sand solution pulp may be of larger dimensions than usual, as settlement of slime in layers is not to be apprehended, and fewer larger units thus employed in place of more smaller ones. For the same reason the bulk of the lime used may be crushed with the ore in the battery, thus reducing the separate grinding of lime in a ball-mill or other machine to occasional periods when unusually acid ore is being delivered to the battery. Naturally the use of sand filters greatly increases the capacity of existing plants whether of the superimposed or other type, by converting all the collectors into treatment vats.

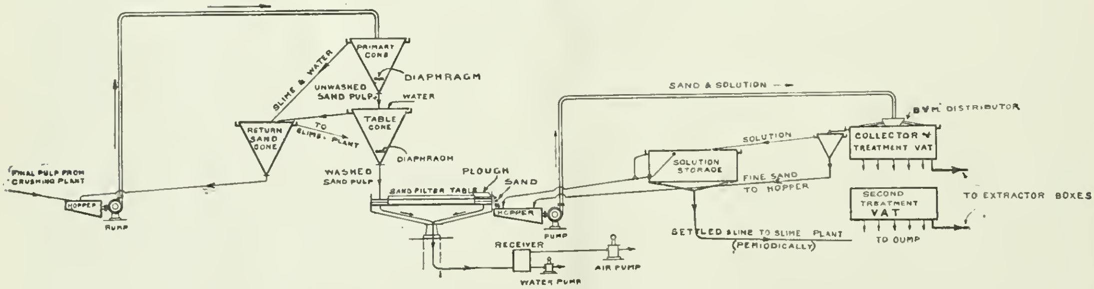
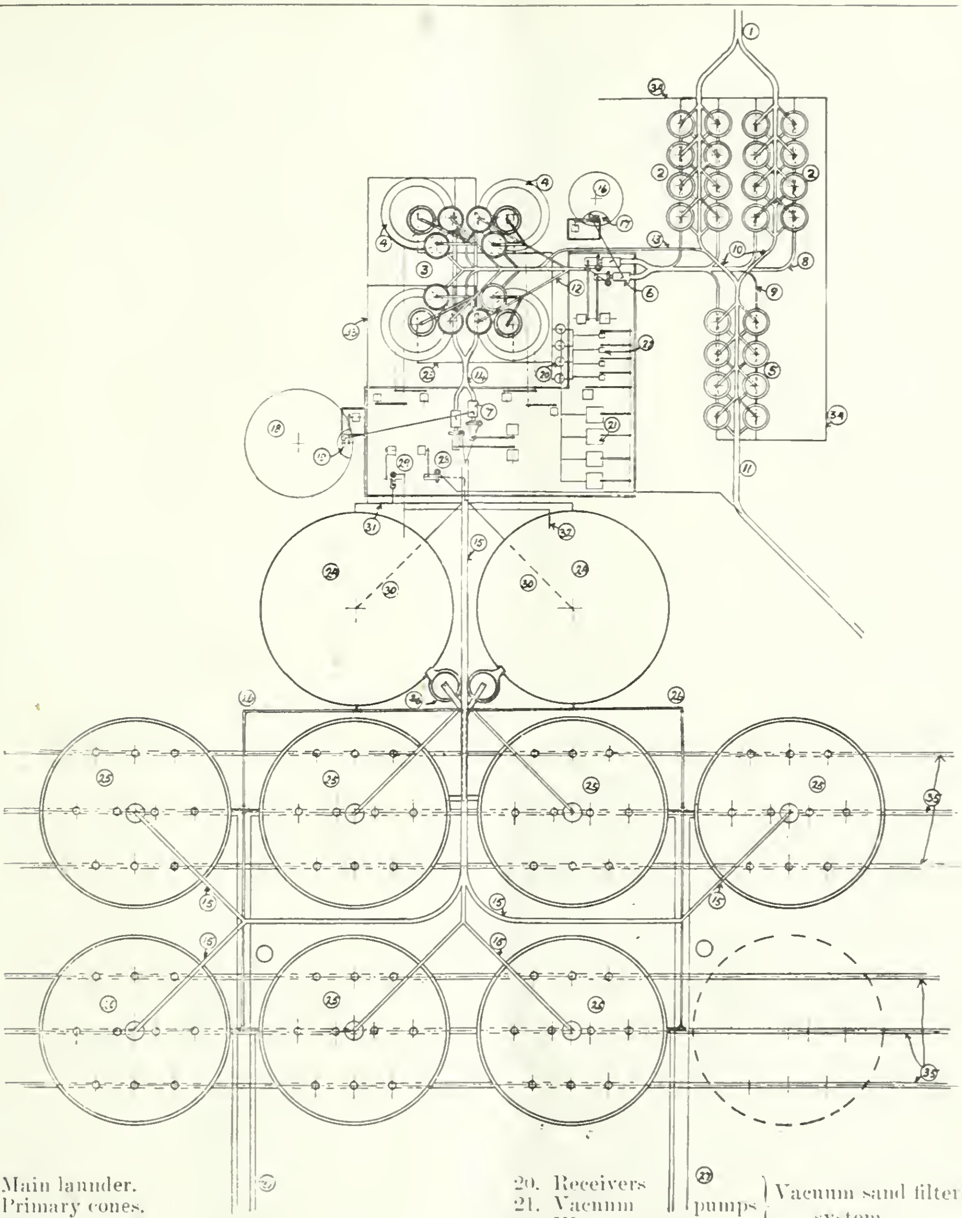


Fig. V.—Diagram showing main features of VI. in elevation, with superimposed primary and table cones as well as superimposed collecting and second treatment vats.

Schematic arrangement VI. (see p. 53) illustrates in plan, of which Fig. V. shows the main features in elevation, a design for a plant handling the sand from 5,350 tons of ore per day, and a summarized estimate of the cost of such installation, including a complete spare filter unit, is as follows :—

4 Filter Tables and Launder complete	£3,048	10	0
16 Primary Cones	2,334	0	0
12 Washing Cones	1,682	5	0
12 Return Sand Cones ...	1,863	0	0
4 10 in. Sand Pumps complete	2,246	0	0
Pumps, etc., for Filter Tables...	1,939	0	0
Sumps, etc.	836	0	0
Solution Storages	3,136	0	0
Treatment Vats and Structure	23,156	0	0
Main Pump House and Floors	1,035	0	0
Gangways and Stairs ...	300	0	0
Electrical Work	400	0	0
Painting	200	0	0
	£42,175	15	0
10% contingencies ...	4,217	11	6
	£46,393	6	6

The following is an estimate of a plant on the ordinary system to deal with sand from the same ore tonnage, one-third of the vats being allocated for collection and the remaining two-thirds for treatment purposes; the capacity of the latter affords the same time of circuit as do the total sand vats with continuous collection by means of sand filter tables.



1. Main launder.
2. Primary cones.
3. Washing cones.
4. Vacuum sand filter tables.
5. Return sand cones.
6. Sand pumps from primary cones to table cones.
7. Sand pumps from tables to vats.
8. Underflow from primary cones to pumps "6."
9. Underflow from return sand cones to pumps "6."
10. Overflow from primary cones to return sand cones
11. Overflow from return sand cones to slime plant.
12. Launder from pumps "6" to table cones.
14. Sand and solution launders from tables to pumps 7
15. Distributing launders from pumps "7" to vats.
16. 20 ft. overflow sump from pumps "6."
17. Pump for emptying sump "16."
18. 40 ft. overflow sump from pumps "7."
19. Pump for emptying sump "18."
20. Receivers
21. Vacuum pumps } Vacuum sand filter table system.
22. Water pumps
23. Vacuum pipes from tables.
24. Solution storages.
25. 70 ft. collectors with treatment vats below.
26. Overflow from collectors to solution storages.
27. Leaching pipes.
28. Sludge pump for storages.
29. Solution pump for hydraulicking.
30. Sludge suction from bottom of storages.
31. Solution suction.
32. Hydraulicking service.
33. Solution to table launders.
34. Water service to underflow from primary cones and return sand cones.
35. Tracks to dump.
36. Fine sand cones

FIG. VI.—Diagram showing general arrangement in plan of sand plant with sand filter table installation.

Collecting and Treatment Vats ...	£14,000
Super and Substructure	37,000
Erection of Vats and Structure ...	6,500
Excavations	34
Foundations	750
Leaching Piping (erected)	850
Launders to Collectors	1,700
Filter Mats and Woodwork ...	1,750
Return Sand Spitzkasten	1,500
Transfer Tracks, Collectors to Treatment Vats	800
Discharge Doors for Vats	1,800
Tracks under Vats, etc. ...	500
Solution Piping (erected)	1,100
Gangways, Stairs, Painting, etc. ...	500
	68,784
+10% Contingencies ...	6,878
	£75,662

For both these estimates I am indebted to Mr. G. H. Thurston, and the comparison shows a very considerable saving in capital expenditure to be effected by means of the sand filter installation.

The advantages which sand filters present are considerable saving in capital expenditure both for sand vats and for belt or truck transfer installation, slightly better extraction than the ordinary system unless prolonged treatment is given, a saving with superimposed vats of belt or truck transfer cost, and more slime to be treated as such at, say, 6d. per ton less than sand in vats of slightly increased diameter. Evidence as to reliability in practice and low cost of operation has been given, and considerable extension of continuous sand collection both for new plant and for increasing the capacity of existing plants is now being designed or is actually in process of construction, so that within a year the present fifty odd thousand tons of sand handled monthly by this method on these fields is likely to be doubled or trebled.

In conclusion my thanks are due to the Consolidated Goldfields, to whose enterprise is due the prosecution of the long continued trials with sand filters at very considerable cost: to the East Rand Proprietary Mines and Anglo-French Corporations, who contributed substantial amounts towards the cost of the experimental work: and to Messrs. G. H. Thurston, W. R. Dowling, G. O. Smart, J. E. Thomas and C. O. Schmitt, whose energy and skill have so materially assisted in carrying this development to a successful issue.

Mr. John R. Williams (*Past-President*): It affords me great pleasure indeed to propose a hearty vote of thanks to Mr. Caldecott for his very able paper in the first place, and much more so for the great work he has undertaken in bringing this sand filter to a most successful issue. Since my return to the Transvaal, I have had the opportunity of seeing this plant in operation, and I must say that the feeling I had on seeing the sand and the clean tanks was that of wishing I was again running a cyanide plant myself. Never before, outside laboratory experiments, had I seen sand in that thoroughly clean state. It should give the highest possible extraction, and I think, at a minimum cost. I do not think I should care to criticise Mr. Caldecott's paper to-night, but it has given me a great deal of pleasure to listen to it, and also to have seen the plant working.

Professor J. Yates (*Past-President*): I would like to second the vote of thanks. It is a coincidence that to-day I took quite a number of students round the plant, and I must say that every one of us were greatly impressed with the novelties which have been introduced. It is just as well to point out, in connection with these novelties, that it is no use having the idea unless you have the courage to apply it, and no one can go round that plant without acknowledging that Mr. Caldecott and his staff have not only got the ideas, but they have had the courage to carry them out.

NOTES ON CORROSION, WITH SPECIAL REFERENCE TO THE CORROSION OF STEEL WINDING ROPES.

By M. Thornton Murray, M.Sc.

Before dealing with the special aspect of corrosion set forth in the title of this paper, it will be necessary briefly to summarise the theories and ideas of corrosion, which are at present in vogue.

By far the greater number of the cases of corrosion of iron and steel which have come under the author's notice, have been cases of "rusting," and the present one forms no exception to the rule: in the corrosion of steel ropes, rust—hydrated oxide of iron—is produced, and the corroded areas are typical of those found in other cases of rusting. The old theory of rusting, which was, until a few years ago, almost universally accepted, was due, in the first place, to Crace Calvert, but was extended by Prof. Crum Brown in 1888, while it has since been added to by Jowett, Dunstan and others. For the sake of distinction, this might be called the purely *chemical* theory. It asserts that the essentials for the formation of rust are:—Iron, or steel, oxygen, carbon dioxide, and *liquid* water (not water vapour).

The equations representing the reactions are as follows :—

Carbon dioxide and water yield carbonic acid

(i) $CO_2 + H_2O = H_2CO_3$

Iron and carbonic acid yield iron "bi"-carbonate, and hydrogen

(ii) $Fe + 2H_2CO_3 = FeH_2(CO_3)_2 + H_2$

Iron "bi"-carbonate and oxygen yield hydrated ferric oxide ("rust"), carbon dioxide and water

(iii) $4FeH_2(CO_3)_2 + O_2 = 2Fe_2O_3.3H_2O + 8CO_2 + H_2O$

The carbon dioxide does not escape, but remains in the water, and is used over and over again. The action, so far as the carbon dioxide is concerned, is cyclic, and this explains the fact that, although carbon dioxide is necessary for the reaction, such a relatively small proportion as is present in ordinary air, is sufficient to do so much damage.

For some time the "chemical" theory, as enunciated above, stood unchallenged ; then, quite recently, it has been assailed from all sides. Dunstan and Traube first asserted that hydrogen peroxide (H_2O_2) was formed during the reaction, and immediately broken up again, thus :—

Iron and oxygen and water yield ferrous oxide and hydrogen peroxide

(i) $Fe + O_2 + H_2O = FeO + H_2O_2$

Iron and hydrogen peroxide yield ferrous oxide and water

(ii) $Fe + H_2O_2 = FeO + H_2O$

Ferrous oxide and hydrogen peroxide yield hydrated ferric oxide (rust)

(iii) $2FeO + H_2O_2 = Fe_2O_3H_2O$ (or $Fe_2O_2(OH)_2$)

Mallet asserted also that magnetic oxide of iron (Fe_3O_4) was intermediately formed during the reaction, whilst others vary the formula in slight details. Next, Dunstan endeavoured to show that carbon dioxide, or, indeed, any acid, or acid-making gas was unnecessary for the formation of rust. He asserted that water and oxygen by themselves would slowly corrode iron with the formation of the well-known hydrated ferric oxide. This assertion paved the way for the boldest theory which has yet been advanced—that due to Dr. Allerton S. Cushman, of the Department of Agriculture, at Washington, U.S.A.* Undoubtedly Dr. Cushman's paper is one of the finest contributions ever made to the literature of rust. He began by casting doubts upon Dunstan and Traube's theory that hydrogen peroxide was intermediately formed during rusting. If the formation of this compound was an essential stage in the formation of rust, why was it that rusting could not be prevented by powerful reducing agents ? Since the weakest "reducer" was strong enough to break up hydrogen peroxide into water and oxygen.

Next he demonstrated, by means of experiments, that pure iron would partially pass into solution in pure, air-free, distilled water, without the presence of any other dissolving agent. Having established this point as a preliminary, he puts forward his theory of the rusting of iron. This is as follows : —

When iron is immersed in water, the water is broken up into hydrogen and hydroxyl "ions." Now hydrogen is electrically classed as a metal. Therefore when iron is brought into contact with a solution containing hydrogen ions, iron "ions" pass into solution, and hydrogen ions collect on the iron ; the action is analagous to the deposition of copper on iron which is dipped into a solution of copper sulphate. Furthermore, hydrogen is electro-negative to iron, so that the hydrogen "ions" actually set up an electromotive force (analagous to the back E.M.F. in a polarised simple cell) with the iron. The action, thus set up, is prolonged. Iron ions pass into solution (and, in the presence of oxygen, are immediately oxidised). Hydrogen ions are ousted from the water to give them room, and are given off, while hydroxyl ions collect at certains points on the iron. The function of acids, in rusting, is, therefore, simply to strengthen the electrolyte and promote ionisation.

Since Dr. Cushman's paper, the electrolytic theory of rusting has been much discussed, and papers have been brought out by Tilden, Friend, Moody, Walker and others, espousing either the chemical or the electrolytic cause. The publication of Dr. Cushman's paper, by the Iron and Steel Institute has, however, firmly established the electrolytic theory—at least in the main.

Whatever be the merits or demerits of the electrolytic theory of rust formation, it is certain that electrical phenomena are very closely bound up with the corrosion of iron and steel. Steel and iron water-pipes, when laid in the vicinity of leaky electric cables, are corroded with extraordinary rapidity, so that exceptional insulating precautions are now taken, when cables are laid near such pipes. It has long been known that rust itself, in the presence of a suitable electrolyte, sets up a difference of potential with clean iron and steel, and it is probably this peculiarity which partly causes an already rusted surface to continue to rust so much more rapidly than a previously bright one. Even when the surface is covered with some rust resisting material—either paint, tar, or another metal—and apart altogether from the adhesive power which a clean surface possesses, it must be borne in mind that rusting will proceed even *under* the protector, if the protected surface be already rusted when covered.

---

* Bulletin 30, Dept. Agriculture, Washington, D.C., U.S.A.

Besides the electrolytic action set up between clean iron and "rust," a very much more serious action is set up when clean iron and black oxide of iron, or "smithy scale" are placed in electrical connection in a suitable electrolyte. In this latter case the iron serves as the anode, or positive pole, while the oxide acts as a cathode. Thus the iron is dissolved away, and precipitated (after oxidation) upon the "scale." In the case of ordinary rust and pure iron, the action is reversed, the rust being dissolved and re-precipitated upon the iron. A somewhat striking illustration of the action of magnetic oxide upon iron corrosion is afforded by the fact that upon painted girders have been found little piles of rust, emanating from a mere pinhole in the paint, which is almost invariably situated over a tiny, embedded particle of "scale."

Mr. E. F. Law, in his paper on "Non-Metallic Impurities in Iron and Steel,"* gives another example. He conducted experiments on welded bars, and found that rusting took place much more rapidly at the weld than elsewhere. This he attributed to the presence of magnetic oxide of iron ("smithy scale") at that point.

The action upon iron and steel of other metals, with respect to corrosion, is yet another proof of the existence of some electrolytic action, in corrosion. The metals in the following table, which are electro-positive, will, if placed in contact with iron tend to dissolve away, and reprecipitate upon the iron, thus protecting it. The protective action of galvanising is thus explained. Metals, however, on the negative side, tend to aid in the destruction of the iron, since, if they form a covering, and that covering is once broken through, and the iron exposed to rusting influences, an electrolytic action is set up, and the iron is dissolved away, to be re-deposited upon the covering metal, after it has been oxidised.

Electro-Positiveness of Metals.—

Positive End

Potassium
Sodium
Magnesium
Manganese
Zinc
Iron
Cadmium
Cobalt
Nickel
Lead
Tin
Bismuth
Copper
Mercury
Silver
Platinum
Gold
Hydrogen
Antimony
The Metaloids and Non Metals

Negative End

Mr. Law has demonstrated the influence of non metallic "impurities" on the corrosion of iron and steel. But it seems to the author that one might go farther, and assert that, under certain conditions, the very constituents of the iron or steel themselves—such as graphite, ferrite, pearlite, cementite, etc.—to which our President has introduced those of us who were not previously familiar with them—that these are capable of promoting corrosion by mutual electrolytic action. The author was first led to this belief by an investigation of the well-known variety of iron or steel, called "double-faggoted iron," which consists, as you know, of alternate layers or "fibres" of wrought iron and mild steel. Under the microscope this specimen exhibited a structure consisting of pure ferrite, and ferrite with small areas of pearlite. Now this specimen was exposed to damp air for some months, as was a piece of wrought iron and a piece of mild steel, of the same composition as the steel in the composite bar. It was found that the composite bar was corroded very much more than either the mild steel or the unwrought iron, and the microscope further demonstrated that the rusting began where the pearlite areas of the steel touched the ferrite of the pure iron. In the steel itself, there was, of course, little evidence of what might be called "intra-granular corrosion," but that was due to the natural and stable contact between the ferrite and the pearlite, attained when the steel was melted, in its manufacture, and cooled. The incomplete junction of the pearlite of the steel and the ferrite of the wrought iron, however, led to the establishment of differences of electrical potential, and consequent rapid corrosion.

A number of samples of steel which had been carbonised to a certain depth and hardened—in other words, "case-hardened"—came under the author's notice. These were fractured so as to expose both the high-carbon hard "casing," and the mild steel core to rusting influences. It was found that corrosion proceeded very much more rapidly than upon a high-carbon, hardened steel, or a mild steel, such as constituted the core. From these, and a number of similar examples, it is possible to assume that lack of uniformity—either in carbon content, in distribution of the constituents, or in the contact of the crystals—tends to promote corrosion. This aspect of corrosion has been dealt with at some length, for reasons which will appear later, when the corrosion of winding ropes is discussed.

Corrosion of Winding Ropes.

For a great deal of the information upon which the argument of the following part of the paper is based, the author is indebted to Mr. Hume, of the Mechanical Laboratory of the Mines Department.

For a variety of reasons, steel cables offer peculiar opportunities for corrosion. Firstly, the atmosphere in a mine shaft—especially in the upcast shaft—generally contains large proportions of "acid" gases, such as carbon dioxide, oxides of nitrogen, etc., together with some water vapour, while the air is generally warm. Secondly, the constructional nature of the cable itself is such as would assist corrosion. It is well known that the presence of accually *liquid* water is essential for the production of rust. In confined spaces, therefore, where evaporation cannot rapidly take place, the rusting action is both prolonged and intensified. Such confined spaces are the interstices between the strands or wires of a steel cable. In many cases cables have been discovered to be internally corroded, when the surface was but little affected. These examples were doubtless due to the phenomenon above described, the cable, at one time or another, having taken up water which could not evaporate easily, even when it was transferred to conditions where evaporation from the surface might have been possible.

---

* *Journal Iron and Steel Institute*, vol. ii., 1907, p. 104.

There is a third reason, the author believes, for the particular aptitude of winding ropes to corrode, and that is, the continually alternating or varying stresses which they have to withstand.

It would be a work of supererogation for the author to point out to the members of this Society how extensive are the ravages of corrosion upon winding ropes, but an examination of Fig. 1 may be of interest. This is a photograph of a wire from the outside of a rope which had been in service some two years. It will be seen that the wire is, in some places, only about one-quarter of its original thickness. Part of this reduction is undoubtedly due to wear, but in this case many of the internal wires, which had been worn but little, were reduced by corrosion by more than 50% of their original diameter

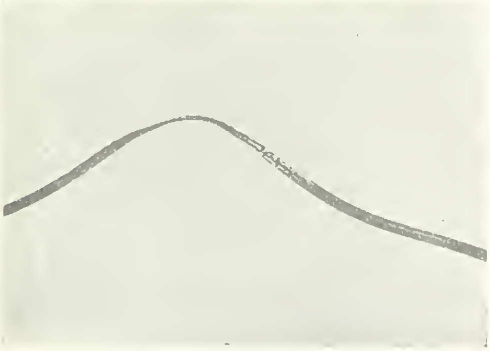


Fig. 1.—Corroded wire.

A noticeable point in connection with the corrosion of winding ropes, is that owing to the continual friction they are subjected to in passing over pulleys and sheaves, the surface is generally kept fairly bright, and evidences of corrosion are consequently not easily discovered, though the rope is corroded all the same.

Perhaps the most interesting aspect of the corrosion of winding ropes is one which is very closely allied to the mechanical effects of their usage upon the internal structure of the metal. At some future date it is hoped that the results of metallographic investigation into this question will be placed before the Society; the scope of the present paper is not wide enough, however, to cover this question, and it will only be dealt with incidentally.

Most of the members will be familiar with the "flattening" effect of continual friction upon the outer wires of a rope. They will have noticed that not only has the wire been worn away by continued attrition, but that it is, in many cases actually *flattened*. In one case a wire, which had been in service only nine months was flattened out at one place until its diameter had increased $10\frac{1}{2}\%$. Furthermore, the wire at such points exhibits remarkable brittleness, so that it may often be broken off short in the fingers when bent with the flattened surface uppermost. Often, too, numbers of transverse cracks have developed on the flattened surface, some of considerable depth and magnitude. Indeed these are only to be expected, for the material, under the microscope, exhibits all the features of case-hardened steel. That is, there is a thin coating, or case, of dead-hard, brittle steel, while the inner core is still soft and ductile.

Fig. 2 shows the depth of this casing, together with one or two minute cracks running through it. The magnification of this photograph is some 1,200 diameters, and this high magnification, taken in conjunction with the difficulty of obtaining a flat polished surface on so small a specimen makes the photograph somewhat indistinct, though it is sufficient to show the contrast between the hard, difficulty-etched "skin" and the soft grey pearlitic core.

Fig. 2.—Magnification =1200 V.
Edge of wire, showing hard "skin."

Fig. 3 is a photograph (see p. 58) taken, at a magnification of some 80 diameters, of the ordinary unpolished surface of a flattened wire, shewing

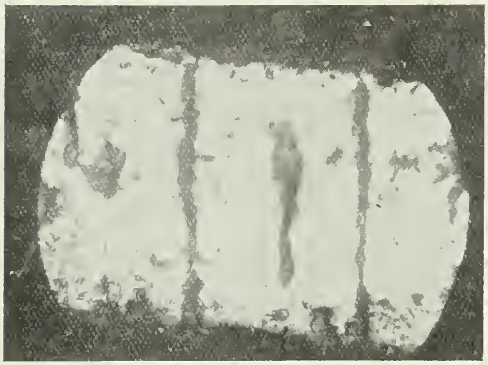


FIG. 3.—× 80, Oblique.
Transverse cracks on the surface of a wire.

the transverse cracks. Fig. 4 is a photograph of a longitudional section of the wire, taken at a magnification of some 800 diameters, shewing the *depth* of the cracks. It will be noticed in connection with this crack that corrosion has already begun in it. The edges have formed the starting point for corrosion, which has spread longitudinally, in each direction. This corrosion is due to two causes. Firstly, the cracks, gathering and *holding* moisture as they do, out of access to the air, and consequently not easily evaporated, tend to promote corrosion. Secondly, the difference of potential set up between the hard outer casing and the soft interior, materially aid in destroying the steel. In this action, too, the transverse cracks carry the liquid which forms the electrolyte. But after the " skin " has been corroded away, or in cases where the skin is not formed, the action proceeds somewhat differently. In most of the specimens which have come under the author's notice the pitting seems to have followed the *length* of the rope in direction (see Fig. 1).

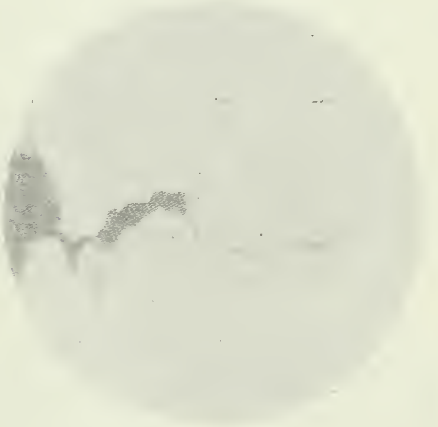


FIG. 4.—× 800 V.
Longitudinal section of wire, showing depth of crack.

This is well shewn in Fig. 5, which shews the commencement of corrosion *on the surface* of a wire. The corroded areas are elongated (they are the darker areas on the specimen), and are interlaminated with the pearlite and ferrite bands.

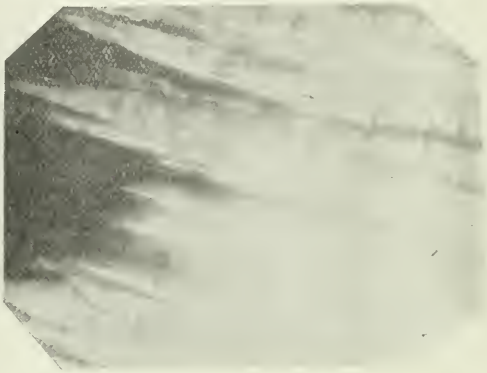


FIG. 5.—× 800 V.
Corroded surface of wire (dark areas corroded).

Fig. 6 shews a micro-section of an ordinary good wire, containing (from its appearance) about ·8% Carbon, which has never been in service. It will be seen that the structure is composed, for the greater part, of pearlite, while there are a few small areas of ferrite. The banded structure is probably due to the wire having been finished at a temperature a little below its critical point, but the " cold-working " has not been pronounced

FIG. 6—× 800 V.
Microstructure of good wire (pearlite (dark), and ferrite (light).

enough to detract from the physical properties of the metal.

Fig. 7, on the other hand, is a micro-photograph of the inner portion of a wire which has been in use for some time, and has been subjected to very hard usage. This wire, when tested, gave a very low breaking strain, and was more or less brittle throughout. It will be noticed that the pearlite areas are farther apart, and that the ferrite areas have become larger, and have formed irregular "lakes." This structure is the result of segregation, due to severe strain and to undue heating during work. This wire was very badly corroded in parts, with the longitudinal corrosion lines very marked. It is only reasonable to suppose, then, that the strains which produced this segregation also forced the crystals somewhat apart from one another, with the result that the contact between the pearlite and ferrite was rendered imperfect. Between these two constituents, as a consequence, differences of potential were set up, and the metal corroded very rapidly.

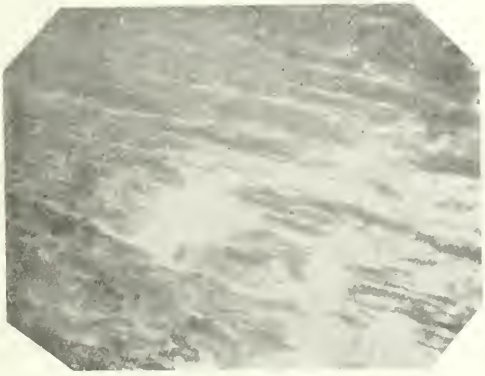


FIG. 7—× 800 V.
Stranded wire, showing segregation.

Briefly to summarise the question of the corrosion of steel winding ropes; undue strains, caused generally by careless usage, tend to produce internal strains, which promote corrosion, while the surface hardening effect and consequent formation of transverse cracks materially shortens the life of the rope, not only by impairing its mechanical properties but by assisting corrosion in the manner which has been demonstrated.

To a certain extent these evils may be mitigated; firstly, by bestowing a greater amount of care on the ropes. Especially is this care needed in the case when the ropes are employed on *compound* shafts, for the greatest trouble, both by hardening and corroding, has been found to occur on such ropes. It has been proved, in this case, that the bend-pulleys are responsible for the greater part of the damage; the friction generated when the rope is running over the pulleys being often sufficient to produce actual sparking. It is obvious that under these conditions even good ropes will not last for long. Secondly, the corrosion can be to a great extent arrested by very careful lubrication. Too often this duty is performed at long intervals of time and then inefficiently, Both on the score of the danger to life and limb incurred by such neglect and of the money saved by careful attention to the rope, the author believes that these little points would reward attention.

In concluding these notes, let it be understood that the author is offering no universal panacea for corrosion; he is merely placing before the Society the results of his investigations into the question, and he trusts that they will be of interest to the members.

The President: In proposing a vote of thanks to Mr. Murray for the excellent paper he has placed before us, I feel very pleased that I was able to give you a sort of introductory to the subject at the last meeting. I can only say that the illustrations which he has shown and the results he has placed before us have been obtained by us also at the Consolidated Gold Fields Laboratory. I have made numerous microscopic examinations of broken wires, and the author's results are practically the same as I obtained. I am sure we are very much indebted to Mr. Murray for the way in which he has presented this matter. It has been brought forward in a straightforward, easily recognised manner, and it is a gain to the Society and to our *Journal* to have such a paper.

Prof. J. A. Wilkinson (*Member of Council*): It gives me great pleasure to second this vote of thanks to Mr. Murray for the paper which has just been read to you, and in doing so I should like, with your permission, to make a few remarks upon some portions of its contents, and a few inaccuracies which have crept in, almost in spite of care which, I am sure, the author has bestowed upon it. In the first place, the theories which have been formulated to explain the chemical changes taking place during rusting have been very briefly stated, namely, the CO_2 theory—I prefer this name to the one used by the author, because I fail to see why this should be called chemical any more than Dunstan and Traube's hypothesis—the H_2O_2 theory, and the electrolytic theory. With regard to the first, why not have written the first equation according to modern practice

$$CO_2 + H_2O \rightleftharpoons H^{\cdot} + HCO_3{}'$$

as we now know this to be the case, and more especially as the acid carbonate is supposed to be formed as an intermediate product to be afterwards decomposed by oxygen? This theory still finds supporters, the English chemists, Tilden and Moody, being, at least until recently, its champions. The former in a recent investigation sums up his results by stating, that he considered the following facts to be established, "The process of rusting is due to electrolytic action with the production of $Fe(OH)_2$ and $FeCO_3$, and may be explained by the presence in the iron of various compounds, which form surfaces of different potential in water or aqueous solution of CO_2, these being the electrolytes, which are necessary for the electrical actions to take place."

As regards the H_2O_2 theory, I know of no modern chemist who champions this, but with regard to the electrolytic theory, I think the author is labouring under a misconception. This theory was first advanced by Whitney in 1903. About the middle of 1907, if memory serves, Dr. Cushman read his paper on rust before the American Society of Testing Materials, and it was shortly afterwards published as Bulletin No. 30 of the Office of Public Roads of the U.S.A. Department of Agriculture. About the same time Dr. W. H. Walker, Miss Cederholm and Mr. Bent read a paper on the same subject before the American Chemical Society, an abstract of which appeared in our columns in December, 1907. Both these papers support Whitney's theory, which was founded on Nernst's conception of solution pressure and the solution theory of electrolytic dissociation. Bearing these facts in mind, I cannot consider it quite fair to ignore the credit due to other workers. Nor do I consider the publication of Dr. Cushman's paper by the Iron and Steel Institute to have firmly established the electrolytic theory, since this problem is entirely a chemical one, and I have yet to learn that this body is the final court of appeal in matters chemical. I agree, however, with the author as to the value of Dr. Cushman's work, but I felt whilst listening to the paper, that his exposition might have been a little fuller as the fundamental conceptions are easy to grasp, and this is one of the few instances which have been presented latterly in this Society, of dwelling upon the theories of modern chemistry as applied to practical problems. Moreover, there is no doubt that the electrolytic or electro-chemical explanation of rusting is now obtaining general acceptance. From the opening statement of the theory, as given by Mr. Murray, one would be led to believe that the iron caused the ionisation of the water molecule, and hence a correction is necessary. Water, even the purest water, now generally spoken of by chemists as conductivity water, is not entirely composed of molecular parts, but does contain some hydrogen and hydroxyl ions. It is on the presence of these ions in the water, that rusting primarily depends, and the equation representing the changes taking place would read

$$Fe + 2H^{\cdot} + 2OH' = Fe^{\cdot\cdot} + 2OH' + H_2$$

the molecular iron passing into solution and becoming diferrion, its solution pressure being less than that of iron. The second action is caused by the oxygen dissolved in the electrolyte water, whereby the diferrion is oxidised to triferrion and finally precipitated, either in absence of air as hydrated ferric oxide, or possibly in presence of ordinary air, first, as carbonate, and finally, as hydrated oxide. If the function of the hydrogen as given above is correct, any substance increasing the concentration of the hydrogen ions will assist rusting. This will serve to explain the function of acids in stimulating the formation of rust, since, as is well known, they dissociate in solution with production of hydrogen ions. On the contrary, alkalis form in solution hydroxyl ions, which unite with the hydrogen ions forming molecular water, and hence alkalis inhibit the corrosion. This explains satisfactorily the non-rusting of steel girders when imbedded in concrete owing to the alkaline character of the latter due to the lime contained therein. Further, the rusting of iron is confined to local spots, a fact easily demonstrated by a low power microscrope, in other words, its solution takes place at some spots and not at others. According to the author this is due to the many and varied constituents of the iron or steel. The non-metallic impurities, as demonstrated by Mr. Law, are merely mentioned, but much of what the author expressed orally, whilst reading the paper, might have found a place in the printed copy, as this would have elucidated very considerably the main points upon which much stress is laid later on, namely, the difference of potential, which must exist between the various carbides of iron and the iron itself.

For the moment, and also as the hour is somewhat late, I do not desire to make any further observations on this paper, which deals with one of the most important phases of mining work and which, therefore, should lead to an ample discussion. We are grateful that Mr. Murray has given us this opportunity, and I am sure that this vote of thanks will meet with your hearty approbation.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

(*Read at September Meeting, 1908.*)

By Robert Linton (Associate).

REPLY TO DISCUSSION.

Mr. Robt. Linton (*Associate*): In replying to the discussion of my paper, I desire to thank the various members who have taken part in it for their interesting contributions. It was in hope of stimulating such a discussion, and especially to enable comparisons to be made between the conditions under which the Rand experiments with zinc dust proved unsuccessful and those which prevail where zinc dust is satisfactorily used on this continent, that the paper was submitted. Mr. Clark's paper adds valuable information on the subject, and answers conclusively some of the questions raised in the course of the discussion.

Mr. Pearce and Mr. Graham describe quite fully and clearly the trials at the Geldenhuis Deep. I have read these accounts carefully and with especial interest, and I must say that I fail to find in the conditions as described any satisfactorily definite explanation as to why zinc dust should not give any better results than were obtained. I would like to suggest that in order to be entirely conclusive the experiments should have been continued beyond the point of determining whether or not zinc dust could be successfully used under Rand conditions, and should have as well definitely located the reason for the negative results. In other words, should have proved whether the difficulty was one that could be remedied by some slight modification of existing conditions, or whether the adjustment involved too great a departure from present methods. It is just this question that Mr. Pearce wants to have answered ; but it appears to me that it can only be answered by further experiments. These experiments, it seems to me, should have especial reference to the matter of silver contents of the solutions, the quality of the zinc dust and quantity of lead used in connection with it, clarification of solutions, regulation of feed of zinc dust, and effect of strength and grade of the solutions. Other points would doubtless suggest themselves as the trials progressed.

In zinc dust precipitation I prefer to use both higher grade and stronger solutions than are usual on the Rand. In all-cyanide plants, such as at Cerro Prieto, where there is no amalgamation and the ore is crushed in cyanide solution, the opportunity is afforded of so circulating the solutions as to concentrate them somewhat before precipitation ; allowing the use of stronger solutions and at the same time keeping at a minimum the waste of cyanide in solutions leaving the works. There does not exist therefore the necessity of using the very weak solutions referred to in the discussions. The strengths used at Cerro Prieto were determined by extraction requirements, and are independent of precipitation conditions, except occasionally when starting up the press after a break. If the solutions are both weak and low grade it is then often necessary to add some fresh cyanide to insure the immediate setting up of precipitation.

With the weak solutions used on the Rand, a higher consumption of zinc dust figured per ounce of metals recovered, would naturally be expected. This is in line with my own experience, as shown by averages of three different months' operations :—

Ounce of Au + Ag per ton of solution.	Pounds zinc dust consumed per oz. Au + Ag
0·47	0·59
0·49	0·57
0·70	0·42

From Mr. Clark's figures it appears that 72 oz. of gold and 36 oz. of silver were precipitated by use of 718 lb. of zinc dust, or 6·6 lb per oz. of metals recovered, the solutions containing about ·02 oz. metals per ton. The Rand practice, as stated by Mr. Pearce, shows 1 to 3 lb. of zinc used per oz recovered, the solutions containing ·033 oz. per ton.

Another point of difference between the conditions I described and those on the Rand is the matter of silver contents. The solutions at Cerro Prieto contained, as stated, about four times as much silver as gold ; Mr. Clark states that the Homestake solutions contained about half as much silver as gold. I am unable to reduce to figures the extent to which the presence of silver facilitates the rapidity of precipitation, but it appears to have a decided effect. In this connection a quotation from Mr. G. H. Clevenger's excellent paper on "The Refining of Precipitates"[*] may be of interest. In describing his experiments in passing silver solution over zinc shavings, he says :—

> "The silver was deposited in a hard, thin, adherent coating, which could not be removed for separate analysis . . . . But I have found an alloy containing from 40% to 50% of zinc and the balance of silver to show much the same characteristics. The rough surface of this alloy has the same copper-red colour, and even the polished surface has a very distinct reddish tinge. With acetic acid little or no action takes place in either case. Other reagents, such as chromic acid, hydrogen peroxide, etc., give the same reaction in both cases.
>
> That this coating is more energetic than zinc, as a precipitant of gold, copper and silver, may be seen by watching the precipitating bottles. At first the precipitation is very slow, but after the shavings have assumed the copper-red hue, it goes on much more rapidly."

The higher cost of smelting zinc dust precipitates is due to the character and greater bulk as compared with zinc shaving precipitates. It requires more flux, fuel and labour for their reduction. To form an idea of this difference in bulk, compare the zinc consumption at Cerro

---

[*] Trans. Am. I. M. E., October, 1903.

Prieto, 0·4 lb. per oz. of metals precipitated, with the published results at the Liberty Bell in Colorado, where zinc shavings are used and the consumption of zinc is given at 0·29 lb. per oz. of metals precipitated. Incidentally, I might add that the zinc consumption at Cerro Prieto is lower than at any other plant I know of that is using zinc dust. Acid treatment has, I believe, been found difficult of application, being more costly by reason of the greater bulk of the precipitates, and because for some reason the precipitates made with zinc dust do not yield as readily to acid treatment as those made with zinc shavings.

Gravity feed is the easiest way to obtain a granular product for filter pressing. It is not the only way, pumps being used with entire success, notably at the Homestake, and at the new 120-stamp mill of the Goldfield Consolidated in Nevada.

NOTES ON PRECIPITATION.

(*Read at March Meeting, 1909.*)

By Mather Smith (Member).

Reply to Discussion.

Mr. Mather Smith (*Member*): While thanking Mr. J. Hayward Johnson for his congratulations, I must object to his assertion that my paper " resolves itself more into a matter of time the solution is in contact with the zinc than the depth or amount of zinc required to a ton of solution." He could easily have calculated the time in which my solution was in contact with the zinc from the figures given in my paper.

Without allowing for the displacement of solution by the zinc the solutions are in contact with the zinc in each compartment for about 3·8 minutes and when he says that "even with boxes having compartments 5 ft. × 2 ft. × 2 ft. 6 in., the solution is only in contact with the zinc for a period of 6 to 8 minutes," he must surely mean in each compartment, otherwise assuming the box to be one of seven compartments the solution would be flowing through it at over 1,000 tons per hour.

When Mr. Hayward Johnson says, "at the same time the author's suggestion that all boxes should be built in the same proportion as those in use by him, would necessitate a box receiving 200 to 250 tons solution per 24 hours to about 12 ft. square," I do not understand what he means. I suggested boxes of five compartments each, with a zinc area of 1 sq. ft. per ton of solution per 24 hours, and admit the area suggested to be much larger than necessary, but there was certainly nothing in my paper to suggest boxes 12 ft. square.

A box to precipitate 200 to 250 tons of solution per 24 hours, constructed and dressed with zinc as these are on this plant, could be one of five compartments, each compartment having an area of 5 ft. × 2 ft., and with less than 2 ft. of zinc in each compartment.

Since writing my paper I had the weak solution box divided into two boxes of four compartments each, and am now able to give the sands a larger percentage of solution. The solutions here enter the strong box at about ·08%, the medium box at about ·05%, and the weak about ·03% cyanide.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(*Read at May Meeting, 1909.*)

By Jas. Gray, F.I.C. (Member).

DISCUSSION.

The President: Before criticising the paper submitted for our consideration by Mr. James Gray, I feel that I cannot pass without comment the slur cast upon many members of this Society by our Vice-President, Dr. Moir. In expressing his suprise that discussion on this paper had not eventuated, he says: " Shall I be accused of prejudice if I suggest that its fate might have been different had it tended towards reduction of working costs, instead of helping, as it does, those on the other side of a lime contract."

With reference to his surprise, it will be remembered that no discussion on this paper in open meeting has so far been possible. Had there been an occasion for doing so, I was prepared with notes to supplement the author's remarks. His insinuation that members willingly withhold fair criticism is certainly very unfair, whilst his concluding remarks show that he has failed to entirely grasp the present position between buyer and seller in these contracts. It is now some eight or nine months since, on behalf of the Consulting Metallurgist of the Consolidated Goldfields, we carried out some tests on the deterioration of lime due to atmospheric influence and these bore out what was already known to us and which had been taken into consideration in drawing up the lime contracts for our mines. The method employed for these contracts is similar to that mentioned by Mr. Gray in method 3, viz., the truck of lime is weighed on receipt at the mine, a sample from several bags immediately taken, ground into finer lumps, re-sampled and bottled. The payment is made on this weight and on the estimated percentage of available caustic lime. Notwithstanding Dr. Moir's innuendoes, I maintain that, in this way, the seller gets paid exactly for what he supplies and the mine pays absolutely only for what it gets.

The author's paper is excellent, but one would wish that he had carried his tests further. His second conclusion is, to my mind, based on insufficient data. Therein he says that the absorption of carbon dioxide is extremely slow when compared with the absorption of water. The following results would tend to disprove this:—

Commercial lime, after being pulverised through a ball-mill, was heated at a low red heat for some time in a muffle and after cooling in dessicators, lots, each of 80 gm., were placed in round shallow watch glasses, 3¼ in. broad by 1 in. in depth at the centre. These were placed side by side on a bench in a partially enclosed part of the verandah, so that no direct sunlight nor rain reached them. Showers were of almost daily occurrence.

Duration of Exposure. Days.	CO_2 Absorbed. Per cent.	Moisture Absorbed. Per cent.
1	1·4	4·2
2	1·6	11·2
3	2·6	14·3
4	3·3	17·0
5	4 0	18·2
6	5·2	18·4
7	5·6	18·8
14	9·2	19·0

Here it will be noted that at the end of two days seven times more water has been absorbed than carbon dioxide, whilst at the end of 14 days the ratio is 2 to 1.

I would also cavil at the author's experiments on lumps of limestone. Here also his premisses are at fault, since he has neglected the fact that, as received on the mines, the lime is packed in bags which protect to a certain extent the lumps whilst the interior pieces are certainly not so liable to be deteriorated as the external ones.

A small bag was made from ordinary sacking, as used in lime transport, filled with 6¼ lbs. of ground re-burnt lime and securely tied. This parcel measured 9½ in. × 7 in. × 6 in. After 14 days of exposure in a covered verandah to atmospheric changes—showers and heavy rain being frequent—the bag was opened and the lime tested.

	Per cent.
Available caustic lime in re-burnt sample	67 58
Increase in total weight after 14 days, 15 oz.	15·0

Estimated on the basis of the original weights the

Available caustic lime amounted to ...	65 80
Or on the actual weight of sample ...	57·21

A part of this sample lying against the sack was taken to a depth of about ¼ in. This gave an available caustic lime, equivalent on actual weight to 39·13%, whilst a portion from the centre gave 63·3%. The former contained 14·2% CO_2 and the latter but 3·76% CO_2.

These are but a few of the results obtained by us, and they tend to show that if "the lime burners are cheating themselves," the consumers are not cheating them. The only way in which the sellers lose is in the absorption of carbon dioxide during transport and we can only recommend, what I doubt not they have all recognised long ago, thorough burning of the lime in the first instance and speedy delivery.

These results I do not put forward to in any way lower the value of the author's figures. Commercially, they are extremely useful and interesting, but variations of atmospheric conditions, in the quantity of lime xperiemented with and in the state of comminution, render corroborative results, except in a general way, extremely difficult to obtain.

Mr. H. A. White (*Member of Council*): I discussed this matter the other day with a gentleman who volunteers to supply our mines with lime, even under these painful conditions. He said he did not find in his experience very much difference in the lime assays in the different seasons of the year. Lime is delivered in closed trucks containing quite a number of bags of lime, and only the outer bags are much exposed, so that when we come to sample the lime the changes Mr. Gray alludes to, practically do not matter unless very considerable delay takes place. It is a commercial matter. If the merchants who supply the lime send one or two pounds under weight absorption supplies the deficit when it comes to the mine, but Mr. Crosse's* suggestion to slack the lime would be a more straightforward way out of the difficulty.

Mr. W. R. Dowling (*Vice-President*): In my experience the burner does protect himself. I checked a number of bags, as suggested by Mr. Gray, and I found that even after they had been some nine days en route, the weight was still light.

STOPE MEASUREMENTS.

(*Read at May Meeting, 1909.*)

By Otto S. Tonnesen (Member).

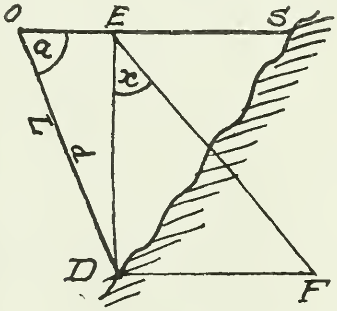
DISCUSSION.

Mr. A. A. Coaton (*Member*): I wish to thank Mr. Tonnesen for his very interesting and instructive paper. I agree with the author's remarks about the hanging compass and clinometer method. My chief objection to the use of the transit for stope measuring is at that stage of the operation where the "horizontal fathomage" is converted to "inclined fathomage." The question of setting up the transit rapidly and safely on steep inclines (not exceeding 60°, as this is about the steepest angle to which the telescope of an ordinary transit can be dipped) can be overcome by adopting some such method as is used on the Wit Deep or the Simmer Deep. The method, briefly described, is to have a hole drilled in the footwall into which a threaded bolt is tightly wedged. On to this bolt a special holder is screwed which supports the transit firmly and allows of its being rapidly levelled up. I would suggest that the dip be determined from several of the longer sights taken not very far from the direction of dip —— the only special sight required being the bearing when a horizontal line of sight strikes the face.

---

* See this *Journal*, vol. viii., p. 85.

Referring to the figure—a horizontal projection



Let O be the set-up, and O S the direction of the line of strike. Let D be a point sighted on the face. Let the slope distance of O D = L, and the dip of O D = *d*, construct the right angled triangles O D E and D E F making D F = L *sin d*. Then the angle D E F = *x* = angle of dip of the plane containing the two lines O S and O D. The angle S O D can be determined by substraction of the field readings—call this *a*. Then *tan x* = D F/ D E. D F = L *sin d*. DE = D O *sin a* = L *cos d sin a*.

$$tan\ x = \frac{L\ sin\ d}{L\ cos\ d\ sin\ a} = \frac{tan\ d}{sin\ a}$$

The transit method is necessarily slow on account of having to book two sets of angles for each sight. Yet with practice in steep stopes, shots can be taken and booked by one observer about as fast as the assistant can climb round the face.

Dr. Moir objects to the use of the parabola in place of the catenary. Take the equation of the catenary $y = \frac{a}{2}\left(e^{\frac{x}{a}} + e^{-\frac{x}{a}}\right)$, for *a* substitute the value 300, for *x* substitute the value 50 ft., *i.e.*, ½ of 100 ft. span. Then on solving for *y*, we get $y = 304{\cdot}1764$ ft. The sag in 100 ft. span being 4·17642 ft. The area of the catenary works out at 278·586 sq ft. This is determined from the formula $\text{Area} = \frac{a^2}{2}\left(e^{\frac{x}{a}} - e^{-\frac{x}{a}}\right)$ Finding the area of the parabola by taking the span at 100 ft., and the sag at 4·17642, we get the area equal to 278·428 sq. ft. The difference, ·158 sq. ft., between these two, for the purpose under consideration, is negligable.

On p. 378 of the May *Journal* Mr. Tonnesen says, "The true error is, however, greater than shown in the above table, owing to the difference in length between the face line, and the straight line A B." I fail to see this, and would like the author to prove it.

From the investigation of the properties of the parabola, it will be seen that the maximum vertical sag takes place in the vertical line drawn through the point midway between the points of support, and can therefore readily be measured underground. The length of the line, the dip of the stope and deviation (measured on the reef plane) of the tape line from the true line of dip being known—the error introduced by all these factors can readily be determined by a mathematical formula. I shall not weary the members by a proof of this as it is rather long, but the result is

$$E = \frac{1}{36} \times \frac{2}{3} \times l \times s \times sin\ a \times sin\ b.$$

Where E = the area in square fathoms of the parabolic segment projected on to the reef plane.

l = the length in feet between the points of support.

s = maximum vertical sag in feet.

a = the angle of dip of the reef plane.

b = the angle between the line of the tape and the direction of dip of the stope—this angle being measured on the reef plane.

Substituting l = 100 ft., and *s* = 1 ft.

The formula reduces to E = 1·852 × *sin a* × *sin b*

From this formula the following table has been calculated for different values of *a* and *b* :—

ANGLE OF DEVIATION.

Angle of Dip	10°	20°	30°	40°	50°	60°	70°	80°	90°
10°	·06	·11	·16	·21	·25	·28	·31	·32	·32
20°	·11	·22	·32	·41	·49	·55	·61	·62	·63
30°	·16	·32	·46	·60	·71	·80	·89	·91	·93
40°	·21	·41	·60	·77	·91	1·03	1·14	1·17	1·19
50°	·25	·49	·71	·91	1·09	1·23	1·36	1·40	1·42
60°	·28	·55	·80	1·03	1·23	1·39	1·54	1·58	1·60
70°	·31	·61	·89	1·14	1·36	1·54	1·71	1·75	1·78
80°	·32	·62	·91	1·17	1·40	1·58	1·75	1·80	1·82
90°	·32	·63	·93	1·19	1·42	1·60	1·78	1·82	1·85

To illustrate the use of the table we will take the following example :—

A stope face is measured from a line 200 ft. long with a maximum vertical sag of 5 ft. The dip of the stope is 30°, and the tape deviates 20° from the line of dip. What is the error in measurement ?

From the table at the intersection of 30° and 20° we find ·32, which is the error caused by a tape 100 ft. long with 1 ft. sag. Multiply this by 2 and 5 and you get 3·2 square fathoms—the required result. The author's example is rather exaggerated, as no one would use a tape 316 ft. long, and run it down the stope at an angle of 41° away from the line of dip, as illustrated in his example. He likewise seems to be rather unfortunate in his choice of tape triangulation used on the Rand? A large number of mines use the tape triangulation method and have no projection error, because the new pegs are triangulated from the old pegs, and not from survey pegs in the drives, or rather from sagging lines joining these pegs as shown in his Figs. 4, 5a and 5b. The tape triangulation system should be carried on independently of the drive pegs, and should be fixed from the triangulation pegs themselves only. Personally, I have never found any difficulty in judging the right angle where the off-sets do not exceed 30 ft.

I disagree with the author when he makes the general statement on p. 379, that the present methods are "expensive, slow and complicated." I guess he will have to go a long way to find a quicker method than tape triangulation for stopes not exceeding 30° in dip. Neither is this method expensive or complicated. When the dip exceeds 30° I agree with him that the surveyor has a tough time of it, and grave errors are likely to be introduced.

The author has certainly shown a lot of ingenuity in devising his Stereometer, and in applying it. Having had no experience with the instrument, I am not in a position to criticise it, but I am of the opinion that it would take a deal of practice before one could use it with facility. The idea of using a triangle net should meet with hearty approval.

With regard to plotting the readings, I would suggest a large paper protractor with a central circular space, 12 in. diameter—the graduations being at the inner edge. If then a circle 12 in. in diameter be drawn round the set-up station—the protractor could be quickly centred, and the readings directly plotted with the aid of a scale. I have mathematically investigated the effect of holding the instrument at an angle to the reef plane, and found that when the dihedral angle between the reef plane and the plane of the instrument is 5° the greatest error in measuring an angle on that plane is 6 mins. 34 secs., or ·19 ft. in a shot of 100 ft. length. When the dihedral angle is 10° the greatest angular error is 26 mins. 19 secs., or ·77 ft. in 100 ft. So it is seen that the effect of this error is very small.

Before I heard of the author's invention, I was trying to effect an improvement in stope measurements, and was in favour of using a miner's dial with split sights, and measuring the angles on the plane of the reef. In steep stopes I proposed to use the method of support previously described. In flat stopes I would use a small tripod and centre under a plumb-bob of fixed length, or attach the instrument to the hanging wall by means of a special holder. The author has, however, solved the problem in a far neater way. At the May meeting he asked for a cheap distinctive light. I would suggest acetylene. An acetylene lamp, however, is an awkward thing to carry when climbing about. This could be overcome in the following way:—

Remove the reflector and burner, fasten the generator to a belt round the waist, attach thick india-rubber tubing to the gas outlet and carry it down the coat sleeve, insert the burner in the free end of the tube and clip it on to a wrist-band. This method would also prove very useful for any kind of surveying work underground as the hands would be entirely free, and the light would be very convenient for booking, reading verniers, climbing chains and ladders, etc.

In conclusion, I again wish to thank the author for his paper, and wish him every success with his Stereometer.

The meeting then closed.

Contributions and Correspondence.

CYANIDATION OF MANGANESE SILVER ORES OF MEXICO.

In the March, 1909, issue of the *Journal* there is a communication from Mr. Robert Linton on the Cyanidation of the Manganese Silver Ores of Mexico. As I did considerable work on these ores about two years ago, and as my conclusions are somewhat different to those of Mr. Linton, I thought it might be interesting to our readers to compare notes on this subject.

When I first encountered this class of ore I did not suspect an intimate association of the silver with the manganese, though I knew both were present. I tried cyanidation with all strengths of cyanide, even up to 5%, on samples of the ore ground to pass 200 screen, and could only extract from 5% to 15% of the silver. I tried the addition of all kinds of reagents, such as lead salts, oxidisers, chlorides, oxygenation with air, hot solutions, etc., without effect. I then started a course of reasoning somewhat as follows:—I ascertained that this insoluble silver was contained principally in the surface ore between the croppings and the 200 level, while ore from the bottom of the mine would yield 90% to 95% of the silver without any trouble; the ore in each case appeared to be very similar, except that the surface rock which gave the trouble was highly oxidised: then it occurred to me that if I could reverse this condition and treat the oxidised ore with a reducing agent I might get a result similar to that obtained on the lower level ore from the sulphide zone. I accordingly subjected the finely ground ore to treatment with solutions of various reducing agents, among which were hydrogen sulphide, sodium sulphide, sodium hydro sulphide, ammonium hydro-sulphide and roasting with sulphur in a closed vessel. The charges were agitated for about twenty-four hours and then well washed with water and cyanided by agitation in bottles. The sodium sulphide had no effect whatever, but both the ammonium and sodium hydro-sulphides and the hydrogen sulphide increased the subsequent silver extraction to about 73%, against about 5% by straight cyaniding. The cyanide consumption was also increased from about ½ lb. per ton in straight cyaniding to 12 lb. per ton for the samples treated by hydrosulphide and to 27 lb. for that treated by hydrogen sulphide. The resultant cyanide solutions carried large quantities of thiocyanate. The sample roasted with sulphur gave about the same yield as those treated with hydro-sulphide and about the same cyanide consumption. To try to reduce this heavy loss of cyanide the ore after reduction treatment was given various treatments before cyanidation, among which were the application of chlorine water and various chlorides and of oxidisers, such as potassium permanganate and hydrogen peroxide, and also the blowing of air through the pulp; they were then well washed before cyanidation. In the case of the intermediate oxidation tests some cyanide was saved in the subsequent treatment, but the extraction of silver was considerably less than when the ore was cyanided immediately after the reduction treatment. The intermediate chlorine treatment made no difference in the cyanide consumption but increased the silver extraction by 3% or 4%.

Owing to the excessive cyanide consumption and the low value of the ore ($8.00 to $10.00 per ton) this method of treatment by reducing agents could not be entertained although, if the cyanide loss had been normal, the method would have been well worth considering. The reason why this treatment increased the silver extraction was rather mysterious, and attempts were then made to ascertain the mode of occurrence of the silver. By grinding the ore to pass 200 screen, and boiling for an hour in strong nitric acid, then washing well with distilled water and assaying the residue, the latter was still found to contain 75% of the original silver content. This residual silver insoluble in nitric acid could not have been in the metallic or sulphide forms, nor was it mechanically inaccessible in the ordinary sense, because a sample of the pure clay slime from the ore was even more refractory to nitric acid than the general sample. It might have been in haloid form, but in that event it should have been soluble in cyanide or sodium thio-sulphate, which reagents had practically no effect on it.

About this stage of the enquiry I obtained the assistance of Mr. G. H. Clevenger, who made many experiments, of which the following is a short summary: (1) Bromocyanide, results rather inferior to those by straight treatment. (2) Addition of zinc dust as a reducing agent, no result. (3) Addition of metallic sulphides with a view to breaking up any hypothetical haloid compounds, no result. (4) Addition of strong alkali, with and preliminary to, cyanide treatment, no result. (5) Preliminary treatment by pan amalgamation, no result. (6) Preliminary treatment with chlorine and bromine prior to cyaniding, no result. (7) Preliminary roast before cyaniding, result—not a trace of silver soluble in cyanide. (8) Preliminary chloridising roast, result—fair silver extraction (75%). (9) Concentration of the ore ground to 200 mesh, result—tailing assayed rather higher than concentrate. (10) Oil concentration, no result. (11) Acid flotation, no result. (12) Preliminary treatment with hydro-sulphides, results confirmed the tests made by myself, as described above. (13) Preliminary treatment with hydrochloric acid, result—94% extraction of the silver.

From the foregoing it appeared that the silver is in very intimate association with some other mineral, probably with the manganese, possibly occurring as a definite manganese-silver compound.

As to how the silver may be made amenable to cyanide solution, however, I have concluded that chloridising is not the only, and probably not even the best, method. I am led to think that instead of trying to attack the silver, which is comparatively small in quantity, the aim should be to attack the predominant partner in the combination, that is, the manganese. In the above detailed tests, a chloridising roast yielded only 75% of the silver on subsequent cyanidation, while the preliminary treatment with hydrochloric acid allowed of an extraction of 94%. Of course, in this latter instance, Mr. Linton may reply that the result was due to nascent chlorine formed by the manganese dioxide; it might, however be equally well due merely to the dissolving action on the manganese, exposing the contained silver to the action of the subsequent solvent, and this view would seem to be strengthened by the fact of the chloridising roast giving results so much inferior to those of hydrochloric acid. But if this latter view be correct, how are we to account for the results due to a preliminary reduction with hydro-sulphide? The only suggestion I have to offer is that the manganese being converted into sulphide became sufficiently soluble in the cyanide solution to expose most of the silver to the action of the solvent, forming thiocyanate and mangano cyanide of potassium.

Working on the hypothesis of the necessity of finding some solvent that would break up the combination and free the silver, I tried a preliminary treatment with many reagents. (1) Nitric acid had been found useless in the early stages of the investigation, probably because it did not attack tne manganese dioxide. (2) Hydrochloric acid while very effectual appeared to be prohibitive in cost. (3) Sulphuric acid was without effect. (4) The hydro-sulphide treatment, while being effectual, was prohibitive on account of the loss of cyanide, though I found it possible to regenerate KCN from KCNS by electrolysis with caustic soda, and it might have been possible to recover some of the cyanide in this way. (5) Finally I tried 5% solution of sulphurous acid. After washing and cyaniding the residue I obtained an extraction of 84% of the silver with a cyanide consumption of only 4 lb. per ton of ore. This method seemed by far the most feasable of any, and if the ore had assayed a few dollars higher than it did the method would certainly have been tested on a working scale. The amount of sulphur required would in this instance have been about 100 lb. per ton of ore, which was more expense than the grade of rock would have warranted.

Thus we see from the foregoing results that there are several methods of treatment besides chloridising which were found to give good cyaniding extractions on the silver in this manganiferous ore; (*a*) Preliminary roasting with sulphur in absence of air. (*b*) Preliminary treatment with alkaline sulphides. (*c*) Preliminary treatment with hydroch'oric acid. (It may be open to those who think differently to put this method in the chloridising class.) (*d*) Preliminary treatment with sulphurous acid. Of all these, the last seems the most practicable and least expensive, and with an ore assaying a little higher than that which was experimented on, would seem to offer a workable solution of the difficulty in the cyanidation of these manganese-silver ores. It may be remarked that the gold in such ores usually yields a uniformly high extraction of about 90% under almost any circumstances regardless of the behaviour of the silver.

E. M. Hamilton.

GRADING ASSAYS AND GRINDING EFFICIENCIES.

In discussing a contribution of mine to the above-mentioned paper, Mr. H. A. White, in the May *Journal*, 1909 (p. 395), breaks a lance in favour of Rittinger's law, which law he claims to be the true theoretical criterion of work done.

From a metallurgical point of view, the exposure of surfaces can certainly be considered as work done—we may call it metallurgical work done—and, giving to the vague expression "work done" this signification, Rittinger's law, by which the "work done" here also the *metallurgical* work done, *i.e.*, the area of surfaces exposed in crushing, is claimed to be proportional to the surfaces exposed, is admittedly correct, but it is no longer true if the *mechanical* work done is to be understood.

The total area of the fracture planes multiplied by a factor determining the resistance which the molecules oppose to their separation by the exercise of any stress (crushing, tensile, shearing, etc.) represents the force required to cause the fracture, and for performing mechanical work this force has to run through a distance. This distance-factor, cheerfully ignored in Rittinger's theory, has been shown to be represented by the deformation which the body can stand before reaching the breaking point. It is immaterial that this distance by which the body can be deformed within the limits of elasticity or plasticity by the application of any external force, is not exactly defined but subject to slight variations, which for highly elastic but unplastic bodies as glass, quartz, etc., are too insignificant to be considered, and by the nature of our grinding appliances they are in addition averaged to an extent that these averages are as liable as correct figures. Dealing with relative values only, we have not even to care about the exact value of this factor, and all that we need is to be satisfied that this factor is a constant function of the diameter of the particle to be crushed.

Mr. H. A. White tells us that in Rittinger's method the "distance" factor has not been ignored, but it "merely assumes this factor to be the same in all cases." Failing to produce a more positive and intelligible explanation for the extraordinary and hardly acceptable assumption that the distance the force has to run through be the same for all sizes of particles, he resorts to hypothetical suppositions of a boldness that even Jules Verne would not have ventured to base his fantastic tales upon. With such "ultimate theoretical cases" all and everything can be proved; the perpetual motion becomes theoretically possible, we shall have heaven on earth as the ultimate theoretical case of human tendency to perfection, and by the assumption of an external force applied at the fracture plane itself, even Rittinger's erroneous contention would become a true and genuine law—as the end—value of a scientific blunder.

In face of all the wonderful discoveries of recent times, no limits are drawn to our fantasy of how mechanical forces might one day be superseded by other known or unknown forces. But in the meantime we have to live with our feet on earth, remembering that all these discoveries are *additions* only to our knowledge, and that by none of them has the truth of our fundamental laws of nature ever been shaken. The mechanical work eternally remains determined by the product of stress change into change of configuration, and however small this deformation might be, so far as mechanical crushing is concerned the external force can never act in the fracture planes itself.

Exactly as we are independent of how the work has been done, and "as our calculations must be based on the product and not on its history, further than a statement of the size of particles commenced with" we have in computing relative mechanical crushing efficiency not to care either about the kind of force applied, or the strength of the material, or the amount of waste inherent to the physical properties of the material or the limitations of our crushing appliances. The computation of mechanical crushing efficiency consists simply in establishing the relation of the effective useful work done in crushing, to the total amount of energy spent for driving the machine, and as this latter, if not known, can easily be measured or calculated, the question is therefore reduced to establishing a standard for the measurement of this effective useful work done from the tonnage and the gradings of the materials taken at the intake and the discharge of the machine. That the method of calculation by surfaces or fractures exposed cannot be claimed "true theoretical criterion" for this effective work done is convincingly proved by the obviously absurd conclusions to which the application of Rittinger's theory would lead in numerous instances.

Many years ago exhaustive series of tests were made by Van Reytt to determine how nearly Rittinger's theory holds in practice, and the conclusions arrived at were*: "That the ratio of work done to increase the surface is fairly constant with coarser sizes, but with finer sizes *the increase of surface is much more rapid than the work required to produce it*." To appreciate this conclusion to its full extent, it must be noted that Van Reytt's tests referred rather to coarse-grinding machines only, as rock-breakers, rolls, stamps, etc., and that the conclusion is not even quite exhaustive, as the discrepancy comes not so much in with the fineness as with the number of the grades by which the bodies are to be reduced. In comparing, for instance, the efficiency of tube mills, where the reduction in size extends over a range of a few grades only, the calculation by fractures exposed gives fairly correct and acceptable figures as far as the reduction of the sands is concerned, but the error becomes at once apparent if also the reduction of the banket ore-feed is considered, or in general by opposing coarse-crushing to fine-grinding. If Rittinger's theory were correct, practically all the work done in crushing would be represented in the finest grades only, and no credit is given for the hard work done in exposing the comparatively small percentage of fractures whilst reducing the larger particles. Even a large amount of coarse sands directed to the tube mills for regrinding would represent only a comparatively small percentage of the mechanical value represented in the battery pulp and contrary to practical experience the efficiency of stamps would increase with the fineness of the battery screens in consequence of the greater output in the finest grades.

In conclusion, I must say that we are rather late in discussing Rittinger's so-called law, which —never recognised by any authority—is taking its last breath in some mining books. More than 20 years ago Prof. Fr. Kick's works and experiments closed definitely the discussion on this matter, but since the withdrawal of a slander does not go through the same channels as the original slander, it will take some time before this recognised error is stamped out in all side-issues.

H. Stadler.

Johannesburg,
26th August, 1909.

Erratum.

PRECIPITATION FROM CYANIDE SOLUTIONS BY ZINC SHAVINGS AND DUST: A COMPARISON OF RESULTS AND COSTS.

By Allen J. Clark (Associate)

The following correction to the above paper (*Journal* for January, 1909, p. 223) has been received from the author:—

In the items of "Dollars recovered, total," the figures as published are:—

As published ...	$1,513	$1,513
Should read ...	1,513	1,712

---

* Richards' Ore Dressing.

Obituary.

The death of Mr. John Davies, a member of the Society, is recorded with regret.

Mr. Davies, who joined the Society in February, 1899, was for many years connected with the Geldenhuis Deep, Limited, both as mine captain and subsequently as mine manager. About a year ago ill-health compelled him to resign his position He always took a keen interest in the Society's work.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Precipitation of Arsenic by Hydrogen Sulphide.—"Arsenic acid is reduced quickly, and the precipitation of arsenious sulphide accelerated, if, to the strongly acid (hydrochloric) solution of the arsenic acid, a small quantity of hydriodic acid be added, before the hydrogen sulphide is introduced. The reduction and precipitation is still further hastened when the solution is heated. The arsenious sulphide is precipitated almost immediately, and the supernatant liquid remains clear; only after a time is sulphur precipitated. Potassium or ammonium iodide may be used in place of hydriodic acid." L. L. de Koninck, Bull. Soc. Chim. Belg., 1909, *23*, 88-93.—*Journal of the Society of Chemical Industry*, April 31, 1909, p. 330. (A. W.)

Sulphur in Coal and Coke.—"The author has made a comparative study of the methods in common use for the determination of sulphur in coal and coke. The conclusions arrived at are: (1) Eschka's method for total sulphur is liable to be inaccurate with coals containing more than 2%, and the process of Hundeshagen (this *J.*, 1893, 465, 1062) gives results which are no better. (2) Sauer's method of determining the so-called volatile sulphur should, especially with coals yielding much gas, be modified as follows:—The combustion tube is constricted to 5 7 mm. for 3-5 cm. of its length near the centre, the space being filled with pieces of platinum wire, and this contact substance is strongly heated before the boat is pushed into the tube; the gaseous products are passed through bromine water or 1% hydrogen peroxide solution, the stream of oxygen being very carefully regulated. This is regarded as the most reliable method of determining volatile sulphur; that of Pfeiffer (this *J.*, 1905, 943) is only suitable for obtaining rapid approximations. (3) The sodium peroxide method of determining total sulphur (this *J.*, 1904, 131) may be used where speed rather than absolute accuracy is essential; a stout platinum wire should however be used for the ignition instead of an iron wire; a more dilute chromate solution should be employed; and the liquid containing the chromate, after neutralisation by ammonia, should be transferred to a graduated cylinder and an aliquot part of the clear liquid titrated. (4) Accurate values are rapidly obtained with the calorimetric bomb, but sulphur may remain in the ash, if the proportion of this be high, and it is sometimes necessary to add a substance capable of raising the temperature of combustion, in order to ensure complete oxidation of the sulphur. (5) Brunck's method (this *J.*, 1905, 1086) in all cases gives accurate results, and the author considers this the most suitable method for determining total sulphur, but it should be modified as follows:—Platinum is used as a contact substance in the combustion tube and hydrogen peroxide solution as an oxidising absorbent; after combustion, the contents of the platinum boat are treated at once with hydrochloric acid and the mixture evaporated to dryness on the water-bath, the hydrogen peroxide solution being made alkaline, boiled to decompose excess of the reagent, and added to the residue; the whole is then acidified with hydrochloric acid and filtered from the silica, and the sulphuric acid determined gravimetrically, titration being inadmissible. The determination of volatile sulphur is looked upon as of little practicable value, since its amount depends on the conditions of burning, and results obtained by combustion in oxygen cannot, therefore indicate what would obtain in an ordinary furnace. The sulphuric acid produced in the above processes of determination may in most cases be titrated, either by the benzidine method, as modified by Friedheim (this *J.*, 1907, 116) or by Bruhn's modification of the chromate method, which should be carried out in the following manner:—An aqueous emulsion of barium chromate, precipitated from boiling solution, is prepared so as to contain about 0·05 gm. of barium chromate per c.c. and 5 c.c. of this well-shaken mixture diluted with 50-100 c.c. of water, are acidified with 1 c.c. of strong hydrochloric acid, and added to the boiling solution to be tested; boiling is continued for some time, and then the excess of barium chromate is precipitated with ammonia, the excess of ammonia boiled off, and the filtered liquid cooled in a closed flask; 20 c.c. of strong hydrochloric acid and 20 c.c. of 10% potassium iodide solution are then added, the solution made up to 500 c.c., and, after half an hour, titrated with thiosulphate."—M. Holliger, Z. angew. Chem., 1909, *22*, 436-449, 493-497.—*Journal of the Society of Chemical Industry*, April 15, 1909, p. 357. (A. W.)

Ferrous Iron in Magnetite.—"Half a gramme of ore is weighed into a 50 c.c. platinum crucible which is fitted, as closely as possible, into an opening in an asbestos board. The ore is thoroughly moistened with water and a couple of coils of platinum wire added to prevent bumping. A cold mixture of 10 c.c. of hydrofluoric acid and 15 c.c. of dilute sulphuric acid (1 : 3) is added and the crucible quickly covered with a tightly fitting lid. A 6 in. tunnel, coated inside with paraffin wax and connected to a carbon dioxide generator is placed over the crucible and its junction with the asbestos board made air-tight by pouring a little water round it. In a few minutes, the air is completely expelled by the carbon dioxide and decomposition started by heating the contents to boiling and maintaining at gentle ebullition. When decomposed, *i.e.*, after a little more than 5 minutes, the crucible is allowed to cool and when the steam inside has condensed, the crucible, with the lid held tightly on, is placed in a beaker containing 400-500 c.c. of cool boiled distilled water. The contents of the crucible are quickly mixed with the water, a solution containing sufficient calcium phosphate to precipitate the excess of hydrofluoric acid added and the solution titrated, as rapidly as possible, with a standard solution of potassium permanganate, the first permanent coloration being taken as the end point. Control determinations should also be made. Duplicate assays should agree within 0·1 c.c. of permanganate. The strength of the permanganate solution is such that 1 c.c. is equivalent to 0·0025 gm. of iron. The calcium phosphate solution for 0·5 gm. of ore is made by suspending 5 gm. of calcium carbonate in 50 c.c. of distilled water and slowly stirring in 10 c.c. of 85% orthophosphoric acid. The excess of sulphuric acid remaining after decomposing the ore is sufficient for titrating but should not be enough to prevent the precipitation of the calcium fluoride."—R. B. GAGE, *Journal of the American Chemical Society*, 1909, *31*, 381-385. —*Journal of the Society of Chemical Industry*, April 15, 1909, p. 367. (A. W.)

GOLD IN COPPER BULLION.—"When dissolving copper bullion in concentrated sulphuric acid, enough cuprous sulphide is formed to interfere with cupellation, unless preceded by more than one scorification. Most of the sulphide is formed at the beginning of the action, but if acid containing copper sulphate is used to dissolve the bullion, very little sulphide is formed. *Assay:*—Heat a mixture of 80 c.c. of concentrated sulphuric acid and 25 c.c of a 16% solution of copper sulphate, and when hot enough to commence immediate action, add 1 assay ton of the borings. When action ceases, cool and add while stirring, 400 c.c. of water, just boil and filter. Dry the residue, ignite the paper and scorify together with 35 gm. of lead and 1 gm. of silica, then cupel and part. If silver is also to be determined, the filtrate is oxidised with potassium permanganate and 1 c.c. of a saturated solution of sodium chloride and 10 c.c. of a 10% solution of lead acetate added and well stirred. After standing over night, the precipitate is collected, and added to the gold residue, the two being scorified together."—F. F. HUNT, *Engineering and Mining Journal*, 1909, *87*, 465.—*Journal of the Society of Chemical Industry*, April 15, 1909, p. 368. (A. W.)

QUARTZ COMBUSTION TUBES.—"Tubes of clear quartz (silica) were found to be well-suited for use in organic combustions, and to be much longer-lived than glass tubes. Silica withstands the action of copper oxide, lead chromate, etc., very well unless the temperature be excessive; at about 900° C., however, copper oxide attacks and destroys quartz tubes. The tubes are also suitable for the determination of oxygen in copper by fusion in hydrogen; the authors heat the tube electrically by passing a current of suitable strength through a long strip of platinum, or iridium-platinum, which is wound round the tube and surrounded by a coating of asbestos cloth. For the determination of carbon in steel, the quartz tube was surrounded by two separate sections of stout iridium-platinum foil at a short distance from each other, each section being enclosed in two fireclay semi-cylinders which, in turn, were covered, with asbestos cloth. The sections were so wired, that the current could be passed through either section, or both. The steel to be analysed was placed in a boat surrounded by a piece of iridium-platinum foil rolled into the form of a tube in order to prevent injury to the quartz by the sparks of oxidised metal. The boat itself was protected from the action of the burning steel by placing a layer of ignited alumina in the bottom. Since steel, when burning in oxygen, suddenly absorbs a large quantity of oxygen, means have to be provided for supplying a sufficient quantity of the gas at the moment that it is required. For this purpose, the supply of oxygen to the combustion tube is passed through a Y-tube, one arm of which communicates through a trap with the open air. The oxygen is passed into the tube at such a rate, that, before the steel begins to burn, a portion escapes into the air, the rate of the gas passing through the potash bulbs at the other end of the combustion tube being controlled by the action of an exhaust pump. When the steel burns, the escape of oxygen into the air ceases, and air itself is drawn into the combustion tube, mixing with the oxygen and passing through the same purifying apparatus. The oxidation of the carbon monoxide evolved during the combustion, is effected by packing the exit end of the combustion tube with a mixture of platinised quartz and platinum scrap, and a supplementary supply of oxygen is admitted to this end of the tube through a side tube which ends just in front of the oxidising mass. A piece of silver foil, placed just before the boat, protects the rubber stopper from the intense radiation characteristic of quartz tubes. The results obtained agree well with each other, and differ by less than 0·02% from those given by the solution method."—A. G. LEVY, *Analyst*, 1909, *34*, 88-95. *Journal of the Society of Chemical Industry* April 15, 1909, p. 384. (A. W.)

NEW APPLICATIONS OF ELECTRO-METALLURGICAL ALLOYS.—"Among the alloys prepared by means of the electric furnace the most important are those containing iron, and particularly the ferro-silicons.

Substances prepared in this way have a percentage of silicon varying from 20 to 75%, the most usual being 20 to 25, 50 to 55, and 70 to 75%.

Ferro-silicons with percentages above 20% are insoluble in acids.

Thus, by using high percentage ferro-silicons, apparatus practically unaffected by acids are obtained.

In answer to the question whether the problem of constructing apparatus for the preservation of acids is solved by making use of metallic silicides, it may be said that (1) these substances are very fragile; (2) they contract considerably on cooling, often breaking in the mould when scarcely solid and still at a bright red heat; (3) their melting-point is low compared with the temperatures occurring in ordinary foundry practice—*e.g.*, the second melting in a coke or furnace gas-heated cupola; and (4) certain of these compounds are powerful reducing agents, so that an effect similar to that produced by aluminium may occur. In consequence, fusion in the presence of the air necessary for combustion of the coke or furnace gas is impossible without considerably lowering the percentage of silicon (especially in rich ferro-silicons—*i.e.*, about 35%) to such a point that sometimes it will become as low as 15%, and thus give alloys which are attacked by moderately strong acids.

Manufacture of Apparatus.—The first part of the process consists in obtaining melted metal suitable for pouring into the moulds.

The simplest and most used of these methods consists in pouring alloys, melted in that apparatus, directly from the electric furnace into a mould lined with some refractory material. The mixture is then made. In a second method, the alloys obtained from the electric furnace are remelted in an oil, or in certain cases in a crucible, furnace and then cast into ingots. Once the liquid metal is obtained, metals for corrective purposes are added. The number and nature of these is practically infinite for any particular case. They are added directly to the metal in the casting ladle, usually with aluminium, which raises the temperature of the bath and increases the fluidity.

When the ladle is ready it is used mechanically, by a travelling crane, and carried quickly, to prevent cooling, to the mould. The pouring is effected by ordinary means without any special precautions.

The casting is removed as soon as it solidifies, is turned out into the warm casting sand, and thus annealed at a bright red heat. It is then ready for use.

Resistance to Acids.—The resistance of the metallic alloys of silicon, which have been called 'Métillures,' to acids is quite interesting, as the following examples will show. Nitric acid, even as a vapour, such as is obtained at the exit of a bisulphate retort or when mixed with nitrous acid, does not affect them at all. A striking example of this is given by a pipe which has been submitted for nearly five years to the daily passage of 660 lb. of nitric acid vapour at temperatures varying from 150° to 200° without its loss in weight exceeding a few decigrams in a total weight of a score of kilogrammes. This loss occurred quite at the beginning of the period, and was probably due to a few impurities remaining on the inner surface of the pipe after fusion.

Sulphuric acid has still less effect, and acids of strength from 20 to 25, known as *petites eaux*, may be concentrated in vessels of this kind by direct heating and continuous evaporation.

Hydrochloric acid is analogous in its action, and use has been made of pipes of the above material for carrying and condensing its gases.

Acetic acid and the mixture produced by treating calcium acetate with sulphuric acid are without effect on 'Métillures,' even in the presence of air.

Resistance to other Reagents. — An important English firm has obtained figures relating to the resistance of 'Métillures' to the action of cyanides, sulpho-cyanides, and prussic acid. The results are very remarkable, for they show both that the unattackability increases with the percentage of silicon, while for a suitable percentage it is practically negligible.

To sum up, it may be said that the production of a material capable of resisting acids is an accomplished industrial fact, and the principal desiderata are realised. It remains, however, to remedy the unmalleability of these substances. This question is actually being studied, and a solution seems probable in the near future."—A. D. JOUVE.—*Transactions of the Faraday Society*, April, 1909, p. 155. (H. A. W.)

FORMATION OF GRAPHITE BY THE INTERACTION OF MAGNESIUM POWDER AND CARBONATES.—"Winkler, in an investigation of the action of magnesium on the carbonates of the alkaline earths, showed that the carbonates were readily reduced on heating, and amorphous carbon was produced in the reaction. The properties of this carbon were not investigated by Winkler or by Maquenne, who also studied the reaction between magnesium and the alkaline earth carbonates.

During an investigation of the action of magnesium powder on various carbonates quantities of carbon were produced, and the object of the present paper is to show that this carbon is a mixture of amorphous carbon and graphite; in some cases it consists almost entirely of graphite. The graphite is non-intumescent.

A mixture of a carbonate and magnesium powder does not require heating before action commences. If such a mixture be ignited in one portion by a match or by means of burning magnesium, a reaction similar to Goldschmidt's occurs, and proceeds throughout the whole mass more or less vigorously; in many cases CO is evolved, which causes much of the contents of the crucible to be ejected; if the metal is of comparatively low boiling-point, this distils off and burns and so renders the reaction more violent. Since this reaction was accompanied by very high temperature, and since at high temperatures graphite is the stable form of carbon, it was thought that the carbon would be present in this form. In all cases this was found to be so, although amorphous carbon was also present, the graphite being formed as such or formed from amorphous carbon at the extremely high temperature of the reaction.

The reaction between barium and strontium carbonates and magnesium powder when carried out in open crucibles always produced the carbide, nitride, and large amounts of cyanide: when the reaction took place in closed tins, the carbides only were produced. With calcium carbonate no carbide, nitride, cyanamide, or cyanide was produced when open crucibles were used; when the reaction took place at high temperatures in closed tins, calcium carbide only was produced. Since results differing from those of Winkler and Maquenne were obtained, a subsequent paper on this subject will be given.

It is well known that most carbides dissolve carbon and deposit them in the form of graphite: that the graphite was not obtained in this way is evident from the fact that the mass had not a fused appearance, but was a very soft powder."—H. RUSSELL ELLIS.—*Transactions of the Faraday Society*, April, 1909, p. 173. (H. A. W.)

NOTES ON THE IODIDE METHOD FOR COPPER ASSAYS.—"There has been considerable controversy on the question of the iodide and electrolytic estimation of copper, but a few further notes on the subject may be welcome.

Perhaps Van Agnew (Siberia) claims more for the iodide method than does any other writer on the subject. His letter to the *Engineering and Mining Journal*, 6th April, 1905, reviews the method and the effect of its operation at the works to which he is attached. Among other things, he says:—'It is mere trifling to 'monkey' with the battery assay, for in my mind all doubt regarding the accuracy of the iodide method, as compared with the battery method, has long been removed.' He goes on to say that they refine their own copper, and the iodide method of assaying is adopted even on metal running up to 99·56 % Cu. He says: 'Our results are checked in Moscow by the battery method, and we invariably agree.'

Thorn Smith, writing in reply to Van Agnew, stated that the battery assay was far ahead of the iodide method. To quote his own words: 'The iodide method cannot be compared with the battery method for accurate work. In work where strict accuracy is not required it answers perfectly. In buying or selling ore, where a few tenths per cent. are of importance, this method is entirely out of place.'

The discussion is an interesting one, and this note is written with the hope that an Australian experience may be appreciated, and perhaps will cause discussion here, and so bring out further points in favour of respective methods.

The writer has had an experience of six months' assaying with the iodide method against the electrolytic whilst employed in the Wallaroo district. The seller adopted the iodide method by force of circumstances, and the buyer adopted or was in the habit of using the electrolytic method. For six months about three samples a week were assayed by the different parties, and during the whole time not 10 per cent. had occasion to be umpired. The limit of difference was 0·3% Cu on mattes and 0·5% Cu on ores. The matte contained about 35 to 40 per cent. Cu, and the ores from 3 to 40 per cent. Cu. Of the umpired samples, 55 per cent. were won by the buyer and 45 per cent. by the seller. The umpire assay was the iodide method. On working out the total difference for the period for six months, it was found to be 0·1 % in favour of the electrolytic. As the ore and matte contained traces of Ni, As, Sb and Se, some of the difference may have been due to the precipitation of these metals on the 'plate.' It was very satisfactory, and as the umpire at times bore out the assay by the battery method, and as the whole work done showed such a small difference, it would be safe to say that at least on one class of ores the iodide for all practical purposes was as good as the battery. It was considered at Wallaroo, so I have been told, that for most ores and mattes—that is, if they were not too high, like refined copper, etc., or too low, like low grade ores and tailings, one method was as good as the other. In Broken Hill, where the selling difference is 1 per cent., these facts may seem rather far-fetched, but it is quite true in every particular.

An experiment was tried by myself to test the accuracy of the method, as follows:—A matte was assayed in the usual manner by myself, then sent to Adelaide to one of the leading firms of assayers there, and another portion sent to Wallaroo:—

Assayer for seller	32·7 %	Cu iodide
Adelaide assayer	32·7 %	,,
Wallaroo assayer	32·78 %	electrolytic

The selling assayer knew nothing about the experiment being made, as his assay was made before the experiment was contemplated.

This matte was then taken, 1 gm. weighed out, and 0·2 gm. of pure copper added, and the mixture assayed in the usual manner.

Cu in matte by above assay .. ... = 0·3270 gm.
Cu added, 0·2 grm. (999 fine) ... = 0·1998 ,,
Total Cu. in mixture ... ... = 0·5268 ,,

Cu actually found by experiment ... = 0·5266 ,,

It can easily be seen, therefore, that the error in the estimation is very small indeed.

My experience with the method showed that the amount of iron in the ore affected the precipitation of the copper by aluminium, and, in fact, the assay generally. For instance, if 1 gm. of a very ferruginous ore was taken, all the nitric acid and oxide could not be expelled by sulphuric acid, even after taking to the densest fumes ; on testing the residue for nitric acid its presence was always shown, but by using a smaller quantity the trouble was got rid of. The same thing has been noticed with zincy ores when assaying for lead. The iron would, besides retarding the precipitation of the copper, be precipitated with it, and later on interfere with the end point if present in too great a quantity.

After precipitating the Cu by Al, I invariably noticed that there was a certain amount of Cu left in solution, especially if the SiO_2 content of the ore was high. A little H_2S water added to the clear liquid after decantation from the Cu completely precipitated the last traces of the Cu in solution. The CuS thus precipitated was filtered off. The Cu from the Al precipitation dissolved in nitric acid, and the beaker placed under the funnel containing CuS, and the CuS dissolved through into the beaker by dropping 5 c.c.s. bromide water over it. The action of the Br water is to dissolve the CuS immediately. Besides being useful in this respect, the Br. has other functions ; it oxidises any arsenious compounds, the higher state of oxidation, and also is material in driving out NO_2 fumes at a later stage.

I found it best even when Br was present to concentrate the solution of copper nitrate to a syrup before neutralising. This lessens the amount of soda necessary to neutralise the solution, and removes all NO_2.

I found sodium carbonate the quickest for neutralising. One c.c. of a saturated solution was usually sufficient. If ammonia were used a further boiling was necessary, and then a wait whilst cooling.

The bulk for titrating was about 75 to 100 c.c.s., and the assay was titrated in flasks. In the hot weather the water bag was utilised for cooling purposes.

A weak solution of thiosulphate was used (N/10), and one drop was sufficient to cause the disappearance of the blue color. If the blue color returned quickly, the assay was considered worthless. Instead of throwing it away, it was evaporated with HNO_3 to a syrup, and then proceeded with as usual, and the result considered as a check on the other assays if the blue color did not again return.

For low grade ores and tailings, I have found the iodide unsuitable in many ways, and after experimenting adopted the "Permanganate" method, as described in "Low's Technical Analysis," second edition. This method has been checked at Wallaroo against the electrolytic, and their experience and mine agree to the fact that the method is a reliable and a quick one.

There is always a necessity to test reagents. Time and again I found the acetic acid to contain Cu."—R. T. D. WILLIAMS, *Australian Mining and Engineering Review*, April, 1909 (H. A. W.)

OZONE MANUFACTURE.—"The value of ozone as a disinfectant is well known, and many apparatus have been designed for the production of this gas. The most familiar method of generating ozone is by means of the silent or brush discharge, and this requires a transformer using an alternating current or an interrupted direct current. A new method of making ozone has just been devised, in which a continuous current is used, and the transformer is eliminated. It consists of a tube provided with a small fan, adapted to draw the air through it. In the centre of the tube is a Nernst rod. The rod is first heated to the proper degree by a heating coil, after which the coil is disconnected in the usual manner, and the current passes directly through the rod, which is thus rendered incandescent. At the same time the fan is automatically started. Ozone is produced by the contact of the air with the incandescent Nernst rod."—*Indian Engineering*, May 22, 1909, p. 333. (A. R.)

DETERMINATION OF NICKEL.—"For the determination of nickel in steels, such as nickel-steel and nickel-chrome-steel, the methods of Brunck (this *J.*, 1908, 284) and Grossmann (this *J.*, 1907, 718) are both to be preferred to the electrolytic method (which, though accurate, is tedious) if proper precautions be taken. Brunck's method leads to good technical results, if an hour be allowed for the reaction, but 24 hours are required for the maximum accuracy. In applying Grossmann's method to the analysis of steel, 1 gm. of the drillings is dissolved in 20 c.c. of hydrochloric acid (sp. gr. 1·12), the solution is oxidised with 6-7 c.c. of nitric acid (sp. gr. 1·20), evaporated to 15 c.c., and allowed to cool. Then 40 c.c. of a 50% solution of Rochelle salt are added, with stirring, followed by 8-12 c.c. of strong ammonia to dissolve the precipitated tartaric acid and render the solution alkaline. 10 c.c. of an 80% solution of potassium hydroxide are next added, which should cause the solution to assume a light green colour, then about 0·3 gm. of hydrazine sulphate, to prevent interference from the small quantity of manganese present, and 1-1·5 gm. of dicyandiamidine sulphate. After 24 or 48 hours, the precipitated nickel compound is collected in a Gooch crucible, washed (not more than five times) with weak ammonia, and dried for 30-45 minutes at 120-140° C.; it contains 22·5% of nickel. The volumetric method of determining nickel with potassium cyanide appears, however, to be superior to either of the above, especially as regards speed, and it may be applied directly to the analysis of steels, without preliminary separation of the nickel. The methods of Brunck and Grossmann are regarded as indispensable, therefore, only in the analysis of substances containing much cobalt."—Prettner, *Chem.-Zeit.*, 1909, *33*, 396, 411-412.—*Journal of the Society of Chemical Industry*, May 31, 1909, p. 525. (A. W.)

METALLURGY.

Laboratory Screens.—Laboratory screens have recently occupied the attention of metallurgists and others whose duties have been the mechanical reduction of ore prior to concentration, or the preparation of cements and other articles of commerce requiring fine grinding.

I have invariably noticed that sizing tests made with nests of sieves give a large amount remaining on every other sieve, and a small amount on those which come between them. For instance, a sizing test made over sieves 40, 50, 60, 70, 80, 90, 100 gives results like the following:—

On 40-mesh = 10·0%
50 ,, = 10·0%
60 ,, = 2·0%
70 ,, = 10·0%
80 ,, = 2·0%
90 ,, = 10·0%
100 ,, = 2·0%
through 100 ,, = 54·0%

100·0%

This falling away in percentage on the 60, 80 and 100 sieves might happen occasionally, but when it always happens it shows that the apertures in these sieves very closely approach those in the 50, 70 and 90 sieves. This, of course, is caused by a too rapid change in the diameter of the wire used in their manufacture. Screens of this sort are of very little use for the purpose for which they are intended, and unless the figures obtained from them are used by people who thoroughly understand the errors to which they are liable, they are apt to be misleading. The above difficulty could be overcome by making the sieves with wire equal in size to the diameter of the hole. This would also enable the user to arrive at the size of the aperture, and would certainly be a great improvement on the present system.

The Institute of Mining and Metallurgy, at a recent meeting, dealt with the matter of standard laboratory screens, and adopted a series on the above lines. In my opinion, these standard screens are still far short of what they ought to be, for the various screens do not show any simple ratio between the volume of a particle passing one and that passing another. This I consider to be of great importance, especially if the screens are to be used in testing grinding machines to ascertain their efficiency. This point was brought under the notice of the Institute by Mr. E. J. Horwood before they adopted the present standard, but they either did not see the advantage to be gained, or they wanted to please those who were standing up for the old system (mesh system) as well as those for the new, or those who advocate that sieves should be known by the size of their aperture. At any rate, I think they have made a great mistake in adopting the present standard, for reasons which I think I can explain.

The object of crushing is, naturally, to reduce the size of the particle, and the amount of the crushing which has taken place is directly proportional to the reduction in the size of the particles. In measuring the size of particles by means of sieves, we must consider the particles to be cubes. This, with some minerals, undoubtedly leads to error (especially in the coarser sieves), for they break into needle-shaped pieces; but in the majority of ores the tendency is for the crushed particles to approach towards the cube or sphere. Consequently, to arrive at a simple ratio between the screens it is the cube of the apertures we have to consider, and not the mesh or the aperture alone. Having this point in view, I prepared a list of apertures which, in my opinion, would be most suitable. I took for my largest aperture one approximately equivalent to 25-mesh, and for the finest, one equal to 120-mesh. The volume of the particles passing the coarsest sieve = 1/100,000th part of a cubic inch. This I took as unity, and went up in 1/100,000ths to 1/1,000,000ths, then to 1/4,000,000ths and to 1/8,000,000ths. This gives a sieve ratio of 1 to 10 in units and 10 to 80 in 10's, giving 17 sieves altogether. This may be considered a large number of sieves, but it is difficult to cut it down if close sizing is desired. From the required volume of the particles passing the screen, I calculated the aperture. These, in many instances, ran into a good many decimal places, and it could not be expected that the screen could be made as accurately. I therefore took the nearest figure in the fourth place, as various catalogues which I have seen give the size of the aperture to the 10,000th part of the linear inch. This adjustment in the fourth place of decimals naturally threw my sieve ratios out a little, but they were near enough for all practical purposes. To have these screens made would undoubtedly be expensive, but if they were generally adopted, so that a fair quantity of each size could be ordered, the price would be materially decreased, and if used in the same quantity as the present standard screens there is no reason why they should not be manufactured at the same price.

Proposed New Sieves.

Aperture.	No. of Particles to cub. in.	Sieve Ratio.
0·0215 in.	100,620	1
0·0171 ,,	200,000	2
0·0149 ,,	302,360	3
0·0136 ,,	397,540	4
0·0126 ,,	500,000	5
0·0119 ,,	593,480	6
0·0113 ,,	693,000	7
0·0108 ,,	793,800	8
0·0104 ,,	889,000	9
0·01 ,,	1,000,000	10
0·0079 ,,	2,028,000	20
0·0069 ,,	3,044,000	30
0·0063 ,,	3,999,000	40
0·0058 ,,	5,125,000	50
0·0055 ,,	6,010,000	60
0·0052 ,,	7,111,000	70
0·005 ,,	8,000,000	80

Screens Adopted by the Institute of Mining and Metallurgy.

Mesh.	Aperture.	No. of Particles per cub. in.
5	0·1 in.	1,000
8	0·062 ,,	4,196
10	0·05 ,,	8,000
12	0·0416 ,,	13,887
16	0·0312 ,,	32,925
20	0·025 ,,	64,000
30	0·0166 ,,	218,600
40	0·0125 ,,	512,000
50	0·01 ,,	1,000,000
60	0·0083 ,,	1.748,700
70	0·0071 ,,	2,793,900
80	0·0062 ,,	4,196,000
90	0·0055 ,,	6,010,518
100	0·005 ,,	8,000,000
120	0·0042 ,,	13,497,462
150	0·0033 ,,	27,826,000
200	0·0025 ,,	64,000,000

E. T. HENDERSON.—*Australian Mining and Engineering Review*, November, 1908. (H.A.W.)

WEST AUSTRALIAN METALLURGY.—"Although the 'Golden Mile' has proved to be the richest area of its size in the Commonwealth, it is very remarkable that so little gold has been found in the alluvium adjoining it. There must have been a great deal of denudation, and yet there is remarkably little debris to be seen. There were numerous quartz veins mixed with the schist forming the lodes, and yet very little broken quartz or schist was found near them. The cement, which in many places covered the lode, had very little ore mixed with it. In stoping the ore up to the cement, the latter was frequently left as a natural bridge with a clean, smooth soffit. The lake-bed to the south has only a few inches of mud on it, and the short alluvial leads that have been worked show very little debris in the form of gravel, and the gold found in them is sharply angular. There are no unconformable strata in the neighbourhood, nor much depth of alluvium in the valleys on either side of the ridge of hills. I would like our friends the geologists, who are so well represented at this meeting, to explain this phenomenon. I would also like them to tell us why so little telluride ore has been found north of the Brownhill.

The good ore in the oxidised zone had a considerable amount of chlorite which gave it a greenish tinge, and it also had a good deal of slimy talc on the fracture faces of the quartz; with depth the chlorite and talc gradually disappeared. Unless the schistose ore was gritty it was very seldom payable. The gold obtained by amalgamation from this class of ore was worth £4 2s. 6d. per oz.; that obtained from the unoxidised ore in the same mines is worth about £3 15s.

Immediately under the oxidised ore the tellurides were most plentiful and varied. Besides the various tellurides of gold and silver there was telluride of mercury, but below 1,000 ft. very few specimens of telluride of mercury have been found. Below this depth sulphides of copper, antimony, cadmium and zinc are seen occasionally, and a little sulphide of arsenic has been found in one lode on the field, from the surface downwards as far as the lode has been worked—over 2,000 ft. Native gold-amalgam was found on one mine on the field in the oxidised zone. The unoxidised ore always contains pyrites. I would also like the geologists to tell us why the coarsely crystallised pyrites are always poor and the very fine always good. The quartz in the lode down to, say, 1,000 ft., was of a more vitreous nature than below that point: with greater depth it became more opaque, jaspery, and harder. Secondary felspar-quartz cross-veins, all dipping at an angle of about 45° northwards, are found in the better portions of the lodes, and seem to have had something to do with the distribution of values.

In the early stages of the field there was the usual uncertainty as to the capacity of the mills and other mining plant that would be required. Owing to the great expense of carriage and the scarcity of water, the first plants erected were very small, and a great deal of anxiety was experienced in running them on account of the water difficulty. The dense subterranean salt water obtained was good for amalgamation, but was very severe on iron and steel, and when it had to be condensed for steaming or drinking purposes it was very expensive in fuel and upkeep of condensing plant. The proportion of solids in the salt water varied from 5% in the high ground at the north end of the field to 17% in the lake-bed, and, owing to the scarcity of water and its density, the simplest form of condenser plant was found to be most suitable. Experience proved that the best type of condenser was a boiler (cylindrical or otherwise in shape) built of, say, ¼-in. steel plates, set in brick or stone work. From this the steam was led into galvanised iron piping or cylindrical vessels which provided the condensing surface. These plants were practically renewed every two years.

The mills were of the simplest form at the beginning, and light stamps answered very well for crushing soft schistose ore, the duty per stamp being frequently six tons per day. The stamps with copper-plated tables, followed by amalgamating pans, were sufficient to extract all the gold amenable to amalgamation. The pulp, after pan-grinding, was very slimy when the ore was schistose, and about one-third of the original assay value of the ore remained in it.

The art of cyaniding gold ore was in its infancy then, and up to that time had practically been confined to the treatment of material that was suitable for percolation. It was, therefore, necessary to devise some other method of cyanide treatment for such residues as were not suitable for percolation, and this problem engaged the close attention of the chemists and engineers of the field for some time, until a system of agitation and filter-pressing was evolved.

The next trouble was in connection with the ore below the oxidised zone. Early in the history of that portion of the field known as the 'Golden Mile,' telluride of gold and silver was discovered, associated with other tellurides and iron pyrites and a large percentage of carbonate of lime. Owing to the oxidised ore being soft and easily milled it was soon exhausted, and the choice of a treatment method and plant for the unoxidised ore had to be made quickly. After much laboratory experimental work by the metallurgists two practical schemes of treatment were evolved, now known locally as the 'wet crushing' and the 'dry crushing' methods. In the first method the ore had to be ground exceedingly fine, amalgamated, and closely concentrated, after which the cyanide salt used as a solvent for the gold contained in the ore freed from the concentrates, required the assistance of an oxidising agent before a satisfactory recovery could be made. The concentrates could be sold to smelters or roasted on the mine and then treated along with the rest of the ore. Zinc turnings were chosen to precipitate the gold from the cyanide solution.

In the second method it was not necessary to grind the ore quite so fine, and no concentration was needed, but it had to be roasted until sweet, and after amalgamation, cyanide of potassium or sodium was sufficient to make a satisfactory extraction without the aid of any other salt. As in the wet process, zinc turnings were chosen to precipitate the gold.

For wet crushing, the choice of plant was fairly easy. The coarse ore was put through gyratory crushers, this type of crusher having first become a favourite owing to the oxidised ore not having so much tendency to stick in its mouth as in the jaw-crusher type, but later on some large crushers of the jaw-breaker type were installed. The old well-known stamp-mill was at once adopted for crushing the ore to a certain fineness, the ore being supplied to the stamps by automatic feeders. The finer grinding was at first done by pans, but subsequently tube mills were adopted where finer grinding was found to be advantageous. In the wet mills the ore, after passing over amalgamated copper plates, is then closely concentrated, mostly by concentrators of the Wilfley type; pans are, therefore, not necessary for amalgamation. In some mills the sandy portion of the pulp is percolated, and the slimes filter-pressed; in others, the whole product is ground sufficiently fine for filter-pressing.

After passing through the pans or tube-mills the pulp is pumped into spitzkasten where the excess clear solution is drained off, leaving the pulp flowing from the bottom of the spitzkasten at a consistency of about one to one. This flows into vats which have, on most mines, a centrally-hanging spindle carrying two or three tiers of iron arms actuated by cog or worm gear to keep the pulp constantly moving. On one or two of the mines the agitation is done by compressed air. In the vats the pulp is agitated for three or four hours with cyanide solution. At the end of this time the oxidising salt (bromide of cyanogen) is added, care being taken that the cyanide solution is not too alkaline. After further agitation for three or four hours the pulp is ready for filtration.

The filter-press was introduced by Mr. J. W. Sutherland, and was adopted by all the mines, to filter the solution from the pulp. This machine also provides the means of washing the soluble gold out of the pulp. To fill the presses, montejuss were first used, but in course of time these were replaced by strong pumps, as it was found cheaper to pump the pulp into the presses than to force it in by means of compressed air. The solution from the first filters is passed through another filter in order to clarify it before passing through the zinc-boxes. Charcoal, zinc fume, and finely granulated zinc were tried as precipitants, but were not found to answer as well as zinc turnings.

The gold slimes and short zinc obtained in the clean-up of zinc-boxes are put into a lead-lined vat where the zinc is dissolved by sulphuric acid, after which the contents of the vat are put through a clean-up press and washed with water. The cakes remaining in the press are then put into a calcining oven and heated to redness. After cooling, the calcined material is mixed with fluxes and smelted in large bottle-shaped crucibles fixed in tilting furnaces. Where the quantity of gold to be smelted is large, these furnaces save a great deal of labour, time and fuel. The gold bullion is practically freed from base metal and poured direct into moulds from these tilting crucibles. Samples of the bullion are obtained by boring the top and bottom of the bars. Dip samples were found to be unreliable, especially when the proportion of silver in the bullion was large, or when the sample was taken from a large quantity of liquid bullion.

The residue cakes in the filter-presses, after being dried by blowing compressed air through them are dropped into trucks, which are removed by horses, locomotives, or dump bridges, the choice of means being governed by the mine's situation. In some mines the cakes are dropped on to belt-conveyors which discharge on to a belt rising at an angle of not more than 30° to the point of delivery. This discharge belt is supported by the dump and is lengthened as the latter increases. For belt-conveying the residues require more drying than in other methods, to prevent them sticking to the belt.

In dry-crushing plants, gyratory rock-breakers are mostly used for coarse and medium grinding. For fine-grinding ball and griffin mills are used. Other appliances have been tried and discarded. There is very little difference between the cost of crushing in griffin and ball mills when the fineness of the product is taken into consideration. The ore is ground to a fineness which will allow 75% of it to pass through a 150 mesh sieve, and it is then roasted until not more than 0·05% of sulphur, as sulphide, is left in it. The roasting is the most delicate operation in the dry-crushing method of treatment, and it is made more difficult by the varying percentage of pyrites (1½ to 6) and also by the large amount of lime carbonate in the ore, about 12%. If the heat is too high or too low, soluble sulphides are formed, which cause the precipitation of gold in the subsequent treatment of the pulp by agitation with cyanide solution, and a method of re-dissolving this precipitated gold while the treatment in the vats is proceeding has not yet been found.

After roasting the ore is cooled and mixed with tail solution from the zinc-boxes and then run into grinding-amalgamating pans where the gold that will amalgamate is caught, after which the treatment of the pulp and gold solution is similar to that in the wet process, except that the addition of an oxidising salt to assist the cyanide solvent is not required.

In treating rich telluride ore the dry process is the more profitable one, but it is not so comfortable a process as the wet, and the costs would be increased to a greater extent than in the wet process if the mines were to strike ore carrying a large percentage of pyrites."—Richard Hamilton.—*Monthly Journal of the Chamber of Mines of Western Australia*, May, 1909, p. 124. (H.A.W.)

MINING.

Electrically Driven Winding Engine.—"Electrically driven winding gears of small power have been in common use for a number of years. The motor is generally of the continuous current type, or of the three phase type, taking its power direct from the supply mains, the control being effected by means of a variable resistance inserted in the circuit of its armature or rotor. This is the simplest method of control, and the least expensive to instal, but it can only be used to advantage where the power required for haulage is a small proportion of the total power generated, owing to the heavy load fluctuations due to frequent starting and stopping, and the large amount of current required for acceleration. Main shaft haulage generally requires a very large proportion of the power generated on a mine, and to successfully operate large electric winders load balances of some kind must be adopted. One of the best known load balances is the chemical storage battery. Some of the first large electrically driven winders installed were operated by continuous current motors, working in conjunction with storage batteries. A good example of this class of plant is in use at the Grangesberg iron mines in Sweden. The mine receives its power from a waterfall about 18 miles distant, from which three-phase current at 8,500 volts pressure is transmitted, and is transformed down to 430 volts at the mine sub-station. A motor generator is installed to provide continuous current supply for the winding engine. The motor-generator runs continuously, charging a battery of 243 cells having a capacity of 600 ampere hours at a very high discharge rate. The battery operates in parallel with the motor-generator when winding is in progress. The winding drums are driven by two 500 b.h.p. motors directly coupled to the drum shaft. The plant is designed to haul a load of 10 tons, including the weight of the cage and rope, at a speed of 1,200 ft. per minute. The time for acceleration is about 10 seconds, and the current reaches a maximum of 1,300 amperes for a few seconds, about one-half of which is supplied by the battery. The control is of the rehostatic type, and the heavy switching gear is operated by compressed air controlled by one lever.

Storage batteries have of late years been discarded in favour of fly-wheel storage systems, which give just as good efficiency as the chemical battery, with the advantage of cheaper price for the output and small cost for maintenance and repairs. At the present time a considerable number of large electric winding engines are operated by the Ilgner system. This system is an adaption of the well-known Ward-Leonard control, with a storage fly-wheel. Briefly, it is as follows:—The winding motor is of the continuous current type, and draws its supply from a special shunt-wound generator, which is used for this purpose only. This machine runs at practically constant speed. A small continuous current machine provides the exciting current for the generator, and also for the winding motor. This exciter is necessary in order to insure stability of the field, and therefore of the pressure of the generator, which is so arranged that, by varying the excitation of its field, it is possible to give any pressure from its maximum through zero to its maximum value in the opposite direction. The current from the generator is taken direct to the winding motor, without passing through any fuses, switches or controlling gear, the control being effected by light switching gear, dealing only with the small excitation currents (which is only 2% or 3% of the output of the generator), and is operated by a single lever. The generator is direct coupled to the shaft of a three phase motor, which takes its power from the supply mains. The storage fly-wheel is also mounted on this shaft. The motor-generator set is kept running continuously. By moving the control lever forward from the centre stop, acceleration of the winding motor is produced in one direction; by moving it backwards from the centre stop, acceleration is produced in the opposite direction. When the lever is returned to the centre stop, breaking immediately occurs, since the motor which is still running pays back current to the generator of which voltage has already been lowered. Consequently there is no loss of power in breaking. At each acceleration the motor-generator set begins to slow down, and the flywheel gives out a portion of its stored energy; when breaking occurs the speed rises, and the flywheel absorbs the power to be given out again at the next acceleration.

The Westinghouse converter equaliser differs from the Ilgner system, in that it deals only with the overload and under-load energy, and its special feature is that an alternating current winding motor may be used, and should the balancing set break down at any time the winding engine may still be kept running under the ordinary conditions of rehostatic control. The winding motor takes its supply direct from the power house, the control being effected by the ordinary rotor rehostat. The converter equaliser may be placed at any desired point along the transmission lines; it is not necessary to have it near the winding engine. It consists of a rotary converter, which is connected to the transmission mains, and on its direct current side to a direct current machine with storage fly-wheel, and is compounded in a special way, so as to automatically compensate for the magnetising currents of the induction motors in the system. The voltage of the direct current fly-wheel machine is controlled automatically by a quick acting regulating apparatus from the transmission mains through series tranformers. This enables the converter equaliser to discharge energy into the supply system whenever the load of the latter is greater than the constant station output, and to store up energy in the fly-wheel whenever the power demand in the supply system is less than the constant station output. Thus the converter equaliser not only balances the winding load, but equalises the load of the whole system.

Electrical winding plant must be selected with due regard to the special conditions of the load with which it has to deal. For shafts where the loads are heavy and hauling is fairly continuous for 24 hours per day the Ilgner system is, at the present time, the most suitable. In cases where hauling is done on one shift only the converter equaliser has its advantages, as the balancing set could be shut down at the end of the hauling shift, and the lighter loads of men, timber, tools, etc., would be dealt with under the ordinary conditions of rehostatic control. On deep shafts, where comparatively small quantities of material are raised, the series parallel system is occasionally adopted. In this case two continuous current motors are coupled to the winding drum shaft. At starting, the motors are connected in series with each other, with suitable resistance which is cut out until the full half potential is supplied to each motor when half speed is attained. The motors are then connected in parallel with a resistance in circuit, which is gradually cut out until the motors have full potential and run at full speed. The starting current taken by the series-parallel system is only about half of that taken by a simple rehostatic control plant, and the amount of energy dissipated in resistance is correspondingly reduced. An electrically driven winder of the series-parallel control type was recently installed on the New Moon Mine, Bendigo, and, according to press reports, is giving great satisfaction.

The bad efficiency of the steam winding engine is mainly responsible for the introduction of its electric rival. The average steam winder may take anything from 100 to 200 lb. of steam per effective horse power in the shaft; very few consume less than 100 lb. per effective horse power. This is not remarkable when one considers the conditions of the load. The power required for acceleration is greatly in excess of the average amount required for the trip, so the engine must either be designed for the maximum power demand or run under wasteful conditions of valve setting. (The latter alternative is usually adopted on account of smaller capital cost.) For a considerable portion of its time it has to run on very light loads; it is very often idle for long periods, yet steam must be kept at the engine continuously; consequently radiation and condensation losses are enormous. These are the worst possible conditions for a steam drive, and it is here the electric winder scores. It receives its power from the generating station, where the steam is made use of by engines of high efficiency running on a constant load, as the generators supplying the winder also provide current for the general power requirements for the mine. There are no stand by losses, for when the winder is not required the power supply is cut off, yet it is available for instant use. The possibility of waste due to careless driving is considerably reduced. In fact, with the Ilgner system, it is almost impossible, as any excess of power delivered to the winding motor during the trip is paid back to the fly-wheel when the winder is being brought to rest. At the steam generating plant considerable savings are effected apart from the actual consumption of the winder, for, owing to the plant having to deal only with an average load, the boiler capacity can be reduced and a much better evaporation obtained, thereby increasing the efficiency of the whole system. Added to these direct economies are the smaller savings of less wear and tear on ropes, cages, and shaft runners owing to the perfect motion of the motor.

The power consumption of an electric winder under ordinary working conditions may safely be taken at 1·5 kw. per effective horse power hour in the shaft. This figure is proved in everyday practice by a number of plants in operation at the present time. The steam consumption depends to a certain extent on the amount of current generated for purposes other than winding, as with large generating units and heavy outputs better steam efficiency is obtained. With fairly small sets, however, say, 400 to 600 k.w., running on a 75% load factor, a steam consumption of 24 lb. per k.w. can easily be relied upon, and may be taken as a fair basis for estimation with winding plant of moderate size. As a steam consumption of 24 lb. per unit generated, an electric winder will therefore take only 36 lb. per e.h.p. hour in the shaft. On mines where large quantities of mineral are raised from deep shafts, much smaller steam consumptions than this are obtained. At the Grand Horner Collieries, in Belgium, where there are two large electric winders hauling 1,400 tons of coal per day from a depth of 2,300 ft., the steam consumption is 32 lb. per e.h.p. hour in the shaft This plant is a pure three-phase system, and no load balances are used.

At the Zollein Mine, in Germany, where an electric winder of the Ilgner type hauls 2,500 tons of coal per day from a depth of 1,000 ft., the steam consumption is 31 1-3 lb. per hour. A still lower figure was obtained on a 24 hours' test under ordinary working conditions with an Ilgner plant at the De Wendel Colliery, in Germany. The coal hauled during the trial was 3,000 tons from a depth of 2,310 ft., and the steam consumption was only 25 lb. per e.h.p. hour in the shaft, including the winding of 1,000 men, tools, timber and rope inspection. The fact that many European collieries have adopted electric winders is the best proof of their reliability and efficiency, for there the conditions of hauling are of the most severe description, and as the coal used for power generation is cheap, the amount of fuel saved must be very great to induce colliery owners to dispense with plant which, apart from economy, has done good service for many years.

The question of electric versus steam winding may be answered in a few words based on well-known facts. The savings of an electric motor over a steam engine are greatest on intermittent work, and especially where frequent starting on heavy loads occur. It is on this account that electricity is so successfully applied to street tramway work. For the same reason winding may be considered the most suitable load for the application of electricity on a mine."—H. H. Weymouth, Scientific Society of Broken Hill, Australia. — *Australian Mining and Engineering Review*, May 5, 1909. (A. R.)

MISCELLANEOUS.

Large Units in Mining.—"The knowledge acquired since the inception of operations on these fields has proved that the formation with which we are dealing is remarkably regular over large areas, but that such regularity cannot be implicitly relied on in any particular but small section. Acting on this knowledge, larger blocks than have in the past been assigned to any one company are necessary. Regular returns on capital are thereby ensured. By developing large bodies of ore, selective mining is avoided as much as possible, and concentration of work averts constant removals of the labour force. Thus underground costs are reduced, especially under the heads of development, stoping and shovelling, and tramming, to the lowest possible limit with due regard to efficiency. Whilst it may be remunerative to instal mechanical appliances for the handling of ore underground where levels contain considerable tonnages, it is not advisable to do so for small tonnages, the cost for amortisation and shifting of plant being too heavy. Fluctuations of yield, which have to be counteracted in small areas by selective mining, can be obviated by large reduction works, which, again, are only justified if the total tonnage to be mined warrants the expenditure. Development work becomes less costly as the number of drives and stations can be reduced. No doubt in some instances shareholders have been perturbed, especially in cases where immediate returns have been somewhat decreased. They forget, however, that the altered conditions have an important bearing on the question of amortisation of investments, and that the total return on their capital is in every case in reality increased. They receive regular dividends, and in most cases where amalgamations have taken place need not for a number of years provide any redemption fund. The considerations so far given are from the shareholders' point of view only. The public and the country generally, however, are no less gainers. It becomes possible to extract large bodies of ore which otherwise would be left standing, as the gold contents would not pay mining and reduction expenses under conditions hitherto existing. As a consequence, the total amount expended in wages and stores is greater, though probably the amount so expended per ton is smaller. The wage-earning and spending capacity of the fields become more diffused —that is, they are spread over a longer period and over a larger number of men, as the proportion of skilled and semi-skilled work called for under the altered conditions becomes greater in proportion to unskilled work. Objections have been raised on the grounds that the number of white workers will be less, and that the country itself derives greater benefits by the expenditure of a larger proportion of the final yield in the shape of cost of production. I have already replied to the first of these objections by stating that the altered conditions created will gradually call for more skilled and semi-skilled work in proportion to the amount of unskilled work. Taken as a whole, the large producers with low working costs employ more than the average proportion of white men as compared with coloured labourers. In December last the nine mines which crushed over 40,000 tons each employed one white man for every eight coloured men, whilst the ten companies crushing 10,000 tons each or less in that month had ten coloured men for every white man employed. The provision of unskilled labour for the industrial development in and around Johannesburg, and its management, have absorbed more time and attention than almost any other portion of our work. At the present moment the producing mines of the Witwatersrand are using the energies of 2·68 coloured labourers for every ton of ore crushed per day. The number of native labourers available in Africa for work in the mining districts of the Witwatersrand is limited, though expansion of the numbers at present employed is no doubt possible. Any saving effected in the use of labour must also be considered a gain, as it would free men for employment in other directions. The second objection is of a more dogmatic character. If the quantity of ore to be extracted were definitely known to be of a certain value, the advocates of the policy of distributing as much as possible of the value so ascertained in wages and stores would no doubt have a good deal of justification for their argument. The quantity of ore, however, is practically unlimited; its average value may be higher or lower than that of the ore mined and exposed to date. It certainly is lower than the average of the total tonnage actually treated, owing to selective mining in the past. It is impossible to estimate either tonnage or value, but it is certain that a considerable portion would never be extracted at the cost of production prevalent, even a few years ago. It is a truism to say that the larger the margin of profit the more easily will capital be attracted. For years past, up to about a twelvemonth ago, even the most tempting mining propositions on the Witwatersrand failed in their attempts to secure funds. Recent developments and reductions in expenses have brought about a welcome change in the attitude of the money markets of the world, and as a consequence quite a number of properties which had lain dormant have been re-opened. Not only the number of men employed, but also the wage-earning and spending capacity of the district has thereby been increased.—MR. REYERSBACH.—*South African Engineering*, May, 1909, p. 81. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE THEORY OF VALENCY. By J. NEWTON FRIEND, Ph.D., M.Sc. Text-books of Physical Chemistry, edited by Sir William Ramsay. (London, 1909: Longmans, Green & Co., 39, Paternoster Row, E.C.) Pp. 180. Price 5s.

"In this work the author presents us with a compact summary of the various theories of valency and structure from the days of Berzelius' and Davy's theories down to the present time. In some of the chapters the more important theories are considered in their general relationship to chemical facts, whilst in other chapters the author deals with the particular elements grouped according to the periodic classification. A special chapter is devoted to the electrochemical theories. Although the author has produced an undoubtedly useful work, and has compressed a large amount of scattered knowledge and speculation into a relatively small space, one cannot help being somewhat disappointed with the result. There is too much of the ancient business of juggling with strokes on paper, and not nearly enough of that quantitative modern physical and electrochemistry which has done so much to bring the valency question within the sphere of exact science. Remsen's trivalent halogens, for example, are a continual source of comfort to the author, and so is tetravalent oxygen. Mercurous chloride is $Hg-Cl=Cl-Hg$, hydrofluoric acid is $H-F=F-H$, and so on. When dealing with water the author says: 'If the molecules are double, the formula is clearly $H_2=O=O=H_2$.' One had thought the days of this sort of formula-juggling were over.

Attention must be drawn to a striking omission. There is no mention of the work of Baly and his collaborators, or of Kauffmann, and yet surely their work is most intimately connected with the subject of valency.

In the chapter on electrochemical theories a good account will be found of the 'electronic' theory of valency. This is, perhaps, the most interesting part of the book, although highly speculative.

In conclusion, it may be said that the author has tried to do justice to many theories, and, if the result is not entirely satisfactory, this is perhaps due to the extent of ground covered in a book of small compass. In spite of this compression, the present work is the most exhaustive monograph on the subject of valency which exists, and so is eminently worthy of careful perusal by every chemist and physicist."—*Transactions of the Faraday Society*, April, 1909, p. 207. (H. A. W.)

THE PRECIOUS METALS, COMPRISING GOLD, SILVER AND PLATINUM. By T. KIRKE ROSE, A.R.S.M., D.Sc. With 46 Illustrations. 8 × 5½. pp. xiv.-295. Price 6s. net. (London: Archibald Constable & Co., Ltd., 1909.)

This book is particularly interesting to all whose attention is engaged in the extraction or manufacture of the precious metals, and especially to members of the Chemical, Metallurgical and Mining Society of South Africa, of which society, as the title page sets forth, the author is an honorary member. The aim of the work, as stated in the preface, is to provide an introduction to the study of the precious metals and an elementary book of reference for those who do not wish to pursue the subject further. With these objects in view details have generally been omitted to enable the author to include such a wide range of view as in involved in the history, metallurgy, minting and manufacture of gold, silver and platinum, with a description of their principal alloys and chemical compounds.

The historical section is particularly good, and it is astonishing to find what great antiquity may be attributed to such useful devices as cupellation, fine grinding of ore and the use of mercury. Incidentally the picture given of mining in ancient Egypt, where the working day appeared to consist of 24 hours with no cessation for meals or rest, though it evidently did not stagger the humanity of those times, has little of encouragement for advocates of state-owned mines.

The diagrams of freezing points of various alloys are useful and interesting, as are the numerous photo-micrographs also given. The sketch given of the metallurgy of gold is illustrated by some excelent photographs of the Knights Deep and other

Rand gold mines, for which the author makes acknowledgement to Mr. W. A. Caldecott.

A careful description of the various methods in use for the refining of gold and silver bullion and the minting of standard alloys may yet prove of peculiar interest in this country, and Dr. Rose's statement that bullion assays should be accurate to ·05 of a millieme warrants the particular attention of London refiners. Under the heading of platinum the author mentions the fact that in ordinary furnaces platinum cannot be entirely freed from lead by cupellation for which the oxy-hydrogen flame is necessary.

The carbonyl of platinum chloride $PtCl_2CO$ is described, and its property of volatilising at 250° C. without decomposition. If a similar salt of gold existed it should be particularly interesting to cyanide operators as a possible source of loss in calcining gold slime. The world's production of gold and silver annually is given, and also an estimate of the present stock, and a perusal of these tables gives rise to the reflection that as our annual output will soon exceed the total quantity of gold coin in circulation in England, a time in the not infinitely distant future may come when an annual increase may take place in the cost of production due to the effect on prices of producing more gold than increasing trade requires for exchange purposes or than art and luxury require for use and ornament.

It will be seen that this book should appeal to a wide circle of readers, and is certainly entitled to a place on the book shelves of all concerned in the handling of precious metals—and in certain forms who is not. (H. A. W.)

THE DRESSING OF MINERALS. By HENRY LOUIS. Pp. 550. Illustrated. Price $8.50. (New York: Longmans, Green & Co.) For sale by *The Mining World.*

"A work designed for the use of students, treating of the principles and practice of the preparation of ores, etc., for metallurgical treatment or for the market. The work supplies a want in the literature on the subject. While, in a book of this size, and treating of so large a subject, it is obviously impossible to treat exhaustively of the theoretical considerations involved in the various departments of ore dressing, and to deal minutely with the details of construction of all the machines and devices that are used, a fairly complete and practical review of the entire field is presented in a clear and logical manner.

The first 478 pages are devoted to descriptions of machines for performing the various operations, such as sizing, sorting and washing, comminution, separation of materials by various means, etc. The machines selected for description are taken to illustrate types. Although the book is prepared for English students it is gratifying free from bias in favour of British practice, American and continental manufacturers and authorities being liberally represented.

The last chapter is devoted to the consideration of the general construction of dressing works.

The volume is bound in cloth and the typography is excellent. The text is clear and supplemented by numerous illustrations." — *The Mining World* (Chicago), June 5, 1909, p. 1081. (W. A. C.)

Ashcroft, G. A. A Study of Electrothermal and Electrolytic Industries. Part 1 Introductory. Cr. 8vo. *Spon.* Net 8s. 6d.

Baron, Harold. Chemical Industry on the Continent and Report to the Electors of the Gartside Scholarships. 8vo., pp. 86. *Sherratt & Hughes.* Net 1s.

Berkeley, Earl of, Hartley, E. G. J, and Stephenson, J. On the Osmotic Pressures of Calcium Ferrocyanide Solution. Part 2 Weak Solutions. 4to. sd. *Dulau.* Net 1s.

Bohr, N. Determination of the Surface-Tension of Water by the Method of Jet Vibration. 4to. sd. *Dulau.* Net 2s.

Brown, William. Mechanical Stress and Magnetisation of Iron. Part 2, being the Scientific Proceedings of the Royal Dublin Society. Vol. 12. New Series, No. 17. *Williams & Norgate.* 1s.

Chatley, H. The Force of the Wind. 12mo., pp. 83. *C. Griffin.* Net 3s.

Coffin, J. G. Vector Analysis. Cr. 8vo. *Chapman and Hall,* Net 10s. 6d.

Daw, Albert W., and Zacharias, W. The Blasting of Rock in Mines, Quarries, Tunnels, etc. 2nd Edition with Addenda. 8vo., pp. 294. *Spon.* Net 15s.

Garcroft, W. E. Suggested Rules for Recovering Coal Mines after Explosions and Fires. 12mo. pp. 71. *Colliery Guardian Co.* Net 3s. 6d.

Garrett, A. Hydraulic Tables and Diagrams for Practical Engineers. With 24 Diagrams. Obl. fol., pp. 48. *Longman.* Net 15s.

Giekie, Sir Archibald. Charles Darwin as Geologist. The Rede-Lectures. Cr. 8vo., pp. 96. *Cambs. University Press.* Net 2s.

Hoover, Herbert C. Principles of Mining, Valuation, Organisation and Administration. With 50 Illustrations. 8vo., pp. 200. *Hill Publishing Co.* Net 10s. 6d.

Iddings, J. P. Igneous Rocks. Vol. 1. 8vo. *Chapman & Hall.* Net 21s.

Louis, H. The Dressing of Minerals. With over 400 Illustrations. Roy. 8vo., pp. ix-544. *Arnold.* Net 30s.

Mehrtens, A. C. Gas Engine Theory and Design. 8vo. *Chapman & Hall.* Net 10s. 6d.

Miers, Henry A., and Isaac, Florence. On the Spontaneous Crystallisation of Monochloracetic Acid and its Mixtures with Naphthalene. 4to. sd. *Dulau.* Net 2s. 6d.

Molesworth, Sir Guildford L. Metric Tables. 4th Edition, Revised and Enlarged. Obl. 18mo., limp. *Spon.* Net 2s.

Rise and Progress of the British Explosives Industry. Published under the Auspices of the 7th International Congress of Applied Chemistry by its Explosives Section. Illustrated. Royal 8vo., pp. xiv.-418. *Whittaker.* Net 15s.

Rose, T. Kirke. The Precious Metals. Comprising Gold, Silver and Platinum. 8vo., pp. 312. *Constable.* Net 6s.

Stone, C. H. Practical Testing of Gas and Gas Meters. 8vo. *Chapman & Hall.* Net 15s.

The National Physical Laboratory Report for the Year 1908. Imp. 8vo. pp. 115. *Parrot* (Teddington).

Twelvetrees, W. Noble. Simplified Method of Calculating Reinforced Concrete Beams. Cr. 8vo. *Whittaker.* Net 6d.

Wanhill, C. F., and Beveridge, W. W. C. The Sanitary Officers' Handbook and Practical Hygiene. Cr. 8vo., pp. 158. *E. Arnold.* Net 5s.

Whetham, W. C. D. The Recent Development of Physical Science. 4th Edition with Additions. Cr. 8vo., pp. xvi.-347. *J Murray.* Net. 5s.

Yu Wang (Chung) Antimony. Its History, Chemistry, Mineralogy, etc. For Students, Manufacturers and Users of Antimony. (Scientific Text Books.) 8vo., pp. x-217. *C. Griffin.* Net 12s. 6d.

Zsigmondy, R. Colloids and the Ultra-Microscope. Cr. 8vo. *Chapman & Hall.* Net 12s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 347/09. G. K. Chambers (1), S. H. Eckmann (2), The British Westinghouse Electric and Manufacturing Company, Limited (3). Improvements relating to electrical hoists. 26.7.09.

(P.) 348/09. J. H. Skinner. Improvements in the manufacture of iron poles for telegraphs, telephones, etc. 26.7.06.

(P.) 349/09. James Paul Griffiths. A new or improved clutch or fastener and parts therefor, for windows, fanlights, shutters and doors. 27.7.09.

(P.) 351/09. Charles Reeves. Improved method for the recovery of gold from liquids and the like. 28.7.09.

(C.) 352/09. John Pomery. Improvements in steam boiler furnaces. 28.7.09.

(P.) 353/09. Daniel Chilcote King (1), Augustine John Danvers (2). Improvements in machines or apparatus for scouring battery plates, also applicable for other analogous purposes. 30.7.09.

(P.) 354/09. Daniel Chilcote King. Improvements in protective receptacles for the holding of amalgam. 30.7.09.

(C.) 355/09. Jose Baxeres de Alzugaray. Improvements in the art of extracting metal from their ores. 30.7.09.

(P.) 356/09. Henry Mechan. Improvements in or connected with discharge aperture fittings of cyanide and other tanks. 30.7.09.

(C.) 357/09. Thomas Ferguson. Improvements in and relating to dynamo electric machines. 30.7.09.

(C.) 358/09. Charles Ridley. Improvements in the methods of obtaining gases from solids, and in apparatus employed therein. 30.7.09.

(C.) 359/09. Oliver Brown Dawson (1), John Bauernschmidt (2). Improvements in and relating to metallurgical processes. 31.7.09.

(C.) 360/09. Henry Selby Hele-Shaw. A variable stroke fluid machine. 3.8.09.

(P.) 361/09. Frank Ondra (1), Alfred Adair (2). Improvements relating to the treatment of zinc solutions for the recovery of metals or metal compounds therefrom. 6.8.09.

(P.) 362/09. Antony Maurice Robeson. A new and improved system of reinforced concrete shaft sett framing. 7.8.09.

(P.) 363/09. David Landale (1), Lewis Grant (2), Lewis Campbell Grant and Charles Edward Douglas trading as Douglas & Grant (3). Improvements in winding and hauling engines. 7.8.09.

(P.) 365/09. Malcolm Stewart Beaton (1), James Eason (2). Improvements in lubricators. 10.8.09.

(P.) 366/09. Edwin Rice. Improvements in means for attaching trucks or other vehicles to a hauling rope or the like. 11.8.09.

(P.) 367/09. Percy Herbert Partridge. A processs for seasoning of timber grown in South Africa which likewise prevents it cracking and twisting. 12.8.09.

(P.) 368/09. William Berthelson. The extension machine bar. 13.8.09.

(P.) 369/09. Thomas Parker. Improvements relating to plant for the destructive distillation of coal. 13.8.09.

(C.) 370/09. Franz Moritz. Disc coupling for cranks of engine shafts. 13.8.09.

(P.) 372/09. John Russell Cawthorne. Rimer for riming battery heads when buried, owing to shoes coming off or otherwise. 14.8.09.

(C.) 374/09. John Richard Williams (1), Henry Wentworth Bradley (2), Benjamin Bradley (3). Improvements in obtaining zinc or copper from complex ores or the like and in electrolytic cells especially applicable thereto. 14.8.09.

(C.) 375/09. Marcus Ruthenburg. Improvements in or relating to magnetic separators. 17.8.09.

(C.) 376/09. Marcus Ruthenburg. Improvements in or relating to electric furnaces for the treatment of ores and the like. 17.8.09.

(P.) 377/09. William Henry Thompson. Improvements in amalgamating pans. 18.8.09.

(P.) 378/09. William Calder. Improvements in means for discharging comminuted matter from tube mills. 18.8.09.

(C.) 379/09. Hans Siegwart. An improved process and apparatus for the production of cylindrical and like bodies from concrete. 20.8.09.

(C.) 380/09. John Bedington. Improvements in air and steam rock drills. 21 8.09.

(C.) 381/09. John Sims Forbes. Improvements in apparatus for heating and sterilising water or other liquids. 21.8.09.

(C.) 382/09. John Warne Phillips. Centrifugal separator. 21.8.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

ADAMS, E. L. *l/o* Johannesburg; Sabi Mine, Belingwe, Rhodesia.
BEATTY, G. H., *l/o* Johannesburg; Cinderella Deep, Ltd., P. O. Box 75, Boksburg.
BOYD, A., *l/o* Fordsburg; Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs.
ELLIS, B., *l/o* Johannesburg; New Rietfontein Estate G. M., Ltd., P. O. Rietfontein Mines.
GIBSON, J., *l/o* Johannesburg; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.
GRUMITT, P. H., *l/o* Mazoe; Witwatersrand Deep, Ltd., P. O. Box 5, Knights.
JOHNSON, H. H., *l/o* Village Deep, Ltd; c/o R. R. Hollins, Esq., Standard Bank Buildings, P. O. Box 289, Johannesburg.
MCCRINDLE, J., *to* 15, Long Street, Germiston West.
PENCIER, H. P. DE, *l/o* Roodepoort; 110, Wellington Street, Ottawa, Canada.
ROBERTS, S., *l/o* Krugersdorp; Midas Deep, Ltd., P. O. Box 8, Randfontein.
WARD, J. P., *l/o* Johannesburg; Aurora West United G. M. Co., Ltd., P. O. Box, Maraisburg.
WAYNE, T. H. B., *l/o* Durban; Claremont Mine, Insiza, per Private Bag, Bulawayo.
WILEY, H., *l/o* Salisbury; Selukwe G. M. Co., Ltd., Selukwe, Rhodesia.
WILLIAMSON, R., *l/o* Kalgoorlie; Great Boulder No. 1, Limited, Mount Magnet, W. Australia.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. X. SEPTEMBER, 1909. NO. 3.

Proceedings
AT
Ordinary General Meeting, September 18, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, September 18th, Mr. McArthur Johnston (President) in the chair. There were also present:—

56 Members: Dr J. Moir, Messrs. C. B. Saner, M. Cullen, K. L. Graham, H. A. White, Prof. J. A. Wilkinson, A. F. Crosse, W. Cullen, E. H. Johnson, R G. Bevington, Prof. J. Yates, H. A. Adams, G. S. S. Anderson, W. Beaver, W. Broom, S. Caris, B. P. Carter, J. Chilton, D. Coll, J. B. K. Dodds, T. Donaldson, J. Gray, C. B. Hilliard, T. Johnson, J. Kennedy, G. T. Lawson, R. Lindsay, Dr D. Macaulay, M.L.A., R. MacGregor, J. E. Metcalf, H. H. Morrell, J. W. Morris, M. T. Murray, S. Newton, W. Nicklin, G. Nicol, F. D. Phillips, H. W. Pidsley, E. T. Rand, L. J. Robinson, W. H. Roe, H. A. Scarf, W. Sharp, S. Shlom, W. H. Smith, T. W. Simmons, S. H. Steels, J. Taggart, J. A. Taylor, A. Thomas, C. Toombs, J. M. Thorburn, J. F. Walker, P. H. M. Whyte and L J. Wilmoth.

16 Associates and Students: Messrs. A. R. Adams, G. F. Ayers, J. Cronin, G. W. Fox, D. W. Greig, W. J. R. Hunter, R. L. Impey, H. L. Jupp, E. J. G. Medlin, G. Musson, F. J. Pooler, H. B. Powter, J. M. Robinson, P. Scatterty, H. Stadler, and W A. C. Tayler.

15 Visitors, and Fred. Rowland. Secretary.

The minutes of the previous meeting, as printed in the August *Journal*, were confirmed.

NEW MEMBERS.

Dr. D. Macaulay, M.L A., and Mr. A. F. Crosse were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Bisset, Arthur Charles, East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand. Surveyor.

Daniel, James, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Surveyor.

Fox, George C., Messrs. Lewis & Marks, P. O. Box 2567, Johannesburg. Consulting Mechanical Engineer.

Herring, Frank, Crown Mines, Ltd. (L.D.), P. O. Box 5, Fordsburg. Cyanide Foreman.

Simpson, Charles Butement, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Foreman. *(Transfer from Associate Roll.)*

Sly, Alfred William, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Surveyor.

Solomon, Lionel, P. O. Box 503, Johannesburg. Surveyor.

Taylor, Robert McGeown, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Mill Manager.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Bell, Trevor Griffith, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanider.

Cawthorne, John Russell, Cinderella Deep, Ltd., P. O. Box 82, Boksburg. Amalgamator.

Coalbank, Vincent Richards, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider.

Dickson, Victor Richard, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider.

Lean, Thomas Frank Rodgers, Occidental G M. Co., Lewis St., Cobar, New South Wales Assayer.

Nalty, Frederick John, P. O. Box 253, Bulawayo, Rhodesia. Cyanider.

Plinke, August Adolph Philipp, Transvaal Bank Building, P. O. Box 273, Johannesburg. Director of Mining Companies.

Pooler, Frederick John, B.Sc., Jeppe High School, 358, Commissioner Street, Johannesburg. Science Teacher.

Thomas, Victor Israel, Wrightville, Cobar, New South Wales. Cyanider.

As Students.—

Brennan, Ernest, New Kleinfontein G. M. Co., Ltd., P. O. Box 2, Benoni. Cyanide Learner.

Coombe, Oswald, Cinderella Deep, Ltd., P. O. Box 75. Boksburg. Assay Learner.

Greyson, Robert, Cinderella Deep. Ltd, P. O. Box 75, Boksburg. Cyanide Learner.

Impey, Robert Lancelot, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanide Learner.

General Business.

The Secretary read the following contribution:

A Note on the Efficacy of the Cyanide Antidote.

Mr. A. L. Edwards (*Member*): During the latter part of August, 1909, twenty-five goats forced their way through the wire fence encircling the slimes residues dam of a company operating in the Klerksdorp district, and drank some of the solution. Fully five minutes had elapsed before I was informed of the occurrence. I went to the spot, taking with me the antidote and half a litre of distilled water; I dissolved the antidote in the whole of the water, and at once started dosing the goats lying nearest to hand. Unfortunately there was only sufficient of the antidote to make eight decent size doses. Of the eight dosed, five recovered and were able to be made to walk home, about one and a half miles, roughly; the sixth could only be got to walk after five hours, and the other two did not recover. A peculiar feature of the sixth goat was that it lay on the ground for fully three hours, and during that time its eyes were practically glazed; within three and a half hours from the time that the antidote was applied, the goat was got on its legs and allowed to stand. Immediately it began to circle round; round and round it went, with intervals for rest; it kept on turning round and round for fully one and a half hours; after that time it was driven home. On making enquiries next day I was informed by the owner that the six goats were up and about and eating well, so I think it is only right that their recovery should be placed to the credit side of the cost of the antidote. I am informed that the antidote had been kept on hand for fully eight months before its use was called for.

Tests of the NaCy solution made at the time of the accident showed—cyanide, KCN = 0·002%, alkalinity = 0·0024% CaO.

HARDINGE CONICAL MILL.

The President: I wish to bring to your notice a small exhibition model of the Hardinge conical mill, which members can examine after the meeting.

COLOURED WRAPPERS FOR EXPLOSIVES.

Mr. W. Cullen (*Past-President*): You will perhaps remember that some few months ago there was a discussion in this Society in regard to coloured wrappers on explosives. At one of the meetings Mr. Kotze, the Government Mining Engineer, made a suggestion which I was pleased to follow up. The suggestion he made was that in order to trace accidents on the mines and to identify the explosive used, the cartridges should be wrapped in coloured wrappers. I am very pleased to say that these wrappers have now arrived from Europe, and in the course of the next few weeks they will be ready for distribution among the mines. I shall be pleased if members of the Society under whose observation these cartridges come will note whether they are easily distinguished among the rubbish of the stopes, as one is led to believe they will be. If any of our members happen to come across them I will be glad if they will either relate their experiences here or communicate with me privately. I think they will probably obviate some of the unfortunate occurrences which have taken place on the mines.

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

The President: I would like to draw your attention to one feature which will appear in the *Journal* containing the report of this meeting. You will find the reprint of a paper by Mr. Caldecott on "The Development of Heavy Gravitation Stamps." This paper is being read before the Institute of Mining and Metallurgy in London, and by arrangement between the Societies we are allowed to publish simultaneously any paper and to bring forward such discussion upon it that any of our members may be pleased to give. I can assure you that this paper will be interesting to all of us and the Society will be indebted to anyone who cares to discuss it.

Mr. K. L. Graham (*Member of Council*) read the following paper on behalf of the author:—

A SKETCH OF THE SMALL CYANIDE PLANT AS ERECTED AND WORKED IN RHODESIA.

By F. J. Thomas (Member).

The number of cyanide plants in Rhodesia eight or ten years ago could have been counted on one's fingers; to-day they number nearly a hundred; to a great extent this increase is due to the introduction of the cheap, small plant.

The cyanide plants erected prior to 1900 closely resemble in the main features of their construction the Rand plants of that day, lacking only the capacity and perfect finish of the latter. Double-tier steel tanks standing on masonry foundations, with iron and steel supports; spitzluten, spitzkasten and, where necessary, tailings wheels were installed. The extractor and smelting houses were constructed on equally generous and permanent lines; lofty, well ventilated buildings with cement floors, the orthodox extractor boxes and offices, and equipped with centrifugal pumps, gold slime filter presses, zinc lathes, etc.

With the birth of the private syndicate, as well as of the tributor and the individual owner—the popularly styled "small man" whose growth during the last five or six years has been phenomenal—came the demand for a smaller, cheaper and simpler style of plant, something more consonant with, and having a reasonably relative cost to, the type of mill that was rapidly arising on all sides; these were 3, 5 and 10 gravity stamp batteries, Tremains, Chilians, Huntingdons, etc. In such cases the capital available for cyanide construction was told in hundreds rather than thousands. Invention—or in this instance, I fancy, imitation—promptly responded to the demands of necessity and abandoned the super-imposed steel structures, paused slightly before the rectangular sheet-steel tanks and again at the square wooden variety, then finally produced the corrugated iron vat which has now become so popular. The reasons for this popularity are apparent. The tanks, boxes and sumps are comparatively cheap and very portable, they can be easily and rapidly constructed in any locality and, when fairly well built, there is no loss through leakage. Some doubt and distrust certainly did exist in many minds as to the strength and durability of tanks of this material, but these misgivings may now be said to be dispelled; the contractors are prepared to guarantee that the strength shall be fully equal to the strain, and with reasonable care and protection they should last for a number of years, in fact if the owner could feel assured that his mine would not collapse before his tanks I imagine several small men would be considerably relieved and could look ahead with greatly increased confidence. In the few instances where these plants have failed to give satisfaction, the fault may be traced to the weakness of the material employed and to indifferent workmanship. To erect tanks of 24 B.W.G. iron is to court disaster, unless the depth is reduced to about 4 ft.; treatment tanks 14 ft. diameter and 6 ft. deep commenced to buckle almost immediately when filled; the same occurred when the depth was reduced to 5 ft. On the other hand tanks of a greater capacity but constructed of 22 gauge iron are as perfect now as when first put up some two years ago.

There have already appeared in this *Journal*[*] (Notices and Abstracts, Sept., 1907) detailed specifications of a typical small Rhodesian cyanide plant, together with an estimate of its cost, and several firms in the country now make a speciality of the construction of such plants; it is therefore unnecessary to elaborate these details again. One or two points may perhaps be emphasised; all joints should be double-rivetted and soldered both inside and outside; it is very necessary to have the top rims strengthened with a band of angle iron or gas piping, since this preserves the shape of the tank; in caulking down the overlapping portion of the filter-mat it should not be ruthlessly hammered into position, as this is liable to break the solder along the joints, and ample space should be allowed between the grating and side of the tank. The owner is responsible for the necessary foundations and excavations. For the former the ground is well levelled off, thoroughly soaked and then beaten down until it resembles the floor of an ordinary hut; a layer of sand, say 2 in. deep, upon the top of this provides a foundation that will accommodate itself to any slight irregularities and upon which the tanks will sit securely.

The success of the corrugated iron sumps and flat galvanised iron extractor boxes has been so pronounced that these have been adopted on mines which one would hesitate to designate as small. The independent compartment style of extractor box is perhaps the more popular, though the full-length partitioned box is frequently found; the circular extractor vat of any material has found but little favour.

The extractor house is invariably a humble erection and but moderately equipped; it is a very diminutive wood and iron building as seen in its most ambitious form, and is at its lowest a mere shed.

As electric power is generally considered to be a luxury and the transmission of power by wire rope not much in vogue, the centrifugal pump gives place to its steam rival, or failing that to a hand pump, say a 5 in. Challenge, though this is unsatisfactory—except for the smallest of small plants—depending, as it does, for its motive power on the "mechanism" of the somnolent Kafir.

But few, if any, of the dumps in the country now being treated are large enough to justify the introduction of methods of mechanical haulage, and when accumulated sands are being handled, the material, in trucks, or sometimes wheelbarrows, is run up an inclined track (unless the lie of the ground and relative position of the dump and tanks offer other and superior facilities) frequently formed of residues, and then on to a gangway passing over the centre or along the sides of the tanks into which the sand is directly tipped and evenly spread. When the residues are discharged they are often allowed to accumulate around and to the same height as the tanks, forming a bank on to which all subsequent residues are shovelled and then conveyed to the extreme edge; this bank thus becomes the residue dump, which gradually extends outwards along the line of least resistance. This method of discharging is undoubtedly quick and easy, but as the tanks are surrounded and embedded in sand it has the grave disadvantage of concealing any leakage that might possibly occur and the safer plan, of course, is to tram the residues clear of the foundations.

---

[*] See this *Journal*, Vol. viii., September, 1907, pp. 90-91.

With the disappearance of the original dumps, direct filling is now superseding the practice of running the tailings into dams. On the more prosperous of the small mines Frenier pumps (where necessary), spitzkasten and collecting tanks are being employed; the collecting or settling tanks being so placed that their contents can be directly shovelled over the sides into either of two, or any of three or four treatment vats.

The possibilities and convenience of the wheelbarrow in connection with the small plant may sometimes be overlooked, yet it is not to be ignored; supplied with these, ten boys can handle 1,000 tons a month, more or less, according to the condition of the sand and the distance it has to be conveyed. Of the actual cyanide treatment as conducted on these 80 odd small plants it is impossible to write at all comprehensively, the conditions and methods varying widely. There can be little doubt that if the average owner could be persuaded to avail himself more frequently of the advantage of expert advice, regarding the matter as an investment, the results, in many cases, could be improved upon. In the absence of complications produced by such minerals as copper ores, stibnite, large quantities of silver, etc., the treatment may be said to follow broadly on the lines freely recognised, I understand, on the Rand. Briefly stated the cycle of operations may be—saturation of the charge with strong solution (without, as a rule, a preliminary wash), contact, draining dry, and standing dry, followed by (*a*) further saturations with weaker solutions which in turn are drained dry, or (*b*) intermittent pumping of weak solution with continuous leaching, the treatment being concluded with one or more water washes. When it is stated that the average time of treatment seldom exceeds 84 or 108 hours as against 144 or 156 hours or more, on the larger plants, it will be obvious, that this question is an important factor: the preliminary wash, though perhaps proved to be advantageous and economical, may have to be sacrificed, and some compromise arranged between the methods of draining dry, etc., and intermittent pumping. The total tonnage of solution employed in a treatment of 84 hours is seldom more than that of the charge; when treating accumulated sands it is not infrequently less. The solutions are ærated in the ordinary way by circulation in the sumps; compressed air, vacuum pumps and other luxuries are found only on the larger mines.

The clean-up, which takes place once a month, is conducted in the frankly primitive manner of early days; the presence of a gold-slimes filter-press lends much distinction to a small plant, but the boy and bucket method is more common. With the aid of a clean-up tank of say 500 gallons capacity, the operations are conducted in a fairly efficacious, if tedious, way; after siphoning off the clear solution from the boxes and washing that zinc which will subsequently be returned for further duty, the contents of the boxes are conveyed in buckets to the clean-up tank, here the slimes settle and the solution is siphoned off and passed through the re-dressed extractor boxes. Should the clean up tank not be acid proof, the settled slimes and short zinc are scooped up and transferred to an acid vat close by—such small vats can be purchased locally—while 68-gallon "Dop" barrels make efficient substitutes. On the completion of the acid treatment the slimes are well washed by the decantation method, the clear liquor being siphoned off on to a filter composed of a double thickness of stout calico supported whilst on a blanket suspended across a frame, beneath this there may be a sand filter through which the liquor passes before running to waste. Finally the slimes are decanted and washed out on to the filter and there allowed to drain dry previous to calcining. If the usual care is exercised and ample time allowed for these operations they should be conducted without loss.

There is nothing elaborate about the construction of the furnaces, and the amateur mason rises to the occasion satisfactorily; the ordinary square, coke furnace with a lining of fire-bricks is used for smelting. In some cases the calcining is done in a tray placed on the mouth of this, or a separate calciner may be built, somewhat larger, and rectangular in shape, the fire door being in front and the tray fitted with a lid or cover, built in on the top of the fire-box. When the two furnaces are erected back to back and served by the one smoke stack the whole forms a neat, compact arrangement. Too often, however, they are very inadequately sheltered from wind and rain, some are entirely destitute of any protection, others possess a roof of sorts, but few are completely enclosed, and thus the loss from "dusting" is, on occasions, distressing to observe. Wood is used as fuel for calcining; coal, charcoal, or coke for smelting; each of the three has its disadvantages, coal requires the assistance of a forced or induced draught for satisfactory work, charcoal is consumed so rapidly that the fire needs constant attention and replenishing, thus the smelting proceeds slowly, and imported coke costs about 25s. a bag. A local coke can now be obtained costing considerably less than the English article, and this appears to be both satisfactory and economical. Though the amount of slimes to be smelted may be comparatively small there is little, if any, advantage in using small crucibles, as with Salamanders, No. 100, the smelt may be rapidly completed. Borax, silica, manganese dioxide and fluor-spar are common fluxes, liners, of course, being employed. On few small mines is the cost of cyanide treatment per ton worked out exactly, but taking into consideration the proportionate charges for power, office and compound expenses, labour fees, etc., the costs would seldom be below 3s. a ton. In an interesting paper entitled "The Small Mines of Rhodesia" (September *Journal*, 1908), Mr. B. I. Collins states that many of the cyanide plants are run without assays ever being taken. This, one must admit, has probably been the case in the past, but the statement, I think, might be modified now, as the owners are becoming more alive to the importance of this matter; special arrangements exist between the Tributors' Associations and Public Assayers. It is a common practice to lump together proportionate quantities of the samples of the week's charges, quarter down and send the resultant sample for assay, the residues being treated in the same fashion. This method may not appeal to precisians, but it is a plan not confined to the small plants of Rhodesia.

To attempt to give the average percentage extraction obtained on these small cyanide plants is to trespass on delicate ground for the sake of a mere conjecture, while there is no reason to believe that the actual figures, could they be obtained, are such as would prove deeply depressing, yet one cannot doubt that there is ample room for improvement; this knowledge should be stimulating and we are, therefore, likely to escape the impending doom of the Rand cyanide managers —the curse of ennui attending a monotonous perfection! Of the impediments to a higher extraction may be mentioned the coarseness of the sands, for, where the capacity of the mill is small, fine crushing may not be regarded with favour; in the absence of a slimes plant the elimination of the slimes has seldom been undertaken with enthusiasm; methods of concentration have been almost entirely ignored and, as previously noted, the time of treatment is frequently unduly curtailed. Considerable improvement, however, in these matters is now being effected, and the cyanide plant is esteemed as other than a mere battery adjunct to receive amateur attention in spare moments.

Individual workers and syndicates find the small slimes plant by no means such an attractive proposition as the sand plant, and in the Rhodesian output for May, only three small properties declared slimes returns. In the other cases the slimes are sometimes allowed to settle more or less in shallow pits and subsequently dug out and sun-dried, then broken up and mixed with the sand charges, or they may be conserved in dams for future consideration, unless the value is low, when they are frequently run to waste. To separately treat the insignificant quantities of current slimes produced daily by the small mills is, in the majority of cases, out of the question; to conserve them—when they are worth it—until a sufficient tonnage has been accumulated to justify the erection of a small plant is becoming the usual practice. As to what particular style of plant will eventually predominate, one can but surmise. None of the three well established processes—the decantation, filter press or vacuum filter—methods are likely to find ready clients amongst the small workers, the initial cost of installation alone will "give them furiously to think." The Adair-Usher process has now stepped into the field and may prove a formidable competitor.

In conclusion, I am anxious to say I feel an apology is necessary in that I have brought before you no new thing. One of our Past Presidents, Mr. Butters, once reminded me, some years ago, that we were engaged in a business undertaking and not conducting a series of chemical experiments, and so our first consideration should be to make a profit. This is particularly the case in Rhodesia. The small man has had much to contend with, and can scarcely be expected to take kindly to a second form of perilous pioneering.

The President: We have listened with great pleasure to these reminders to many of us of the early Rand days, and I am sure the paper appeals to all old workers on these fields. There are one or two things I would like to say concerning these. First, I would like to take exception to the words "the curse of ennui attending the monotony of perfection," and applying it to Rand cyanide managers. My experience of them is that ennui rarely visits them. Again, I think Rhodesia might well benefit by more expert advice than they have had in the past, since the more expert advice they have in connection with small plants, the more these plants will benefit. Next, with regard to slimes plants I would beg to differ from Mr. Thomas, for I consider that even on a small mine a slimes plant is essential. The advantage of treating slimes fresh is so great that one can only express the hope that slimes plants will become more general in Rhodesia. The place for the individual as compared with large companies is a description which has been applied to Rhodesia and even with regard to small slimes plants Rhodesian individuals will benefit thereby. The paper also gives many of us an indication of what we may expect if we go and try our luck on Rhodesian gold mines, and I am sure it will in this way benefit many of us connected with this industry. One part that appeals to me specially is the remembrance called to my mind by the use of these calcining trays for the furnaces. One of the first plants I was engaged on in this country was a cyanide plant where they used these calcining trays. One Sunday the cyanide manager, who was smelting, took it upon himself to enter the assay office and take out one of the borax casks. He put it back without saying a word to me, and my screens then went up 30 or 40 dwt. It was a worrying time for myself and the manager before I found out where the trouble was.

In connection with the next item on the agenda paper, I may say that some ten days ago, Mr. Cullen told me through the 'phone that we were going to have a Modderfontein night. I assured him that we would welcome such a night, but I failed to understand that it included the presence of ladies.

SPECIFIC GRAVITY OF BISULPHATE SOLUTIONS.

By Thos. Donaldson (Member).

I need hardly mention in bringing this short paper before you that sodium bisulphate is now used quite extensively on the Transvaal gold fields, for the purpose of dissolving gold-bearing zinc during "clean up."

As far as the Transvaal is concerned, the only source of this chemical in the meantime, is the Dynamite Factory at Modderfontein, and as one interested in its production, I may say that more than once enquiries have been made at the Factory as to the most convenient and economical way of using bisulphate for the purpose referred to.

In June, 1905, a very valuable paper on this subject was read before your Society by Messrs. Thomas and Williams, and later, in November of the same year, in reply to discussion on their paper, these two gentlemen gave a description of their method of using bisulphate on the large scale. I have recently seen this method in practice and I must admit the difficulty of suggesting any improvement on it.

Messrs. Thomas and Williams first make a stock solution by dissolving the salt in warm water, in a tank specially designed for the purpose. This solution is brought to something approaching saturation point. Proportions of the solutions are then run by gravity into the zinc dissolving vats, and in these vats it is diluted to a convenient working strength by addition of water. At this point in the process I recommend the more general use of hydrometers, graduated to specific gravity direct, or to degrees Twaddell, because even although the stock solution has been made up very roughly, the working solution can easily be adjusted to the desired strength by means of these instruments.

After adding, say, an equal volume of water to the strong solution in the vat, and stirring a little by means of the mechanical arrangement provided, a sample of the diluted solution can be drawn through the manhole and its specific gravity determined by use of a hydrometer.

Reference to the table appended to this paper will show the equivalent of H_2SO_4 contained by the solution.

In the event of the solution being appreciably hot it will be advisable to take advantage of the correction for temperature, provided on the table.

Having determined the strength of the solution, more water, or more stock solution, can then be added as desired. On mines where no special plant has been erected for the use of bisulphate, but where the solution has to be made up in the zinc-dissolving vat, a hydrometer will be found a most useful guide in regulating

Table for Bisulphate of Soda Solutions.

Grammes Bisulphate per 100 c.c. Solution.	Specific Gravity at 15° C.	Twaddell.	H_2SO_4 per 100 c.c. Solution.	Equivalent of H_2SO_4 in Zn.
		Deg.	Gm.	Gm.
20	1·145	29·0	6·4	4·27
21	1·152	30·4	6·72	4·48
22	1·159	31·8	7·04	4·69
23	1·165	33·0	7·36	4·91
24	1·172	34·4	7·68	5·12
25	1·178	35·6	8·00	5·33
26	1·185	37·0	8·32	5·55
27	1·192	38·4	8·64	5·76
28	1·198	39·6	8·96	5·97
29	1·204	40·8	9·28	6·19
30	1·211	42·2	9·60	6·40
31	1·218	43·6	9·92	6·61
32	1·225	45·0	10·24	6·82
33	1·232	46·4	10·56	7·04
34	1·239	47·8	10·88	7·25
35	1·246	49·2	11·20	7·47
36	1·253	50·6	11·52	7·68
37	1·259	51·8	11·84	7·90
38	1·265	53·0	12·16	8·11
39	1·271	54·2	12·48	8·32
40	1·277	55·4	12·80	8·53

Allowance for Temperature.—For every 5° C. above 15° add ·0028 to the specific gravity reading.

The table is made out for bisulphate of 32·06% available H_2SO_4.

the strength of the working solution. Messrs. Thomas and Williams recommend the use of a working solution containing about 9.25 gr. H_2SO_4 per 100 cc., and they show that at this strength the zinc dissolves rapidly, but that there is no danger of crystallisation, or of the action becoming too violent. I may be allowed to add that the following table of specific gravities agrees in the main with the one already given by Messrs. Thomas and Williams.

In my table, degrees Twaddell are also shown for the reason that the use of a Twaddell hydrometer tends to greater accuracy than the direct specific gravity type, on account of the small value per degree of the former.

In conclusion, I desire to thank my colleague, Mr. Ayers, for his kind assistance in connection with the preparation of the preceding table.

The President: I am sure we are indebted to Mr. Donaldson for bringing these few particulars before us. The table he has produced will be of interest to any one who takes up this method of testing the strength of the acid.

Mr. A. F. Crosse (*Past-President*): I deprecate the criticism of any paper on the evening on which it is read, more especially when read in a hall such as this where one cannot hear very distinctly. I cannot help thinking that we are mistaken in dissolving the zinc as a sulphate and throwing away the product. Some day we shall wake up to the fact that 90% of the zinc remaining in the zinc box can easily be recovered as metallic zinc. I am sure that some member of this Society can bring out a better method than the present one of dissolving the zinc in sulphuric acid or bisulphate of soda.

The President: I am sure we agree with what Mr. Crosse says, and I do not think we can do better than fall back upon Mr. Crosse for bringing such a method to a successful issue.

Mr. G. F. Ayers (*Associate*) read the following paper :—

A METHOD FOR THE RECOVERY OF ZINC FROM SOLUTIONS OF SULPHATE.

By Wm. Cullen (Past-President) and G. F. Ayers (Associate).

Although several attempts have been made to recover the zinc used for the precipitation of gold, and transformed in this operation from the metallic state into that of a solution by the solvent action of sulphuric acid or soda bisulphate, none of them, either for economic or practical reasons have attained anything more than an academic value.

One of the important factors in all industries is economy, whatever branch it may affect, and it is with reference to the recovery and saving of this zinc, which for many years has been run to waste, that we have turned our attention.

The alkali hydrates and carbonates of sodium and potassium, although fulfilling the purpose of precipitating zinc as hydrate or carbonate, proved too expensive to justify their use for recovering the zinc economically. Burned lime was experimented with, and although it proved to be a thorough, successful, and cheap precipitant, the formation of $CaSO_4$ rendered it unsuitable. The free sulphuric, as well as the sulphate combined in the form of $ZnSO_4$, produced with the lime a precipitate which adulterated the $Zn(OH)_2$ to such an extent that the final product only contained about 40% $Zn(OH)_2$. The further difficulty of removing the $CaSO_4$, or separating the $Zn(OH)_2$ decided its non-utility.

For practical and economic purposes we found magnesite the starting base for the recovery of the zinc. The chemical reaction is one of the simplest :—

$$ZnSO_4 + Mg(OH)_2 = Zn(OH)_2 + MgSO_4.$$

It is the application of this simple equation, however, upon which its success depends, and upon which much time and experiment have been expended. When magnesite is calcined, MgO and CO_2 are produced. Hitherto the gas has been the chief object of the operation, the magnesia being a bye-product. If the calcination is conducted at a white heat "*dead-burned*" magnesia is formed, but at a red heat (800-900° C.) caustic magnesia is produced. (See *Journal of the Society of Chemical Industry*, "The Indian Magnesite Industry," H. H. Dains, No. 10., p. 503, 1909.) It is with the latter we are interested, as it combines with water, forming $Mg(OH)_2$, which is the successful precipitant of zinc. The solubility of $Mg(OH)_2$ in water seems to vary considerably according to the various sources of reference. Calculated from the electrical conductivity of $Mg(OH)_2 + Aq$, 1 litre of H_2O dissolves 9 mg. of $Mg(OH)_2$ at 18° C. (Kohlrausch and Rose). For all practical purposes it may be regarded as insoluble in H_2O.

When zinc slimes are treated with either sulphuric acid or bisulphate of soda, the zinc goes into solution as sulphate.

According to the Government Mining Engineer's returns for the year ending June, 1908, the total value of zinc and zinc discs employed by the mines was £107,500, all of which was lost.

By the process dealt with in this paper, it is only possible, however, to recover portion of it, *i.e.*, that portion referred to as zinc slimes. Unfortunately there are no statistics of the absolute quantities of these slimes handled, but fortunately the equivalent of Zn in sulphuric acid and soda bisulphate gives one a fair estimation, *e.g.*, 98·08 H_2SO_4 : 65·4 zinc. The approximate amount of 100% H_2SO_4 from these sources is 220 tons per month, and assuming an excess of 5% H_2SO_4 used in practice, this figure becomes 210 tons or its equivalent 140 tons of recoverable zinc, having an approximate value of £3,000.

The process of recovery is simple, and consists in brief of adding to the acid solution of zinc sulphate an extremely fine emulsion of magnesium hydrate. An interchange of acid radicles takes place, insoluble zincic hydrate and soluble magnesic sulphate being formed. As the calcined magnesite always contains a certain amount of impurity, its precipitating value depends upon the percentage of caustic magnesia present, the precipitate of zincic hydrate being adulterated by the amount of gangue, uncalcined magnesite, etc.

The *modus operandi* is as follows:—The quantity of free acid and zinc in the solution to be treated is determined. An equivalent amount of available caustic magnesia, to neutralise the acid and precipitate the zinc is added in the form of a fine emulsion of magnesic hydrate and water, the whole being mechanically stirred. Heat aids and quickens the precipitation. The precipitate is then isolated, either by the decantation and washing process, or by filter-presses.

Our experiments have been conducted with a large sample of local magnesite, of which the following may be taken as a typical analysis:—

	Per cent.
MgO	82·61
$MgCO_3$	3·82
SiO_2	10·75
$Fe_2O_3 + Al_2O_3$ ...	0·42
H_2O	1·90
	99·50

The methods used for the determination of the free acid and zinc, were based more upon rapidity, convenience, and practical utility, than scientific accuracy. The probable error is about $\frac{1}{2}$% on the total quantity estimated. For the estimation of free acidity a measured quantity of the solution containing the zinc was titrated with $\frac{1}{5}$N, KOH solution, the zinc present acting as indicator, by its instantaneous precipitation as hydrate, the moment the alkali was in excess. The solubility of freshly precipitated $Zn(OH)_2$ in dilute K_2SO_4 solution is ·00187 gm. in 50 c.c. (Snyders B. 11. 936), the amount present at the end of titration, a quantity far too small to effect the result. The end point is very decisive. The zinc was determined in the original solution volumetrically, using standard sodium sulphide with ammoniacal solutions and ferric chloride as indicator. In estimating the zinc unprecipitated after the treatment with magnesic hydrate, ammonic chloride has to be added to keep the magnesium hydrate (formed in rendering the solution ammoniacal) in solution, the same volumetric method being then employed.

Standard Na_2S solution 1 c.c. Na_2S indicates 0·02 gm. Zn. Ferric chloride indicator 5 c.c. of 0·3% solution of $FeCl_3$.

Experimentally the following process was adopted:—To 100 c.c. of warm zinc solution (containing from 2 – 8% zinc and 0·15 to 3% H_2SO_4) a weight of calcined magnesite containing an equivalent neutralising and precipitating amount of caustic magnesia (made up into a fine emulsion with pestle and mortar), was added during mechanical stirring. The stirring and warming were continued for 10 hours, the whole allowed to settle till clear; the clear solution poured off, the precipitate washed and the washings added to the solution. Hydrochloric acid and then ammonia added in excess, cooled, and titrated for the zinc which might not have been precipitated by the magnesia. With freshly calcined magnesite of 82·61% magnesia purity, 99·5% of the theoretical precipitation of zinc was obtained.

The most essential and important points of the process are:—

(1) The calcination of the magnesite.

(2) The thorough disintegration and emulsification of the calcined magnesite with water. (Regarding the latter, perhaps nothing would be more suitable than a tube mill).

Dealing with the process from a commercial standpoint, the following data present themselves:—One ton of commercial calcined magnesite of 80% purity and costing £7 per ton, will precipitate 1·3 tons of metallic zinc (value over £28) in the form of 2 tons of zincic hydrate and 0·25 tons of impurity. (The price of £7 per ton for calcined magnesite is the maximum figure, f.o.r. Johannesburg.)

This resultant precipitate might be shipped home for the recovery of the metal, or form the basis of a white paint industry, for which there is a large demand. Alternatively the zinc might be recovered locally by any of the well known processes.

Seeing that we have a simple means for the recovery of zinc, it might be interesting from a metallurgical point of view, to find out, if it would be advantageous to employ more zinc for the precipitation of the gold than is now considered necessary. In other words if the precipitation of gold could be improved by using more zinc the extravagance might prove an economy. On this point we do not speak as experts, but only throw out the suggestion for what it is worth.

Mr. C. B. Saner (*Member of Council*): I have great pleasure in proposing a vote of thanks to Mr. Cullen and Mr Ayers for this short paper. It is a pleasing factor that our Past-Presidents after they have passed the chair do not consider they have done all the work they need do, but still gird up their loins for further effort. I think it behoves the lesser lights of the Society to emulate them. They do not weary us, they interest us. Following on Mr. Crosse's lines I will not criticise the paper to-night.

Prof. J. A. Wilkinson (*Member of Council*): I beg to second this vote of thanks with pleasure, but, it would be unfair to criticise the paper at a moment's notice. One thought, however, occurred to me whilst the paper was being read, namely, whether the authors had carried out any experiments to recover the zinc electrolytically. As we are shortly to have a large supply of electrical power this might prove an economical method of extraction.

Prof. G. H. Stanley (*Member of Council*): I should like to support this vote of thanks to the authors for calling attention, in an interesting paper, to a source of waste in the gold industry for which it would seem that a remedy can be found.

I had occasion quite a while ago to look into the same thing, and came to the conclusion that ordinary chemical precipitation processes did not present, from the commercial point of view, much prospect of success, for reasons which I propose now to apply to the process under consideration.

In the first place the precipitate, after drying, will contain something like 58% zinc, and such a material would be worth only £8 to £9 a ton at the smelter oversea. Against this would have to be set the cost of magnesite (1 ton for 2·25 tons precipitate), say, £3 2s., freight, bagging, and other realisation expenses, which I do not think would be much under £2 10s. and more probably over this figure, cost of labour, etc., in precipitation, filtering and drying, very problematical, but in view of the comparatively small amount to be treated on any individual mine, likely to be fairly high, but, say, for the sake of illustration, £1 per ton or total of £6 12s., which leaves such a small margin of profit that it seems scarcely worth while. Further, I very much question whether the estimate of the amount of zinc recoverable is not rather wide of the correct figure since there are other bodies which use up sulphuric acid in the zinc precipitate. It is certainly very difficult to arrive at a close approximation of the actual quantity of zinc dissolved by the acid, but by the author's figures there are 140 tons, or 4,083,000 oz. of zinc accompanying about 600,000 oz. gold in the precipitate treated monthly, or in other words, the precipitate would contain 12·8% gold if it consisted entirely of zinc and gold. It does not, however, and it is stated by T. K. Rose that from 20% to 40% of other matter may be present which, of course, has the effect of still lowering the gold percentage to, say, 10% or under in the assumed case. We know that, for financial reasons generally, there is often a lot of zinc dissolved which should not be, but I question whether the average of the precipitate treated is as low in gold as 10%, or if it is, whether this is not due to other matter than zinc in high proportion. Still, perhaps the cyanide managers, who should have a lot to say in this discussion, will tell us what the average richness is, or what is more important in this connection the ratio between the zinc and gold present. Obviously, though, if the precipitate is richer than 10% the recoverable zinc is diminished accordingly.

Turning to the possibility of recovering the metal locally: this promises better economic results, although it must be remembered that not more than 85% could be recovered by any of the well-known distillation processes. Smelting costs would probably be not much less than £3 per ton of precipitate, but freight charges oversea would be saved each way; still it would obviously be impossible to do this except at one central works, and it is still questionable whether any but a few of the largest producers would consider it worth while to precipitate their zinc-bearing waste solution, since over 70 producing mines the average zinc recoverable is only 2 tons per mine, taking the author's figure.

I have no information as to the suitability or value of the precipitate for paint manufacture, but at first sight this appears to present a more likely opening for the product if the market could absorb such a quantity.

It seemed to me though that a method which would recover the zinc in metallic form *on the mine* has much more chance of being adopted and with this idea I made a number of experiments in 1907 on the electrolytic precipitation of zinc. I started with the idea of depositing it in the form of a coherent sheet which could be easily melted without undue loss and cast into slabs for rolling, but soon found it a very difficult matter (as is, of course, well known) to arrange the necessary conditions and so would expect this to give too much trouble in a works. It was quite easy, however, to deposit it in branching crystalline forms and also in spongy masses, which I think could be washed and used direct in the zinc boxes (possibly after a dip in lead metals if found to be too pure to act very effectually), thus saving the expense of melting, casting, rolling and cutting into shavings. I still have by me some specimens illustrating the form in which the zinc deposits, which I will show at the next meeting. At the time I mentioned the matter to one or two cyanide managers, but their opinions were not encouraging, chiefly in view of the small quantity to be recovered, and as other matters claimed my attention I dropped the experiments. I still think, however, that the electrolytic method has possibilities, particularly in view of the amalgamations now effected and to come, and because of the probable cheapening of electrical energy in the near future, and with such a concentration of output as may be expected from these amalgamations the corresponding concentration of zinc waste at any rate justifies further experimental work towards its recovery, particularly with a view to determine the best conditions, current efficiency, etc, and, above all, probable recovery and cost.

Mr. D. W. Greig (*Associate*) read the following paper :—

ANALYSES OF GASES FROM BURNING NITRO-GLYCERINE EXPLOSIVES.

By W. CULLEN (Past-President) and D. W. GREIG (Associate).

It will be remembered that in the paper by one of us, which appears in the November number of last year's *Journal*, a large number of gas analyses were given. These were all results obtained by detonation. In no case were oxides of nitrogen observed. At that time we had not investigated the behaviour of any of the ordinary or even the special explosives on burning, but we were hopeful that the new explosive which we had provisionally designated "Antifume," might prove an improvement on the old. It is surprising how very little work has been done in this subject. Here and there one finds figures given, but they nearly all emanate from the one source. Of course, everyone out here knows in general terms what happens, but no definite figures have ever been advanced excepting those relating to mortality. In other directions a certain amount of pioneer work has been done, but the conditions have nearly always been artificial. What one desires to know out here is how explosives behave on burning in the open, or at atmospheric pressure. Most of the experiments referred to have been carried out in steel bombs in which the conditions are almost exactly the reverse of those which generally obtain when burning takes place at atmospheric pressure. In a bomb there is a rapid acceleration of pressure culminating in a maximum which can be determined, but Sir Andrew Noble proved many years ago that quite apart from the question of sufficiency, or insufficiency of oxygen, pressure has a most profound influence on the gaseous products of explosion.

Sarrau and Vielle, whose work is quoted by Berthelot, have put several analyses on record. In one they show that out of every 100 volumes of the gaseous products evolved on burning nitro-glycerine 48·2% is nitric oxide—an appalling figure. For nitro cellulose the corresponding figure was 24·7%. Taking these figures as a basis, Dr. Weiskopf, in giving evidence before the Miners' Phthisis Commission, said that the amount of nitric oxide which would be produced by the burning of one cartridge (250 gr.) of Modderfontein blasting gelatine was 84·81 litres, or 2·99 cub. ft. = 41·65 vol. per cent.

After these preliminary remarks we shall pass on to our own work, the exact details of which are given in the appendix. It is to be observed that our figures are calculated on a somewhat different basis from the usual; they give the absolute quantity of nitric oxide produced by unit weight, on burning. That is, according to table I, every 100 parts of ordinary blasting gelatine has produced 33·46% of its weight of nitric oxide.

TABLE I.—Absolute quantities nitric oxide found in the gases resulting from the burning of the following nitro-glycerine explosives :—

		p.c.
I.	Ordinary blasting gelatine ..	33·46
II.	Antifume blasting gelatine ...	29·23
III.	Blasting gelatine containing chlorate	33·27
IV.	Gelignite ordinary	21·00
V.	Gelignite modified	19·58
VI.	Gelatine dynamite	25·00

TABLE II.—Volumes of nitric oxide produced by the burning of one $1\frac{1}{4}$ in. cartridge (about 250 gr.) of the following nitro-glycerine explosives :—

		Litres.		cub. ft.
I.	Ordinary blasting gelatine	80·68	=	2·845
II.	Antifume blasting gelatine	70·42		2·487
III.	Blasting gelatine containing chlorate... ...	80·15		2·830
IV.	Gelignite ordinary ...	50·59		1·787
V.	Gelignite modified ...	47·17		1·666
VI.	Gelatine dynamite ...	60·23		2·127

These figures call for no particular comment. Those of Table II. are just the same as Table I., but put another way. One thing is perfectly clear, and it is that Antifume does not behave very differently from ordinary blasting gelatine. True, the amount of nitric oxide produced is less by about 12%, but relatively speaking this is a small reduction. From the results it would appear that there is a direct ratio between the percentage of nitro-glycerine present and the amount of nitric oxide produced.

What the experiments have made us realise is that with all nitro-glycerine explosives tremendous quantities of nitric oxide are produced on burning. The inference is so very obvious that one need hardly dilate upon it, but some inventors have actually gone the length of endeavouring to develop the potential power of nitro-glycerine explosives by burning to start with than which no more efficient method of killing those underground could well be devised. One tiny $\frac{7}{8}$ in. cartridge of blasting gelatine can produce enough nitric oxide to kill many people. Experts who have made a special study of the toxic effects of this gas will be able to translate our figures, but as in the case of carbon-monoxide —only much more so—they can only be described as astounding.

Appendix.

Description of Method Employed.—The explosive was burnt and the gases absorbed by potash solution, the idea being to titrate the excess of potash left after absorption of the gaseous products, with acid. The difficulty experienced was to find an indicator which while showing up alkaline to sodium carbonate of which there was bound to be considerable quantities, would behave as an acid with, and would not be decolorised by, nitrous acid.

It was found that a combination of potassium permanganate and alizarine answered the purpose. A weighed quantity of the explosive (about 0·8 of a gramme) was suspended on a piece of platinum wire, connected with a battery, in a bottle of 14 litres capacity, into which had previously been introduced a known solution of potash (50 c c. $\frac{N}{5}$ answering the purpose). The explosive was burnt by raising the platinum wire on which it was suspended to a red heat. After burning, the bottle was allowed to remain unopened for thirty minutes to make certain that all oxides of nitrogen had been absorbed by the potash, and the solution then washed into a porcelain basin. The quantity of water used in washing out the bottle was about 700 c.c., the total volume of solution in the basin amounting therefore to about 800 c.c. This solution (after making sure that it was alkaline) was then raised to 90° C., and when everything was ready for the titration a drop or two of dilute $KMnO_4$ solution added. The delicate pink colour of the permanganate was at this temperature, almost immediately changed to a light yellowish brown colour, due presumably to the precipitation of MnO_2 in a fine state of division. Standard sulphuric acid was run into this until the colour just disappeared. To the faintly acid, decolourised solution, alizarine was added, and while it was still hot, $\frac{N}{5}$ potash solution introduced drop by drop until the characteristic end point of alizarine was reached. Five or six drops of potash was found to be all that was necessary, when the first decolorisation had been carefully executed. The first indicator ($KMnO_4$) was used in order to obtain the final solution just on the acid side of the neutral point; as to have had this solution decidedly acid at 90° C. would have resulted in the loss of nitrous acid.

Dr. D. Macaulay, M.L.A. (*Member*): I came here to hear especially what Mr. Cullen and Mr. Greig had to say on the results of their experiments with regard to the improvements they have been trying to introduce in the matter of the nitro-glycerine explosive, which are so largely used on these gold fields. I think we have to congratulate ourselves, and to offer them our congratulations for the honesty with which they have produced their figures. This is a very serious subject, and I am more and more impressed with it every day. Attending here for the first time for many months I wish to confess that I do think the discussion in which we were engaged several years ago has been productive of good results, and as one of the opponents of this theory I must admit that I think the gases produced in the detonation or combustion of nitro-glycerine explosives have a great deal to do with the production of disease among mine workers. The authors have proved to us, although unfortunately their results are negative, that there are men amongst us experimenting and working hard to obviate the danger of these explosives. I have no doubt that if the members of the Society continue in the steadfast course they have set before themselves, it may be possible to arrive at the happy position when an explosive may be invented which will not have deleterious effects. The sum total of this paper is that their new explosive is not very much better than the old. We all know, that in actual practice in the mines for reasons which it would be well for the Society to investigate, the results are very deleterious indeed, and that the nitro-glycerine explosives used on these fields are not theoretically perfect. The practical result of Messrs. Cullen and Greig's experiments is, that although the behaviour of this new explosive is such that it enables miners to return to the face sooner, yet it produces an amount of carbon monoxide on explosion which would probably poison every one in this room. The pleasing feature to me is the honesty of the paper. I think we are called upon to propose a hearty vote of thanks to Mr. Cullen and his coadjutor, and to wish them good luck and success in the future.

Prof. J. A. Wilkinson (*Member of Council*): I desire on your behalf to second the vote of thanks proposed to Messrs. Cullen & Greig, and again to emphasise the honesty of purpose shown in bringing these results before the Society. I quite agree with the authors in their statement that the results obtained are astounding. All of which means that further research is necessary before we can arrive at a satisfactory method of destroying the effects produced by these gases, or, better still, of preventing their production. Those of you who have followed the technical papers during the last few months will have noticed descriptions of what is styled the United States Government Explosives Testing Station. Briefly, it consists of galleries where all kinds of tests, both chemical and physical, are carried out with explosives. On the various mines here a much larger amount of explosives is used than anywhere else in the same area in the world, and the establishment of a similar station here—or better, if possible—is a subject which might well merit the attention of the Mines Department, since the results obtained by its means would amply repay its institution. Another feature of such a station is that it enables experiments to be carried on in the rescue of men by the use of safety helmets. By reference to last month's *Journal* I see that Mr. Cullen drew attention to the fact that he had transgressed one of the bye-laws of the Society, and stated in response to a request by the President that he would make good the omission at this meeting. I therefore hope that he will now come forward with the information promised.

Mr W. Cullen (*Past-President*): It is quite true I made that promise at our last meeting, and I hope to supplement the promise before I sit down. With regard to the short paper we have submitted, the results are a very great disappointment indeed. We anticipated that the modified explosive would have been a great advance in the matter of the production of nitric oxide, but there is no getting away from the fact that they are no advance whatever. The results indicate that the amount of nitric oxide produced bears a ratio to the percentage of nitro-glycerine contained in the explosives. With regard to the future I am not at all hopeful that we shall find it possible to evolve an explosive which will behave satisfactorily in regard to the production of nitric oxide on burning. Of course two or three years ago we could not have said that we should be successful in doing away with carbon monoxide, and this gives us zest to carry out our experiments still further. Mr. Greig, in reading the paper, referred to the appendix, which deals exclusively with the carrying out of the experiments and the method of analysing the gases. This latter is perhaps open to criticism, but it gives approximate results, which is as much as one can expect. We wished to bring these results before the Society as soon as possible in order to avoid all possibility of misadventure, because many people even to-day do not realise that there is a difference between detonation and burning. With regard to the composition of Antifume Blasting Gelatine, I would refer members to Transvaal Specification No. 466 of 1908, where full particulars are given.

Dr. J. Moir (*Vice-President*): I hope to be able to discuss the scientific part of Mr. Cullen's paper at the next meeting, for I did a little work on the same subject a few years ago, but I want now to take advantage of the reading of this paper to make a remark on a subject connected with it. I want to sound a note of warning concerning certain newspaper paragraphs which have recently appeared, dealing with the "Antifume" explosive of the Modderfontein Factory. These paragraphs, which I am assured do not emanate from the factory, state that when this explosive is used, it is safe to return immediately to the face blasted. This is I consider very dangerous matter to appear in the public press, for if miners are to trust in it there will be an alarming increase in gassing accidents. My attention has been called today to the death of a native from gassing by the fumes of Antifume gelatine, as a result of this practice. It is true that an extra quantity of fumes was present as a result of the first "cut" having failed to come away, but the fact remains that there is still 0·2 or 0·3% of CO in these fumes as produced, and although they are very much less dangerous than the fumes of ordinary gelatine (which contain 1% of CO) they are still dangerous to life if inhaled for more than a minute or two.

Mr. W. Cullen (*Past-President*): I would just like to endorse the words which have fallen from Dr. Moir's lips. It is a great pity that an erroneous impression has been conveyed by certain newspaper paragraphs which are altogether unauthorised. Although the carbon monoxide has been greatly reduced there is still quite enough present immediately after explosion to be a source of danger.

SOUTH AFRICAN COALS AND THEIR ECONOMICS.

(*Read at April Meeting, 1909.*)

By Arthur J. Andrews.

DISCUSSION.

Mr. Tom Johnson (*Member*): Mr. Andrews, in his paper, speaks of the collection of smalls and rubbish in the bottom of the bunkers. From this cause engineers get many anxious hours when consignments are late. This trouble I have seen lessened very much by making a practice of working from one end of the bunkers to the other clearing the bottom regularly, thereby keeping down the accumulations and using the coal before it had time to become seriously deteriorated, and I would recommend the practice to our engineering friends. Many engineers keep their bunkers full because of uncertainty of arrival of consignments, as it is a very serious thing for a mill to be hung up for want of steam. Those who wished to keep a stock of coal on hand for time of emergency could keep it under water; in this manner the deterioration is very small, and it would not be a great expense to make and maintain a small dam to carry 100 or 200 tons of coal. As Mr. Andrews and Mr. McArthur Johnston point out, waste is caused by negligence in firing and by using poor boys. Other reasons are, having fire doors open too long, not using the damper, also allowing too much encrustation on tubes, etc., leaks in boiler seatings, leaky steam pipes, etc.—all these are coal wasters.

I have always thought that it would pay to have a separate engineer on each large mine to look after the generation and use of steam itself, *i.e.*, coal, boilers and seatings, steam lines, valves, taking cards, etc. On many of our mines an engineer could save as much as his salary by going round having leaks stopped, steam pipes properly covered, water from exhaust steam conserved, etc.

Mr. Andrews says that there seems to be, in many cases, a want of interest shown in the different coals in use. This is not the fault altogether of the engineer, who may regulate his system of firing for a particular coal, also his furnace and bridge, then some one else orders a different coal, requiring a larger or smaller furnace and a different system of firing; then away goes all heart in the work, for fresh orders have to be given and the boys trained to a new system, and probably before the engineer can get them going well in the new way, he gets the old coal again and he reverts to the old system, and the boys will say to one another that the boss must be a *malkop*.

Our President, in his contribution, mentioned that in the United States Government tests of coals, good hand firing increased the efficiency of the boilers by 10% to 15% under ordinary commercial conditions; that is so. They also found that if the gases were passed through a boiler with twice the usual speed, practically twice the power could be got from the boiler, proving what Prof. Perry has said for a long time. The reasoning, to my mind, is easy to follow, seeing that the conductivity of the metal of the flue of a boiler is so great, that there is very little difference of temperature on the two surfaces of the flue, so that the heat given to the film of gas on the one side, will be transmitted to the film of water on the other side, and if twice the mass of gases of a given temperature are passed through a flue in a given time twice the amount of heat will be passed through the metal of the flue, providing the circulation of water is good enough. The circulation of water in a boiler does not increase as fast as the capacity of the boiler, so that to get high capacity and efficiency from a boiler or boilers, adequate circulation must be provided to carry off the bubbles of steam as they are formed. This is important, as the circulation may be good enough in a boiler where the amount of coal burnt is, say, 20 lbs. per hour per square foot of grate, but quite inadequate if 40 lbs. per square foot were being burnt. In a case of this kind twice the capacity would not be obtained for twice the coal, but a high increase of stack temperature and waste would result. Prof. Perry, in his book on "The Steam Engine," gives instances of boilers being tested with half the tubes plugged up, the same work being got from the boiler by increasing the draught, and passing the same weight of gases through the remaining half of the tubes. If the initial draught in these cases equalled 0·5 in. water, what would the draught have to be when the half of the tubes were plugged. As the area of rubbing surface was reduced to one-half and the friction in the open half of the tubes increased to four times, the draught necessary would be twice or 1·0 in. of water. Really it would be something less, as the friction other than in the tubes would remain constant. But if it was necessary to get twice the power from a boiler or boilers, leaving all the flues open, then it would need more than four times the draught as measured by the water gauge, for the friction in the stack would also be increased, due to the larger amount of gases; this friction the water gauge does not measure. The stack friction could be kept down by increasing the diameter of stack. It is this question of draught that will hamper our engineer, I fancy, that is, not having it. It is of primary importance, especially when coal is bad.

The different mines should be served continually with approximately the same kind of coal—not necessarily from the same mine—the furnaces and combustion chambers arranged for that coal, care being taken to fix good mixing arrangements to break up the gases (a long strip of metal twisted into a spiral, and placed in fire tubes of boilers of the F. & C. kind will cause the gases to give up more of their heat) bunkers should be covered, the bottoms should be so formed that the smalls cannot accumulate; in flat-bottomed bunkers the practice of clearing regularly from end to end should be followed, boys taught to fire as quickly as possible once the doors are opened; also they should be taught the use of the damper while firing; all air leaks in the boiler settings should be stopped, steam pipes covered, and steam leaks at joints and valves stopped.

The quantity of air used, or, more correctly, the way in which the air is used, has a bearing on the economical use of coal. Large excesses of air within reason—say between 25% and 75% excess—may be used with advantage if it is really passed through the fire and not through holes in the fire.

It does not always pay to get too great a vacuum in the condensers; it is better to get the water back to the boiler as hot as possible. I have seen engineers striving to get perfection in their condensing arrangements, and wasting more power to get those last two inches than they gained. More use should be made of the exhaust steam from winding and other engines exhausting to the atmosphere. Boilers should not be run too long without cleaning, and good, *i.e.*, suitable water should be used. I have seen many pounds spent on mines for purifying apparatus, the boilers being worse after using the supposedly purified water than they were without. If engineers would collect the water from the exhaust steam, separate the oil from it, and use this water, they would have less trouble with their boilers, and save coal. Of course, this water should be returned as hot as possible.

In reference to mixing arrangements mentioned earlier, these all create friction, so that extra draught would be needed. At mines where this extra draught is not available and boiler power not sufficient, then extra draught should be provided before increasing the boiler plant. This applies to most mines, as everyone is wanting to handle greater tonnages, thereby needing more power. In conclusion, I would recommend to the "powers that be" the idea of using separate engineers, with, of course, the necessary training, for handling the generation of steam and the using of it. I am firmly of the opinion that it would pay handsomely. I would also recommend to our engineering friends and others the study of Prof. J. Perry's book, "The Steam Engine," particularly the chapter on how fluids give up heat and momentum; also, "A Study of Four Hundred Steaming Tests" made at the fuel testing plant, St. Louis, Mo., U.S.A. The latter can be obtained free from the Director, Department of Interior, U.S. Geological Survey, Washington. Some of the general conclusions derived from these tests are that while fairly high average efficiencies were got, better efficiencies could be obtained if furnaces and other conditions were more adapted to particular coals. High CO_2 contents in the gas analysis is not a definite indication of high efficiency. The best efficiencies of tests recorded in the paper were obtained with about 10% of CO_2. The rate of combustion is affected by the chemical composition of the coal, the size of the coal, the intensity of the draught, and to some extent by the thickness of the fire and the formation of clinker. Bituminous coals, high in "volatile matter," burn quicker than those high in "fixed carbon." Coals ranging in size from $\frac{1}{4}$ in. to $1\frac{1}{2}$ in. burn much more rapidly than either very small or very large sizes. This is undoubtedly due to better distribution of air with these sizes. With higher rates of combustion, less air is used per pound of combustible. Capacity varies almost directly with the rate of combustion. The presence of ash up to 15% in dry coal has very little effect on efficiency and capacity. Above this percentage, however, the efficiency drops. Moisture reduces efficiency and capacity perhaps, by hindering combustion. The presence of sulphur in coal is not detrimental to the value of coal as fuel for steaming purposes, except when it exists in certain combinations with other constituents of the ash so as to form fusible clinker which may adhere to the grate. Bituminous coal should be fired in small quantities every three or four minutes. It is best to fire only on one-half the grate area at a time. The best thickness of fire ranges from 5 in. to 10 in., varying with the intensity of draught 0·5 in. to 0·7 in. under the stack damper. For hand firing, a furnace with tile roofs and large combustion chamber containing good mixing structure is well adapted for burning bituminous and lignite coal. Inasmuch as the boiler plant is at present an expensive portion of the steaming plant, and as it probably is possible to cut its cost down to a fraction of the present cost, and at the same time obtain a higher efficiency, it is felt that there is good ground for anticipating a great improvement in the production of steam power during the next few years.

The author sees no reason why boilers cannot be constructed, and probably operated with entire satisfaction, which will produce about ten times the amount of steam now obtained per square foot of heating surface and with no difficulty in obtaining dry steam. It will probably be found on attempting this reduction of dimensions and cost that the limit will be not in the boiler itself, but in the combustion chamber. Burning a large amount of coal on a small grate area is largely a question of draught and continual riddance of ash, but the rate of travel of gases through a combustion chamber is dependent, practically, only on the amount of carbon burnt per unit of time, the rate being about the same, no matter what the air supply per pound of carbon.

Effort in completing of steam generating plants of small dimensions must be largely concerned with the construction of a combustion chamber containing many gas-mixing appliances.

REPLY TO DISCUSSION.

Mr. A. J. Andrews: I wish to thank Messrs. McArthur Johnston and Cullen for their support, and to express my appreciation of the valuable, additional information they have brought forward. I quite agree with the former that the subject is a very large one: I found it far too wide to do anything like justice to in a single paper.

Regarding the weathering of coal, raised by Mr. Cullen, it might be of interest to mention that certain samples of South African coal, thrown down upon the hard ground and exposed fully to weather conditions, lost from 2 to 5% of their calorific value in ten days. I understand, from his discussion, that Mr. Laschinger is inclined to regard the fossil tree casts at Vereeniging, as proof that South African coal seams generally were mainly made up of trees rather than twigs, leaves, bark, &c. For my part, I have always considered that such fossils, as those referred to by Mr. Laschinger, indicate an exceptional rather than a regular ingredient of the coal seams of South Africa. Mr. Tom Johnson has written, at considerable length, both as regards the direct and the indirect saving of fuel: in support, of which latter consideration, he has urged certain classical methods of steam economy, pure and simple; and, although most of these measures are well known, I think that their frequent neglect, in practice, may justify their importance being urged so earnestly.

In conclusion, I desire to thank the Society for having enabled me, as a visitor, to "open up" such an important subject.

ROUTINE ASSAYING ON A WESTRALIAN MINE.

(*Read at December Meeting, 1908.*)

By W. B. Blyth (Associate).

REPLY TO DISCUSSION.

Prof. J. A. Wilkinson (*Member of Council*) read the author's reply to discussion on his paper, as follows:—

Mr. W. B. Blyth (*Associate*): In bringing forward this paper for discussion the author was actuated by a desire to bring before members details of working methods prevailing in Westralian assay offices at the present time. The assayer here, as elsewhere, is expected to get through a large amount of work with a minimum of time and expenditure; and this can only be satisfactorily accomplished by evolving and practising a thoroughly systematic method of procedure. I had hoped to see Rand assayers rouse themselves sufficiently to attack and criticise the operation described; but as no one has accepted the challenge to any extent, I will reply very briefly to what little criticism I have been subjected to.

Mr. A. McArthur Johnston attacks my method of keeping checks of mine samples. Now my article described in detail the operation conducted in one office, and the method of keeping checks was quite applicable and reliable in the case mentioned, owing to the fineness of the Au and to its even distribution through the rock. In the presence of coarse Au it would naturally be necessary to reduce the rock to a considerably finer condition before one would expect two samples taken in the method described to agree. Like many other assayers, I should like to see all samples ground to pass through a 200 mesh screen before arriving at the office. Unfortunately economic considerations scarcely warrant the attainment of such idealistic condition; and one has perforce to be satisfied with a degree of comminution which experience shows will give a fairly satisfactory check under ordinary circumstances.

Mr. White, "as usual," waxes satirical at my advocating the use of borax as a flux, when I state that "fluxing without borax has been tried with satisfactory results." What I meant was that with the non-borax flux I got the Au out all right. If Mr. White had gone further he would have seen that I immediately modified my statement by saying that "the presence of a little borax tended to make the slag more fluid and to shorten the time of fusion." Now, I do not want to inflict any more of the borax versus non-borax controversy on members, so will content myself by observing that I am in complete accord with Mr. L. J. Wilmoth's remarks on the subject. Mr. Clennell also makes out a good case for the use of borax in moderation, and in view of the almost universal agreement among assayers on this subject, it seems to be about up to Mr. White to shelve his somewhat quixotical ideas.

Mr. Wilmoth questions whether I grind my samples fine enough. Now, I would like him to take some low grade sand samples, and pass representative portion through a 40 to 60 and a 100 mesh screen; to then assay same and to note the result. I will guarantee that with proper fusion he will get as much Au from the 60 product as he will get from the finer material. I went to considerable trouble to convince myself of this before reverting to the coarser screen.

Referring to the method described for assaying solutions, I am now using 200 c.c. for the assay instead of the 100 c.c. sample I mentioned before. The procedure is exactly the same, and the results equally accurate. With a 200 c.c. sample, ·001 gr. of Au obtained shows a value 9d. per short ton, and as the balance employed is quite sensitive to ·0003 gr., a difference of 3d. per ton can be noted. Greater accuracy than this seems only of academic interest.

Owing to a typographical error, I was quoted as using only ·5 gr. for a bullion assay sample, and Mr. Wilmoth is quite right in assuming that it should read ·5 gm.

SUGGESTIONS FOR A NEW ATOMIC THEORY.

(*Read at April Meeting, 1909.*)

By James Moir, M.A., D.Sc., F.R.S.S.Afr. (Vice-President).

Reply to Discussion.

Dr. J. Moir (*Vice-President*): The criticism on my paper divides itself at once and naturally into two classes; namely, (1) intelligent criticism, and (2) *a priori* criticism. The latter variety of criticism is entirely based on what I might be supposed to have said by critics who have not troubled to read what I actually did say; and this kind of criticism is, I regret to say, only too common in this pretentions but superficial community. Of actual criticism I have only one piece to deal with, viz., that of Mr. Watson; and I have to class together Messrs. Wilkinson, Roux, and Adair, as having either not read my paper at all or failed to grasp it.

Prof. Wilkinson did not fulfil his promise of reading my paper and discussing it, so I have nothing to say in that quarter. As for Egerton's paper to which he refers, I must say that I am not unduly impressed by it, being convinced that Egerton's "discovery" is partly due to mathematical necessity, and partly to sheer coincidence. I have, however, discussed this point elsewhere, and need not therefore repeat it here. I regret only that Prof. Wilkinson has been unable to point out any of the weak points in my theory—of the existence of which I am myself fully conscious.

I now turn to Mr. Adair. His first accusation is that I have not done a stroke of practical work in support of my theory: and that I compare very unfavourably with certain "chemical giants of last century." This accusation may be true; but in reduction of sentence for my offence, I have to say that I have so far had only a dozen years of working life, and yet have in that time been responsible for thirty or forty contributions to knowledge. Mr. Adair, on the other hand, claims forty years' experience, in which time he appears to have been too immersed in routine to accomplish anything more than what I must call a few mere "pot-boilers." It is true that Mr. Adair makes an excuse for his attack on me by saying that he wishes to save the younger members from being misled. To this I reply that such a theory as mine is very obviously not written for babes and sucklings.

With regard to the first half of his criticism, I have to point out that chemistry has long ago got past the classificatory stage. It is no mass of isolated facts; and what is certain is that any person who attempted to acquire a knowledge of its hundreds of thousands of compounds without the aid of theory, would very soon qualify for a madhouse. It is only by means of chemical theory that a chemist can predict (as he *can* do) all the essential properties of a substance which he has never seen or even of compounds still unknown; and in fact quite half of the current organic research consists in creating new compounds predicted by theory. I must give a point-blank denial to my critic's statement, that "even the greatest chemists and physicists nearly always fail in their predictions"—for somehow I cannot help thinking that Mr. Adair has been "confounding small things with great," and that he really had some Rand metallurgical process or other in his mind.

As for the rest of it, Humphrey Davy and the other pioneers invented molecular formulæ to account for such chemical transformations as they knew. I really do not see why I, a younger worker, and so far as possible an "heir of all the ages," may not be allowed to invent sub-molecular formulæ to account for the more intricate relations of the atoms which we now know. If Mr. Adair really holds the contrary opinion, I am compelled to say—harsh as it is—that it is a pity that he ever went in for Science. As things are, one can only contemplate with regret the waste of 40 years of his life in self-satisfied reactionariness—like the Major-General in "The Pirates of Penzance," which I bring up to date as follows to accommodate my victim:—

> "For his scientific knowledge, though he's plucky and adventury,
> Has only been brought down to the middle of last century."

However, at the end of a page of abuse, Mr. Adair gives us one tangible piece of criticism, that my structural picture of the sodium atom is not electropositive. This is due to a misunderstanding of the functions I ascribe to the element x, which is not a halogen and does not neutralise H (except perhaps when both H and x are attached at the same point). My formula for sodium is $Na = (Li - x) + (O + x)$, or what comes to the same thing $(Li - x) + (Ne - x)$; and this, although not quite satisfactory, is very far from being absurd. It should be noted that mere increase in size of the molecule would cause increase of electro-positiveness.

Mr. Roux offered no criticism of my paper, but instead gave us some of his own ideas. His conception of adding up the beginning and end of the list of elements is quite new, and is rather remarkable in the degree of agreement reached. I think, however, that this agreement is most probably due to coincidence, for, although Mr. Roux gets 17 out of the possible 28 pairs to add up to 239 (HU, LiTh, NRa, NaEm, PBi, PbS, TlCl, KHg, PtSc, MnW, NiTa, YSa, CeNp, PdCs, AgX, CdI and SnSb), yet I find that 14 pairs can be got adding up to 252, also 14 pairs adding up to 256, and 13 pairs adding up to 248 —all this indicating doubtless an obscure law in the original growth of the elements, but not supporting Mr. Roux's idea that the beginning and the end are complementary to each other. For example, I notice that the series osmium to bismuth is very nearly a repetition of the series glucinum to aluminium, thus Os = TaGl, Ir = TaC, Pt = TaN, Au = TaO, Hg = TaF, Tl = TaNa, Bi = TaAl; but I should nevertheless be loth to prophesy from this that elements of the formula TaB, TaNe, and TaMg must exist because they are missing from the above series. Another most remarkable coincidence I have noticed is that nearly all the multiples of 7 occur in the atomic weights when taken on the $H = 1$ basis.

Mr. Roux's periodic curve also contains some new presentations of the problem, notably the turning-points at the tetrad elements (see my paper, first page); but it will be noted that his curve is not symmetrical about the Sn-Sb point, which fact in itself negatives his 239 law. The curve is, however, nearly symmetrical about the point 150, so that 300 is the number which should give maximum agreement when the beginning and end are added together—instead of 239.

As for the rest of his paper, of course I reject without hesitation his conception of radio-active degradation of gold—there is plenty of experimental evidence against it; also his conception of "involution" (by which I imagine he means synthesis); and of "pre-established harmony" and other intangibilities too numerous to specify. Although I have myself been guilty of imagining structures to account for the properties of the atoms, I have no sympathy whatever with people who (like Miss Annie Besant) imagine hundreds of pages of structures of atoms, *without* knowing the properties of the atoms which are to be accounted for (see *The Theosophist*, 1908, in about ten monthly numbers).

To return to Mr. F. W. Watson's contribution, I have first of all to thank him for having taken the trouble, before discussing me, to understand the theory so well as he has done, and secondly, for presenting his views moderately and without prejudice—which is so refreshing.

As for the details, I have to say that I *did* mean nitrogen to be Cx and oxygen Cx_2—very nearly as methenyl is CH and methylene CH_2, except that x is not detachable at our present known temperatures, and probably always stays inside the molecule. I am not without hope that some day oxygen will be broken down into nitrogen; something of the sort is indicated by the similarity of the high temperature spectra of CN and CO.

I did intend urea to be isomeric with acetic acid, and benzyl alcohol with phenylhydrazine. By the way, I wonder why Mr. Watson chose these out-of-the-way samples to "show me up" with, because he could easily have found more familiar substances which I make isomeric. My theory makes CO isomeric with N_2, hence CH_4O is isomeric with N_2H_4, and the examples he has chosen follow at once from this. I see no real difficulty in such isomerisms, since we are familiar already with plenty of cases where isomerism depends entirely on arrangement and not on composition. Thus CH_2N_2 may be diazomethane or cyanamide—substances quite as far apart as acetic acid and urea. Again aldehyde is totally different from ethylene oxide, although both are C_2H_4O.

The argument from the non-existence of optical activity in phenylhydrazine is negative and not worth much on its merits, because all the known optically-active nitrogen compounds are unstable and tend by interchange of groups to become inactive; in any case, all the known processes for separating optical isomerides make the nitrogen pentad and (according to me) alter the configuration.

Again I have to agree with Mr. Watson that the O : H ratio is not 16 to 1, and my theory, which did not deal with the third place of decimals, is unaffected by taking $H = 1·008$ as usual. I have published elsewhere a discussion of the small fractions in the atomic weights, and I will only say here that I assume the existence of a very small element $= ·009$, the cause of valency, such that if oxygen contains two such and hydrogen one, the ratio O : H calculates out as 16 : 1·0078 as required.

As for the reason why He and Ne were chosen as the bricks in my edifices, Mr. Watson probably now knows the reason—namely—that they have actually been obtained as products of decay of radium and actinium and their emanations. This is experimental proof of the pre-existence of He and Ne in these higher elements. I may mention that I tried to transmute thallium into gold by means of uranium, but it did not succeed.

As for my element x, I naturally cannot define its properties exactly, but it appears to be identical with the α particle from radium (which is known to have $e/m = 5{,}000$, whence m is 2, and is either my x or H_2 or $\frac{He}{2}$). The nearest analogy to x is fluorine, which has enormous affinity and not much electro-negative effect when combined. Mr. Watson, however, finds difficulty in accepting the conception of its mobility and unsaturated (or rather unsaturating) character, but I think that analogies can be found to help him, such as the mobility of small groups like methyl (conversion of methylaniline into toluidine for example), and the "residual affinity" even in NaCl (whereby it is convertible into $NaClO_3$). Even the ionisation of HCl in water is an analogous case to the conversion of triad nitrogen into pentad—the result being in each case the creation of two new valencies; but in the case of nitrogen the x is supposed merely to become loose without escaping from the molecule.

In conclusion, I should like to indicate that from farther study of the question I am convinced that the supposition made in the body of my paper about the atomic weight of helium is untenable, and that I now prefer the other hypothesis (given in the note at the end) of the existence of a sub-atom of atomic weight about 0·10 (given as 0·09 in the note). I am now satisfied that helium is 4·00, and that my theory must be changed by putting $(2x + 0{\cdot}1)$ wherever He was used in my formulæ, and that zoikon may be an inseparable group (viz., Hx) rather than an element. This latter idea I find has also occurred to Sir J. J. Thomson—see *Corpuscular Theory of Matter*, as follows:—

"A system of four atoms, each possessing unit positive and unit negative valency rigidly attached to each other and forming the four corners of a regular tetrahedron, would possess the same chemical properties as the carbon atom; two such atoms could be united by one, two, or three bonds, while the free valencies not satisfied by the connection between the atoms would be satisfied by any univalent atoms whether electro-positive or electro-negative."

I have studied somewhat closely the evidence for the existence of the extraterrestrial gaseous elements, and wish to put the results on record here. Coronium, from the sun's outer atmosphere, accompanies hydrogen, but lies farther out and therefore is lighter than the latter. I propose to annex it as the representative of the atomic weight 0·1 which I require to explain glucinum, magnesium, etc. The spectrum lines of coronium are λλ *5,303*, 5,536, 5,118, 4,686 (?), 4,361, 4,231, 4,086 [*3,987*, 3,801, 3,643, 3,456],[*] 3,388, 3,381, and are characterised by remaining sharp whilst the chromospheric lines are distorted by solar storms—the only explanation possible is that coronium occurs high above the storm zone. In the upper chromosphere also appear H and He, together with λ 5,316 of unknown source, and λλ 5,184, 5,173 and 5,167 due to magnesium or a decomposition product of magnesium; also λλ 5,169 and 5,019 ascribed also to iron or its decomposition.

The other mysterious element is nebulium, the main constituent of the gaseous nebulæ. It is apparently heavier than both hydrogen and helium, which can be explained by making it the same as my zoikon (which if of diatomic molecule would have a molecular weight of 6). Its spectrum lines are λλ 5,007, 4,959, and 3,727. Taking all the irresolvable nebulæ together, some 40 lines have been measured, of which 11 belong to H, 5 to He, 3 to nebulium, and the following 17 are of unknown origin and may in part belong to my element x: λλ 5,680, 5,540, 5,313, 5,183, 4,790, 4,743, 4,715, 4,662, 4,363, 4,265, 4,145, 4,067, 3,968, 3,869, 3,460, and 3,390; also 5 lines very dubiously assigned to nitrogen, oxygen and silicon. In connection with this, Mr. Eugen Schilz, of Pretoria, has called my attention to an investigation of the Lyra nebula by Prof. Max Wolf, who by using a slitless spectroscope found the rings of different colours to have different sizes; that of λ 4,688 (generally classed as "Rydberg hydrogen") was smallest; the 5 ordinary hydrogen rings were equal and rather larger, viz., λλ 6,561, 4,861, 4,339, 4,101 and 3,968; next came 4 still larger rings due to nebulium, viz., λλ 5,007, 4,959, 3,870 and 3,727, of which the last was the largest and brightest in the spectrum. Of course, however, there is still doubt whether segregation of these gases into rings has taken place in this nebula, since other explanations of the phenomenon have been given; but there is no doubt that nebulium is an element, and there is great probability that it represents the effect of an unimaginably intense heat on our ordinary elements, because, for example, when a new star dies down and its metallic spectrum lines disappear, the line 5,007 remains to testify to the conversion of the colliding worlds into nebular gas.

---

[*] These are sometimes ascribed to another unknown light element in the corona.

Finally, I am not without hope that human ingenuity will achieve means for attaining higher temperatures than it has now reached, and that, when this is done, many of our elements (particularly the halogens, the pentads and the other classes of elements which do not appear in the sun) will be broken up and will exhibit spectrum lines given in the above lists. Even then we shall be far from knowing the atomic weights of these disintegration-products; but I still venture to hope that my theory, even though it is as a whole of very unequal merit, will in its main lines be thus verified by experiment.

I wish to thank Mr. Innes, the Director of the Government Observatory, for the use of modern works on astrophysical research, from which the foregoing data have been compiled.

GRADING ASSAYS AND GRINDING EFFICIENCIES.

(*Read at December Meeting, 1908.*)

By Arthur Yates, A.I.M.M. (Corresponding Member of Council.)

REPLY TO DISCUSSION.

Mr. A. Yates (*Corresponding Member of Council*): I am sorry that some of the members have not come forward with gradings and efficiencies for crushing and grinding machines other than the stamp and tube mill, and so shed a little more light on the subject under discussion. I would also have liked to see some grading assays produced by those who have adopted the "all slimes" process, with a view to justifying the use of such processes or otherwise.

It is pleasing to know that the Mines Trials Committee are making a thorough investigation of the tube mill, and I trust their results will be published in the Society's *Journal*; it is to be hoped their work will extend further and cover all types and makes of crushing and grinding machines, and so establish once and for all their individual merits and thus save untold sums of money that are at present being wasted owing to the general ignorance on this subject.

Edge running machines, such as the Chilian mill will, I believe, be found to give a high efficiency, both mechanically and metallurgically, and I have been pleased to see the prominence given to the Chilian mill in recent papers and discussions on the Rand. All ores are not alike; the fine state of division of the gold and silver in the Redjang-Lebong ore is, I think, rather exceptional, though some of the Mexican ores approximate to it. Many ores will be found to yield the largest profit when ground to all pass a screen between ·0082 in. (60 mesh) and ·0065 in. (80 mesh) on account of the rapid increase in the capital cost of grinding plant required and the working costs. Taking into consideration all details of cost of plant, power, working, depreciation and recovery for two plants, one with stamps only and the other with stamps supplemented by tube mills, simple stamping will be found to deserve very careful consideration. Where fine stamping is desired some method such as returning the classified coarse product from the screens to the mortar boxes again and recrushing with fresh coarse ore, if properly followed up and good classifying apparatus employed, seems to hold possibilities of success.

I thank Mr. White for his kind remarks on my paper, and regret that I am unable to give him any precise information on the size of the gold and silver particles in the ore, but if they do not occur as molecules they must come very near to doing so as will be seen from the elutriation experiments appended.

In considering the sands − 200 it has to be remembered that we leave the measured size of particle from screen grading for particles of greatly varying sizes but of equal falling velocities in water. The ore contains many minerals, most of them heavier than quartz, one of these, marcasite, with which the gold appears to be associated to some extent, crushes more easily and to smaller particles than the quartz; these minerals, on account of their high specific gravity, came down freely with the − 200 sands, thus explaining the increase in percentage extraction between the sands + 200 and − 200.

A correct divisor for water separations could only be obtained by microscopic measurements of a large number of samples of the ore, sands or slimes, as the case may be, but I believe it will be found that by estimating on the total stamped product and the total product after the tube mills going to the cyanide works little difference in results will be obtained by using all − 200 = ·002 in., or sands − 200 and elutriated slimes = ·0025 in. and ·0015 in., or ·002 in. and ·001 in. respectively. The material is apparently steadily reduced from grade to grade, and consequently the assumption of a mean size between two quite distant grades does not materially affect the calculation.

	Sands − 200 and E. Slimes = ·002 in. % effi.	Sands − 200 and E. Slimes = ·0025 in. and ·0015 in. % effi.	Sands − 200 and E. Slimes = ·002 in. and ·001 in. % effi.
3 samples: 70 stamps, 20 mesh, 3 tube mills	58·97	56·73	59·00
7 samples: 70 stamps, 16 mesh, 4 tube mills	66·1	66·2	68·4

To obtain some idea of the fines in the ore and the efficiency value of the ore going to the battery I took two trucks at random and had their contents graded. The crushers (Blake-Marsden) were set to 1½ in. I must say that the fines came out much lower than I expected, though I know of no reason why I should doubt that they represent the ore. On these gradings the battery does not appear to be a favoured machine. Dry weight of ore in trucks, 1904 lb. and 1947 lb. respectively.

Screen.	+ ½	+ 6	+ 8	+ 16	+ 100	+ 150	+ 200	− 200	
% 1	70·03	19·90	1·73	3·78	2·85	0·53	0·10	1·03	= 99·95%
% 2	67·48	17·00	5·54	3·44	2·14	0·54	0·12	3·72	= 99·98%
Average%	68·75	18·45	3·63	3·61	2·50	0·53	0·11	2·37	= 99·95%
Mean aperture	1·25	0·3	0·084	0·051	0·020	0·0047	0·0034	0·0020	inches
Efficiency No.	0·55	0·61	0·43	0·70	1·25	1·12	0·32	11·85	= 16·83

The mean of two twenty mesh battery gradings = 263·4, therefore efficiency per cent. = 1467·8%, or at 4 tons (new figure with two stage breaking) and 2·62 h.p. the relative efficiency = 376·4.

The average of several tube mill samples corresponding to the above was: leaving 318·5, entering 136·8 = 181·7 = 132·8% 181·7 × 51·15 tons = 9293·9 ÷ 47·5 h.p. = 195 6 relative efficiency.

As Mr. White suggests, other separations between sands + 200 and sands − 200, also sands − 200 and elutriated slimes would be of advantage and, I should say, might be carried out on close trials of plant and machinery, but my aim was to put forward a series of gradings and assays which can be carried out by the usual plant and assay office staffs in ordinary working routine. In my own case, the five products give the useful information desired for this mine, and serve as a fairly good check against work in other mining districts.

If Mr. White had the more complete gradings of the two − 90 residues he mentions, one from stamping going 1 dwt., and the other from grinding going 0·6 dwts., I feel sure he would find the explanation of the difference in their gold contents; in working for a given screen product the stamp, on account of the more rapid discharge, will always give a coarser average product than the tube mill. The reason I could not use efficiency factors for calculating extractions during changes in the milling conditions was not due to any differences in crushing detail but to an increase in tonnage reducing treatment time in the sands and slimes plants; extractions were thus adversely affected, the sands plant got a larger tonnage of sands to treat, the fineness of which partly balanced the reduction in time, and the slimes plant got a much larger tonnage of slimes together with an increasing percentage of fine sands, the nature of the material and size of plant both being against good results.

We have taken temperatures of the pulp entering and leaving the tube mills and found a decided increase, representing a considerable loss of energy; this loss of energy, I think, will be found to increase with the fineness of grinding. It becomes a question of whether to give the tube mills a small feed and slow through passage with minimum pump or wheel circulation, or large feed, rapid discharge, and maximum circulation, both of which methods waste plant and power.

It is interesting to record that a mill run with silex liners and imported flint pebbles shows only one-half the wastage attending local liners and reef "pebbles," the angular reef "pebbles" are not so well shaped for grinding and tend to wear flat and slide in the mills. The registered horse-power consumed is as given in the paper, the tube mills being direct driven from motors and ammeter readings regularly taken.

I have to thank Mr. H. Stadler for a copy of his paper read before another Society and for advance notes of his able contribution to this discussion. Mr. Stadler following up the experiments of Professor Kick with falling spheres has evolved a law for the mechanical work done in crushing departing materially from that of Rittinger used in this paper for estimating efficiencies. I must confess that I do not altogether follow Mr. Stadler in his interesting dissertation which has just come to hand and to which as yet I have not been able to give the consideration his views undoubtedly deserve. However, as Mr. White has dealt with many of the points raised I may pass on to other matters. The relative efficiency method given in my paper is a combination of theory with practice, taking into account work done as represented by the measure and weight of the original and final products with the mechanical power actually used.

By taking the figures obtained for relative mechanical work, and calculating as for relative efficiency, *i.e.* $\left(\frac{\text{R.M.W.} \times \text{Tons}}{\text{H.P.}}\right)$ a table of order is obtained corresponding to the order for relative efficiency to unity, though the proportional differences between numbers is more marked in the former, which is to be expected owing to the great divergence from theory of the power actually consumed. Mr. Stadler has misread me in saying I give an example of the unreliability of the efficiency numbers for the finer grades; the example given was to illustrate the need for working with precisely similar figures for all purposes of comparison, if one sample is based on all—200 = ·002 in., then another cannot be worked out with sands—200 = ·0025 in. and elutriated slimes = ·0015 in. If looked at in this light Mr. Stadler's fears for cigarette ash or dust may be dispersed.

The sizes for screens and times for screening as given in my paper were only decided on after a long series of tests. It is absolutely necessary in screening to work to a fixed time on a given size of sample, otherwise the personal equation will be found to give very varied results. From my own work I should say a large number of gradings is more necessary for checking the extractions on grades than for estimations of mechanical work; with the latter, little variation in results will be found even between very distant grades.

Appended is a table of all the stamp and tube-milling results for the year 1908, 348 samples in all; this table gives a considerable amount of information on stamping and tube-milling, and it will be seen that the efficiency-tonnage number for stamping is very regular, also the relative-efficiency number. The table shows that tube mills do much better work on coarse material than on fine, their efficiency falling off rapidly. As the efficiency number expresses the amount of work done by measure and the tonnage by weight (power remaining the same) it is evident that the products of the two dimensions should agree. We may write it: "Within the usual practical limits of coarse and fine crushing by stamps of any given weight, and working under average conditions, the work of stamping through any screen as represented by the relative efficiency to unity should always be 1·000." From the results shown in the table I have constructed a curve based on the "efficiency-tonnage" results of the 12, 16 and 20 mesh, on account of their close agreement. (Diagram 1.) The central portion of the curve applies very well to working conditions, or, as I have stated, "within the usual limits of coarse and fine grinding by stamps"; but when we come to the ends it appears necessary to take other conditions into consideration, two of these no doubt being the weight of particle (heavy or light) and the water used. At the two tons mark we reach a condition of 100% – 200, where the lightness of the particles would make crushing by stamps very difficult. At the other end of the curve the small amount of fracturing for stamping, the large pieces of ore and their rapid removal from the mortar box come forward for consideration. Has any member of the Society crushed ore through a considerable range of screens and plotted out the results with tons as ordinate and screen aperture as abscissa? Here we have only worked through a narrow range of screens, but so far as we have gone I set down the results. (Diagram 2.) It will be seen that with screening finer than 35 mesh a rapid drop in tonnage—due no doubt as above mentioned to the difficulties attendant on fine stamping—must take place, and with screening coarser than 16 mesh the line begins to flatten out; a better and more powerful water feed, to effect a quicker removal of the crushed material from the dies and to lift the large particles to the screen, also the use of a lower discharge, might yield improved results.

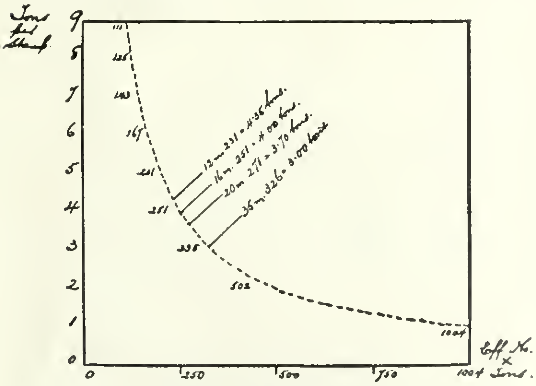


DIAGRAM I.—Efficiency Tonnages.

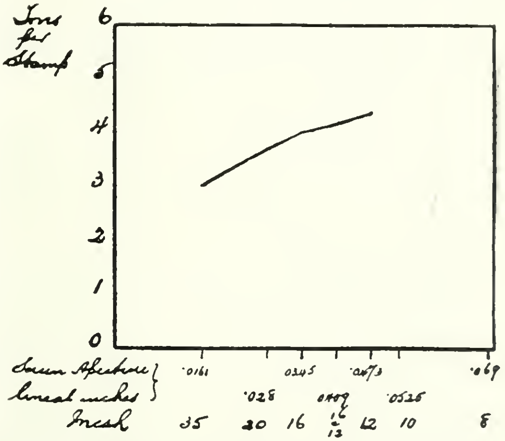


DIAGRAM II.—Screen Aperture and Tonnages.

I have heard of some experiments being conducted on the Rand some years back with screens of ·02 in., ·04 in., ·06 in. and ·08 in. aperture, resulting in 5, 6·5, 7 and 10 tons crushed; were these short trials on small amounts involving cleaning out of the mortar box, etc., or long working runs such as made here? These figures for the Rand give practically a straight line showing the maximum possible duty.

From the grading results obtained from the ore, battery and tube mills, I have sketched a

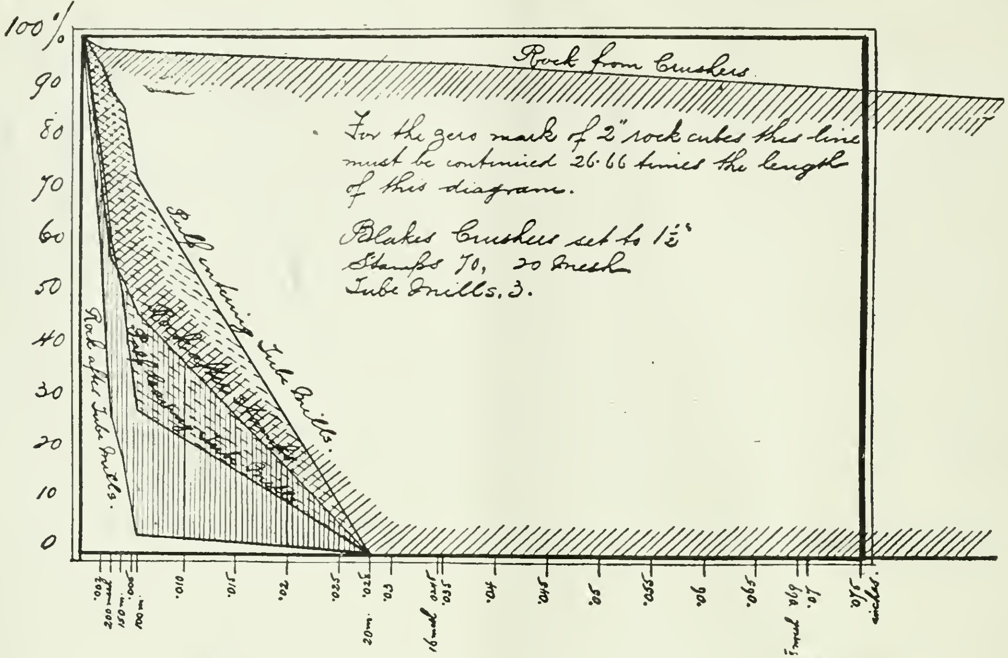


DIAGRAM III.—Cumulative Percentage Curves for Stamps and Tube Mills.

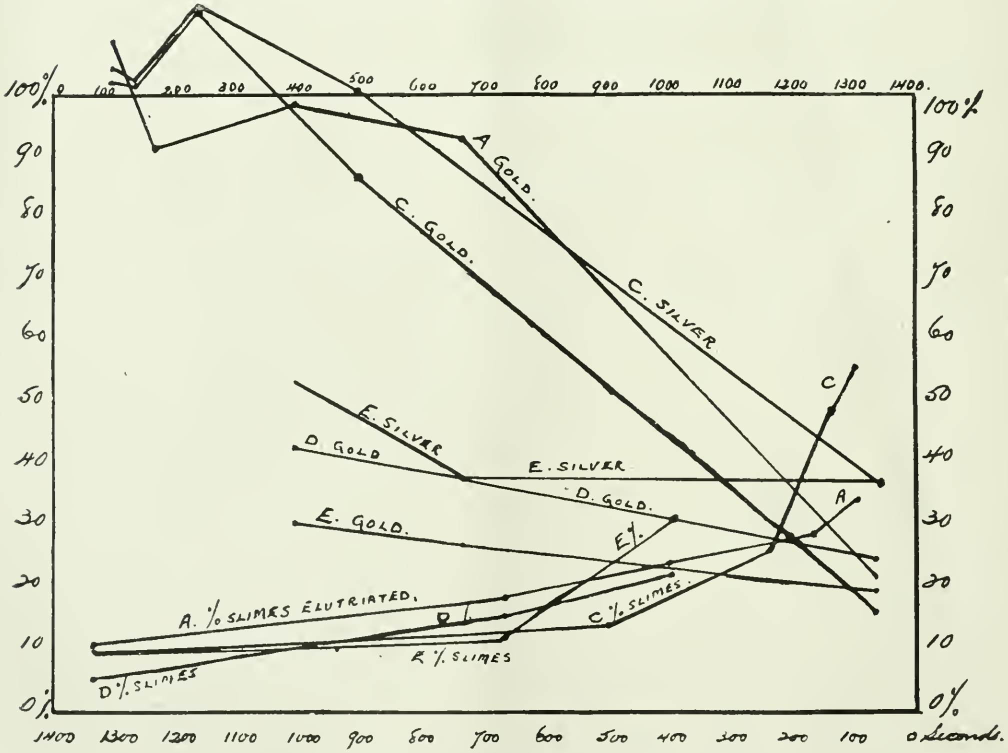


DIAGRAM IV.—Elutriated Slimes Assays. Series A. C. D. E.
Series A.—The alkalinity of the water was about 0·006% CaO.
,, C. ,, ,, ,, 0·004 ,,
,, D. ,, ,, ,, 0·004 ,,
,, E. ,, ,, ,, 0·004 ,,

cumulative curve which shows very clearly what a small part the tube mill plays in the reduction of the ore. The edge hatching is the boundary of the stamp diagram of work, the vertically hatched portion is the tube milling diagrams, the dotted figure is a diagram of work done on classified pulp in one passage through the tube mills. (Diagram 3, see preceding page.)

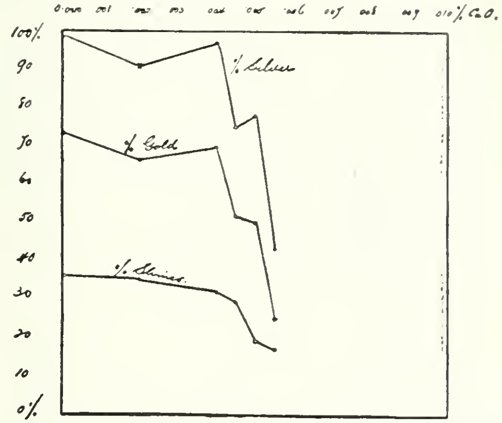


Diagram VI.—Elutriated Slimes Assays. Alkali Series.

All tests done in duplicate and the mean of results taken. Settlement time in every case = 1,334 secs.

I had noticed that the very fine slimes passing out of an old slimes pit, owing to lack of lime in the water for proper settlement, assayed considerably lower than the average daily slimes assays: this led me to carry out a series of elutriation tests on the slimes with the hope of being able to separate a portion as too poor for treatment, and thus save plant and treatment costs. Some of the results obtained are set forth in the Diagrams 4, 5 and 6. Although I did not find a sufficient reduction in value on our slimes to enable me to discard any of it, a similar percentage reduction should on many other mines render the slimes valueless for treatment, so the information may be of use to some of our members. The diagrams show the value of a continuation of screen grading assays into the elutriation stage and also convey some idea of the very fine state of division of the gold and silver in the Redjang Lebong ore. The vessels used for the experiments were white enamelled buckets, 300m.m.

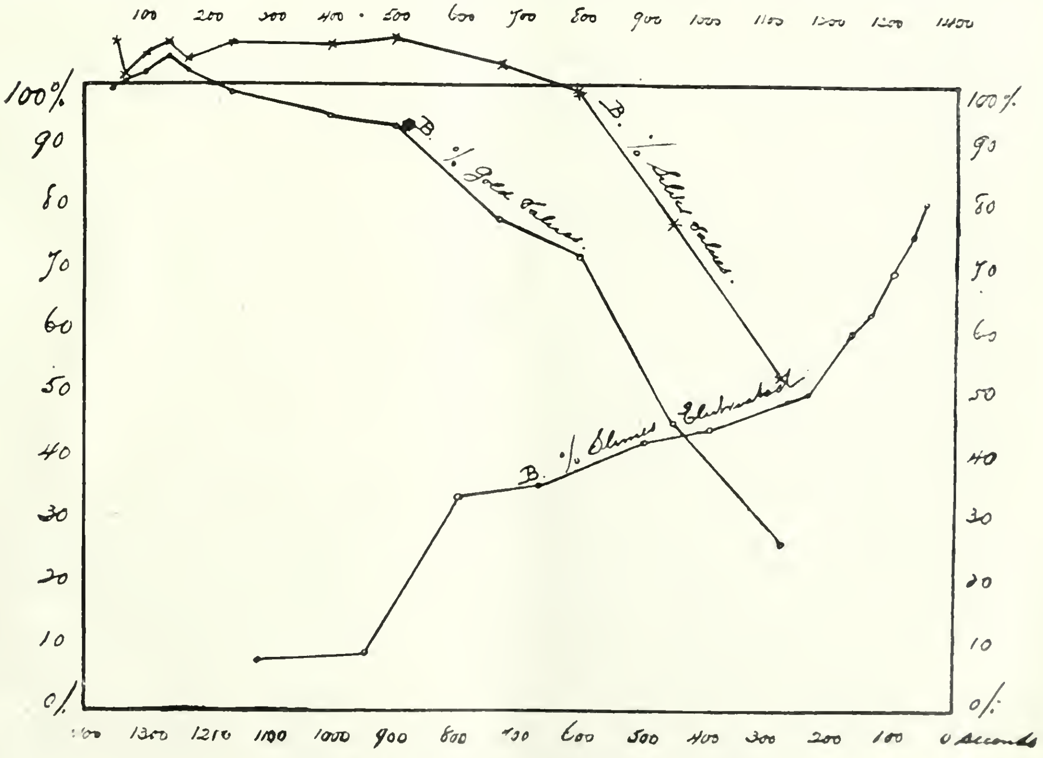


Diagram V.—Elutriated Slimes Assays. Series B.

The alkalinity of the water was kept between 0·004 and 0·006% CaO. The last two tests in the series were made on a new sample of slimes, the original sample having met with an accident (946 secs. and 1,121 secs.). Each result is the mean of three tests.

dia. at the top and 225m.m. at the bottom by 280m.m. deep ; three 22½ m.m. plug holes were made in the side of a bucket, 125m.m., 175m.m., and 215m.m. to the bottom of the holes as measured from the top of the bucket. At first separations were done using the bottom plug hole, but from fear of drawing out some of the settled product by suction, all separations now finish with the middle hole. The following table (Table 2) gives the sizes and settlement times for free settling particles of quartz, according to Prof. R. H. Richards. The times given are for fresh water, but in these tests there was always a little lime present to correspond to working settlement conditions. One and a half litres of thick slimes pulp was taken for each test, the bucket filled with water, the contents mixed well, then allowed to settle and the plugs drawn at the stated times. The filling up with water, mixing, settling and decanting was repeated five times for each test. The total decantations were then perfectly settled, the supernatent water

Table I.—Stamp and Tube Mill Efficiencies.

January to June, 1908.

No. of Stamps.	Screens.	No. of Tube Mills.	Efficiency Numbers.	Tons.	Efficiency Tons.	No. of Stamps.	Screens.	No. of Tube Mills.	Efficiency Numbers.	Tons per Stamp.	Total H.P. consumed	Relative Efficiency	Relative Efficiency to Unity.
	35		326	3·00	978		48 stamp samples					388	1·000
	20		271	3·70	1,003	70	16		259	4·00	183	396	1·02
	16		259	4·00	1,036	70	20		271	3·70	183	383	·987
*60	16	3	363	4·00	1,452	70	35		326	3·00	183	373	·961
*60	16	2	378	4·00	1,512	*60	16	2	378	4·00	252	360	·927
70	20	3	415	3·70	1,535	70	16	3	386	4·00	325	332	·855
70	16	3	386	4·00	1,544	70	16	4	438	4·00	373	328	·845
60	20	4	442	3·70	1,635	70	20	3	415	3·70	325	320	·824
70	20	4	444	3·70	1,643	70	20	4	444	3·70	373	308	·793
70	16	4	438	4·00	1,752	*60	16	3	363	4·00	299	291	·750
						60	20	4	442	3·70	347	282	·726

July to December, 1908.

No. of Stamps.	Screens.	No. of Tube Mills.	Efficiency Numbers.	Tons.	Efficiency Tons.	No. of Stamps.	Screens.	No. of Tube Mills.	Efficiency Numbers.	Tons per Stamp.	Total H.P. consumed	Relative Efficiency	Relative Efficiency to Unity.
	16		251	4·00	1,004	70	12(½)&16(½)		246	4·15	183	390	1·005
	12		231	4·35	1,004	70	16		251	4·00	183	383	·987
	12(½)&16(½)		246	4·15	1,021	70	12		231	4·35	183	380	·979
45	30	2	448	3·00	1,344	70	16	3	413	4·00	325	355	·914
70	16	3	413	4·00	1,652	70	12	4	417	4·35	373	341	·878
70	16	4	432	4·00	1,728	70	12 & 16	4	428	4·15	373	333	·858
70	12(½)&16(½)	4	428	4·15	1,776	70	16	4	432	4·00	373	324	·835
70	12	4	417	4·35	1,814	45	30	2	448	3·00	212	285	·734

* Sands separators between battery and tube mills are of insufficient capacity.

Table II.— Sizes and Settlement Times for Free Settling Particles of Quartz.

Diameter of Quartz Particles.		Settling Velocity mm. per Second.	Settling Velocity in Seconds for 175 mm.	Diameter of Quartz Particles.		Settling Velocity mm. per Second.	Settling Velocity in Seconds for 175 mm.
in Millimeters.	in Inches.			in Millimeters.	in Inches.		
0·076	0·003	3·6	48·6	0·0188	0·00075	0·265	667
·063	·0025	2·52	67·4	·0182	·000725	·22	795
·05	·002	1·78	98·4	·0161	·00064	·185	946
·044	·00175	1·30	134·5	·0144	·00057	·156	1,121
·0377	·0015	1·06	165	·0126	·0005	·131	1,334
·0319	·00125	·746	234	·00884	·00035	·110	1,591
·025	·001	·442	395	·00589	·00025	·0924	1,893
·0219	·000875	·352	497				

drawn off, and the sample dried, weighed and assayed: this gave the percentage and value of the elutriated slimes, after drying and weighing the remainder in the bucket. The experiments were made on different samples of slimes, as it was thought that some change in the nature of the slimes might take place by standing for a time in the alkaline water due to the lime added at the battery. The quantities of slimes elutriated and the assay values varied somewhat, as will be observed in the diagrams; this I believe to be due to differences in the slimes tested, but we may say generally that the first elutriations show a slight concentration of gold and silver (as might be expected owing to specific gravity in the case of pyrites and quartz carrying gold and silver), then follows a regular curve of decreasing values until we reach a point showing approximately that 10% of the slimes assaying only 20% of the original gold value and 36% of the original silver value, might be separated. Such a separation might be made in well designed pyramidal slimes spitzkasten. In conjunction with the above tests, I also tried the effect of working with a fixed time and varying amounts of lime in the water.

I would take this opportunity to correct the following misprints which appeared in my paper as published:—Page 188, line 8, the gradings should read +100+150+200, sands—200 and elutriated slimes; line 13, read (Greening's SWG brass wire screening); page 190, table of sands grading samples, under "General Sample," the silver should be 29·8 dwts., and the percentage silver extraction 73·8. The dates for the slimes grading samples on the same page should be June 1—15 and 16—30.

STOPE MEASUREMENTS.

(*Read at May Meeting, 1909.*)

By Otto S. Tonnesen (Member).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I criticised the author's treatment of the sag of the measuring tape, and Mr. Coaton now defends the author by saying that the approximation gives good results. My point was that an approximation should never masquerade as truth, unless it is specifically stated to be an approximation.

Mr. Tom Johnson (*Member*): There are some statements in Mr. Tonnesen's paper that should not be allowed to pass without being challenged. I have waited, expecting that some of our surveyors would get around and cause a good discussion. It is no use their saying the statements do not need troubling about, as people know that is nonsense. Our proceedings are noted all over the world, and I do not think that the statement or inference that fathomage cannot be measured correctly by our present methods should be allowed to go forth without challenge, for notwithstanding what Mr. Tonneson has said, I do not think that there is any reason whatever for much error in present methods of measuring if competent men are employed to do the work. I know I have met many surveyors and none of them would be guilty of the slovenly work necessary to cause the drawbacks to the several methods in use at present as pointed out by the author.

As Mr. Tonnesen truly says there are many miners who are very suspicious of stope measurements. "I have known them suspicious of drive measurements, and would not go to the expense or trouble to run a tape over their work," and for that reason the author should have been more guarded in his statements.

The stereometer may, no doubt, make stope surveying easier work, but that is no reason to imply that stopes cannot be measured properly by present methods, or that surveyors cannot place reliance on their work. Many miners on reading the paper would naturally come to the conclusion that the rumours of mismeasurement are correct, and for that reason exaggeration should have been kept down.

On the night this paper was read one of our members instanced a case of his own. If he is correctly reported the manager asked him if he could recognise his stope on paper if he saw it. He said as soon as he saw it he knew he could draw a better plan with his eyes shut; where he had a 12 ft. to 14 ft. box-hole there was only a 6 ft. box-hole shown on plan, and the drive pillars were shown as 6 ft. where they ought to have shown 16 ft. Now there is a peculiarity about that plan that I would give all of a penny to see. To be able to show a box-hole as only 6 ft. in a 16 ft. rib is quite an accomplishment, I should say, either with eyes open or shut.

My friend, Mr. Saner, mentioned that when he was a surveyor, he had to make himself scarce for two or three days after the end of the month. Why, had he no faith or reliance in his work? This accusation of using bad methods is up against him as well as others. Had he no words of defence or does he agree with Mr. Tonnesen?

Stope measurement by theodolite is rather troublesome no doubt, but there is no need for greater errors than the limit. As Mr. Coaton points out, a little brains help to smooth things.

Tape triangulation, sag, errors of long offsets, these Mr. Coaton has dealt with. He points out that the triangulation should be carried on independently of the drive pegs. Once the base line is carried through the winze there is no further need of drive pegs. This saves trouble, as then the incline distance in crossing faults, changes of dip, etc., do not worry the surveyor in plotting as the distances are laid on the flat on the one sheet of paper. Fig. 7 in the paper would not need plotting in three main sections, it would simply be laid out flat. As to sag, many surveyors tie themselves to base lines of no more than 100 ft. I have not met a surveyor who would dream of using a base of 312 ft. under the conditions shown in Fig. 1. The offsets shown in Fig. 4 in the paper are extreme, the shortest distance will surely come nearer than there shown.

Mr. Tonnesen does not like the stretched strings and German dial and Mr. Coaton agrees with him. Did either of them think of how the idea of stretched strings could be used in stope measurement? If tapes, with say two feet of thin wire at the outer end, were used the angle between tapes could be measured with a large protractor. I have used the method at times for setting off winzes and raises.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(*Read at May Meeting, 1909.*)

By Jas. Gray, F.I.C. (Member).

DISCUSSION.

Dr. J. Moir (*Vice-President*): The President hauls me over the coals for saying, as he puts it, that "members willingly withhold fair criticism on papers," and in so doing states that discussion on this paper in open meeting had not been hitherto possible. Seeing that this is the last meeting for discussion this assertion seems due to oversight, for, though absent I have been assured that there have been meetings at which the whole agenda was gone through by 9.30.* As for the main question, I have to say that if the President's description of the procedure on the mines is correct, then I am in the wrong; but I venture to submit that the procedure he describes is not the rule but the exception. Unless the lime is paid for by weight *received* by the mines (not that despatched from the works), the burners are cheating themselves. I have no doubt whatever on the other hand that what members say about the burners having other means for equalising matters is quite true; but, as I say, the only way of being fair to both sides is to weigh lime on receipt.

Messrs. K. L. Graham (*Member of Council*) and **R. Lindsay** (*Member*): The author's paper raises several points about which considerable divergence of opinion exists amongst those who control its use on our mines. Attempts have been made to standardize a method of testing the quality of the lime supplied which could be carried out in the reduction works, but none that we know of have proved satisfactory. The form of argument usually adopted by the lime burners or their agents is to produce a certificate showing a full analysis of their lime or the total CaO by the sugar test. These results are often economically misleading, when the manner in which the lime is used in practice is considered, the percentage of caustic lime available depending on such factors as its state of division and the point in the cycle of operations at which it is fed. It appears to us therefore that the only useful test would be one in which working conditions were simulated as nearly as possible. An attempt is made to do this in the following method, which has nothing novel about it (see E. H. Croghan's paper in *Journal* for August, 1907), and can be carried out by anyone with ordinary intelligence within an hour of the time of taking the sample. Five bags of the lime are taken at random from the truck as received, the bags being weighed as a check on the consignees. The contents are dumped on to a sampling floor and quartered down till the sample is about two or three pounds in weight. From this two or three samples of 1 gm. each are weighed off, each gm. being shaken in a litre of distilled water (in a bottle of about two litres capacity to allow proper agitation) for one minute, after which the undissolved matter is allowed to settle for 15 mins. 50 cc. are drawn off with a pipette, and the alkalinity as CaO determined with oxalic acid. The sample is again shaken for one minute and settled again for 15 mins., when another 50 cc. are drawn off and tested. The sample is shaken again in the same way and a third test made. The lime sample is also graded through sieves of 100 and 200 meshes to the linear inch, and where a sliding scale of payment is arranged on a basis of this test, the balance of the sample should be placed in a stoppered bottle to be kept for reference till the accounts are settled.

Before proceeding further let us refer you to Mr. Gray's remarks. He, after drawing your attention to the rapid deterioration of quicklime when exposed even for a few days to the weather, proceeds to suggest remedies. The first of these only appears to us either practicable or feasible, but as it solves the problem no others are necessary. He says, "all lime should be slaked before being despatched to the mines." In this we cordially agree with him. The fact that you handle and pay for 32% of water is more than counterbalanced by the advantages derived from using lime in a very fine state of division, and therefore in the most suitable condition for our purpose, a condition which, when dealing with quicklime, could only be arrived at after a considerable outlay for power and labour. For those and other reasons nothing but slaked lime has been used at the Geldenhuis Deep Works for over four years.

---

* Dr Moir has been incorrectly informed.—Ed. Com.

In June, 1905, a series of experiments were conducted here with the object of comparing the results obtained from blue and white limes under certain clearly defined conditions, which were these:—At that time all the lime was fed automatically into the main slime pulp launder at a point immediately beyond the return spitzkasten, the common practice in those days, baffles being fitted to agitate the pulp in transit. The time available for its solution, that is, between the point at which it was fed and the delivery from the launder into the slimes settling tank being about one minute. It must be borne in mind that our interest finished with the percentage of alkalinity in the slime pulp at the point of inflow to the settlers, it being necessary to maintain this at a certain figure for coagulation purposes. Any lime undissolved on reaching these tanks settled and was quickly covered by the slimes, becoming unavailable for this purpose. To arrive at the lime dissolved in the one minute period we used the test previously described, thus simulating working conditions as nearly as possible. Table A shows the results of three representative tests.

"A."

	63/- per ton White Slaked, % CaO.	40/- per ton Blue Slaked, % CaO.	55/- per ton White Slaked, % CaO.
1 gr. shaken in a litre of distilled water for 1 min.	15	14	14
After shaking for 1 hour	42	20	36

GRADINGS OF ABOVE LIMES.

"B."

+100	40·82%	27·22%	39·76%
+200	21·36%	12·94%	17·96
−200	37·82%	59·84%	42·28%

From table "A" it will be seen that when working under the conditions described the time available for the solution of the lime being limited to one minute, the blue lime was the cheapest, being equally as efficient for our purpose. On plants where part or whole of the lime is added in the mill a higher solubility would naturally be expected, but anything like the total CaO is not obtained. With a lime showing 52% available CaO, and using a 600 screen in the mill, only 30·7% of this available lime (or 16% on the weight of the sample) was obtained according to the alkalinity of the slime pulp entering the collectors, and with the coarser screens now in use even this low percentage will not be obtained, as almost all the coarse particles will pass through easily, and, as is well known, these coarse particles dissolve but slowly under the most favourable conditions. Only the very coarse particles are separated out from the mill pulp by the spitzlutten, and consequently a tolerably small percentage of the lime fed in the mill gets ground in the tube mills. Without making any attempt to belittle the value of a full analysis of any lime, it may be pointed out that the figures derived from it are really not of so much guidance to the consumers here as would seem at first glance, but what is really of service is any method which will give us the percentage of CaO we are likely to dissolve out under working conditions, and the sample tested in this way and giving the highest percentage of alkalinity to the slime pulp in the limited period of time, taken together with the price, should enable us to decide which lime to buy for our purpose.

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

(Read at May Meeting, 1909.)

By the PORTLAND METALLURGICAL SOCIETY (edited by THOS. B. CROWE).

DISCUSSION.

Mr. Thos. B. Stevens (*Associate*): Some few years ago I did a large amount of experimental work on the treatment of low grade Cripple Creek telluride ores by cyanidation unroasted. I have therefore read the paper of the Portland Metallurgical Society with very great interest.

On referring to one of my old note books, I find a record of some experiments made in London in 1903, on ore from the North Boulder Mine, Kalgoorlie, in which ammonium persulphate was used as an accelerator to the ordinary alkaline cyanide solution. The results, using ·1% of ammonium persulphate and ·2% of potassium cyanide on 1 oz. ore showed an increased extraction of 1 dwt. compared with ordinary straight cyanide agitation, but the extraction even then amounted to only 50%. The addition of iodide was not thought of, and as other work was pressing at the time, the experiments were not continued further.

The subject of "oxidising accelerators" has always been a very much debated and also patented one, and it is a remarkable fact that on a working scale cyanogen bromide is the only one which has stood the test of time, and even this continues to be used in one place only, viz., Kalgoorlie.

I should like to ask the authors if they have made any comparative experiments using bromo-cyanide against potassium cyanide and persulphate, and if so does the latter method give a very much better extraction of the gold? for I consider that the bromo-cyanide treatment would be considerably cheaper in operation. In comparing the two processes we are at once confronted with two questions; firstly, does it get out more gold? secondly, is it cheaper?

Viewed from a chemical standpoint only, it seems probable that cyanogen iodide would be more efficient than cyanogen bromide, as the action of the iodide would be the slower and, therefore, more prolonged. The bromide is found to be entirely decomposed three hours after it has been added to the ore pulp, and it is for this reason that high grade charges have often to be recharged with the "bromo" in order to obtain low tailings. I have also experimented with cyanogen chloride, and found the decomposition to be so rapid as to give no increased solution of the gold.

The cost of chemicals for bromo-cyanide treatment at Kalgoorlie has always been one great disadvantage of an otherwise successful process. As I have said previously, I think that the substitution of iodide and persulphate would considerably add to the cost. Potassium iodide is a very expensive salt, and I consider it is unsafe to assume that it would remain in the solutions and be used over and over again with practically no loss. There is a possibility of loss by precipitation into the residues as insoluble iodide; there might be considerable loss by volatilisation of cyanogen iodide (there is with bromo), and there must always be loss owing to the accumulation of weak solution, even if filter presses or vacuum filters are employed; I think we may regard decantation out of the question where such expensive chemicals are used. The present average charge of bromo-cyanide in use at Kalgoorlie is ·5 lb. and some low grade mines (say, under 8 dwt.) use as little as ·2 lb. Bromo salts cost 1s. 11d. lb., so that the loss of iodine would have to be exceedingly small indeed to enable it to compete in cost with bromide.

Probably some of the authors have experimented with bromide instead of iodide together with persulphate and cyanide. I am afraid that hypo-bromite, instead of hypo-iodite, is not likely to be formed, thus preventing another possible process, but if the experiment has not been made it would be well worth trying.

My own experiments with Cripple Creek ores showed that a good extraction could be obtained by bromo-cyanide treatment, fine grinding being a *sine quâ non.* At Kalgoorlie it is found most economical to take out the pyritic portion of the ore by concentration, and treat it separately by roasting and cyaniding; otherwise in treating this very refractory material together with the bulk of the ore, the cost for bromine salts, to get a good extraction, is found to be excessive. With Cripple Creek ores I found this also to be the case, and, in this latter district, I believe that the small amount of high grade concentrate could be more profitably shipped to the smelters instead of treating it at the mine.

In concluding these few remarks, which I trust may give some small help towards solving this interesting problem, I should like to thank the Portland Metallurgical Society, on behalf of our Western Australian members, for giving us the benefit of their research on a subject of such common interest.

The meeting then closed.

(A paper submitted to the Institution of Mining and Metallurgy, London, and to be discussed simultaneously by the members of the Institution and this Society. Reprinted from the Bulletin of the I.M.M., August 12, 1909.)

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

By W. A. Caldecott (Member).

The history of ore crushing by means of gravitation stamps shows a progressive increase in their weight and in corresponding efficiency. This holds good in the development from the square wooden non-rotating stamps in use in Germany in the Middle Ages[*] to iron-shod wooden stamps in Cornwall, and then to rotary iron stamps in California and mild steel stamps in South Africa. The latter have greatly improved during the last few years, and probably represent the highest existing stage of development. About 1835, the first stamp mill in the United States was started at Tellurium, Virginia.[†] The stamps weighed 50 lb., and were made of square wooden stems with iron shoes and dies, the cams operating in slots in the stems.

---

[*] See Remarks by Bennett H. Brough, *Proc. Inst. of Civil Eng.*, vol. cviii. (1892), during discussion upon paper by A. H. Curtis on "Gold Quartz Reduction."

[†] See Crane's "Gold and Silver," p. 473 *et seq.*

Ten years later the Vaucluse Mine installed 380-lb. stamps, of which the head weighed 125 lb. When the first batteries were erected on the Rand over 20 years ago, Californian practice probably represented the highest state of the art, and was introduced upon these fields by J. S. Curtis, J. H. Hammond and other well-known Californian mining engineers.

To illustrate the type of battery in operation on the Rand in 1889, in which year the writer's experience of stamp-milling began, the following details may be given of the Du Preez Gold Mining Company's mill, premising that this was before the introduction of the cyanide process, and that concentrators, other than blanket-strakes and buddles, were not in common use.

The mill was erected in that year, and was of Sandycroft make, designed in accordance with the best Californian practice to date. It consisted of 20 stamps weighing 900 lb. each when new, and was run at the rate of 90 7-in. drops per minute. The stamp duty was about 3 tons per 24 hours, using a screen of 900 meshes per sq. in.

By 1899 some progress had been made and 1,250-lb. stamps were in use at the Robinson Gold Mining Company and elsewhere, though in Australia and the United States 900 lb. per stamp remained the almost universal maximum limit, and this remains true of the celebrated Homestake mills at the present time. At the Mount Morgan Gold Mining Co. (Westralia), in December, 1902, 30 stamps weighing 1,500 lb. each were erected. These were eclipsed, however, by the neighbouring Millionaire Gold Mining Company's battery of five stamps weighing 1,750 lb. each.* The advance made on these small companies was not generally realised, and in 1907 few stamps dropping on the Witwatersrand weighed above 1,250 lb., with the exception of certain batteries of Consolidated Goldfields companies, as will be noted later.

In 1904, the author was authorised by Mr. R. M. Catlin, Acting Consulting Engineer of the Consolidated Goldfields, to carry out a series of milling tests at the Knights Deep battery, which was equipped with stamps weighing 1,350 lb. when new. The object of the trials was to discover some means whereby stamp-milling efficiency could be increased.

The initial experiments were made to test the effect of fine breaking before stamp-milling, and also whether the increased discharge area afforded by a double discharge mortar-box increased the stamp duty. A pair of rolls was installed to effect the preliminary fine breaking, and also a double discharge mortar-box. A staff of men was specially employed to carry out the trials, and most complete and detailed records were kept.

The tests extended over more than a year, and a considerable amount of money was expended in testing many variations of conditions, such as varied angles of screen frames, heights of discharge and of drop, and different methods of water introduction into the upper portion of the mortar-box.

The following statement illustrates some of the many experimental runs :—

	(A) Single Discharge Mortar. Ordinary Mill Feed.	(B) Single Discharge Mortar. Ore from Rolls set 0·5 in. apart.	(C) Double Discharge Mortar. Ordinary Mill Feed.
Running Weight of Stamp ...	1,343 lb.	1,342 lb.	1,356 lb.
Set Height of Drop	$7\frac{1}{2}$ in.	$7\frac{1}{2}$in.	$7\frac{1}{2}$ in.
Drops per minute	98	98	98
Duty per Stamp per 24 working hours	5·85 tons.	5·68 tons	5·81 tons.
Height of Discharge ...	$2\frac{3}{4}$ in.	Level	$3\frac{3}{4}$ in.
Screen used and Aperture ...	700 (·024 in.)	700 (·024 in.)	700 (·024 in.)
Ratio of Water to Ore by Weight ...	7·67 to 1	8·35 to 1	10·70 to 1
Percentage of + 60 (0·01 in.) grade in screen pulp	29·00%	27·50%	26·00%
Ore Feed + $1\frac{3}{8}$ in.	54·8	1·1	49·9
− $1\frac{3}{8}$ + $\frac{3}{4}$ in.	14·3	19·3	16·3
− $\frac{3}{4}$ in.	30·9	79·6	33·8

The results of the above typical trials with a double discharge mortar-box and with fine breaking before stamp-milling are almost identical within the limits of experimental error.

The above conclusions were confirmed by the fact that the average stamp duty on ordinary mill-feed for six pairs of parallel trials was 5·82 tons with a double discharge mortar, and 5·82 tons with a single discharge mortar.

The average stamp duty for four pairs of parallel trials on the product from rolls set $\frac{1}{2}$ in. apart was 5·64 tons with double discharge mortar, and 5·78 with a single discharge mortar.

---

* See Special Edition of *Australian Mining Standard*, December 8th, 1904; also *Journal of Chemical, Metallurgical and Mining Society of South Africa*, vol. vii., September, 1905, p. 92.

There was no increased duty to compensate for the extra cost of preliminary fine breaking* and the only difference with the double discharge mortar-box was a considerable increase in the amount of water used.

This last conclusion is in accordance with the fact that the ordinary single battery screen has about 40% of its area available for discharge, or, say, 1½ sq. ft. of discharge area for the small stream of pulp yielded by five stamps.† It will be observed that in trial (B) many flat pieces of ore which had passed the rolls set ½ in. apart yet remained on a coarser screen used in the grading analysis.

Finally it became obvious that to crush more rock it was necessary to strike a heavier blow. This was sought by affixing helical springs round the stem above the tappet so as to be compressed by the upper guide during the stamp's rise, and so that when released the stamp would fall more rapidly, rendering a higher drop practicable, and would likewise strike harder. Whilst this object was in some measure fulfilled, the noise in the mill was much increased and the life of the springs did not promise to be long. Hence, finally, the conclusion was reached that a simple increase of stamp weight was the best course.

A series of runs were then made, whereby direct comparisons could be obtained of the actual tonnage milled by alteration in the weight of the stamps. The table following exemplifies the increased tonnage obtained by increased weight of stamp on ordinary mill feed, both under ordinary milling conditions, and with a high discharge and back water feed to obtain a very fine product.

Another run with 1,337 lb. stamps having a height of discharge of 2 in. and a 64-mesh screen gave a duty of 10·23 tons per 24 working hours, of which 52·8% remained on a 60 screen. The water feed adopted in this test consisted of inlets at an angle of 45° at the back of the mortar-box, as suggested by the late Mr. H. T. Pitt.‡

Encouraged by the results of these and many other tests, including a duty of approximately 6½ tons through 1,500-mesh screening with stamps weighted up to 1,622 lb., the writer felt justified in recommending that the next 360 stamps erected by Consolidated Goldfields Companies, namely, at the Simmer East, Robinson Deep and Luipaards Vlei, should have a weight of 1,550 lb. each when new.§

Such a departure excited much adverse comment among experienced millmen generally, on the ground that no foundations could stand the shocks, but the reasoning that if foundations could be built to endure the blow of the colossal steam stamps, it would be possible for them to be built to withstand gravitation stamps proved correct. All the above mills are satisfactorily at work to-day after periods of service ranging up to three years, and the foundations promise to last as long as did those of their lighter predecessors.

Apertures per sq. in. in screen used.	Running weight of Stamps in lb.	Set Height of Drop.	Height of Discharge.	Tons of Water used per Ton of Ore.	Tons of Ore crushed per stamp per 24 working hours.	% of + 60 (·01 in.) grade in screen pulp.
981 (0·021 in.)	1196	8 in.	3 in.	5·8	5·88	22·63
,, ,,	1279	,,	,,	5·8	6·58	22·23
,, ,,	1531	,,	,,	5·7	6·74	20·86
1,512 (0·016 in.)	1216	8 in.	11 in.	5·40	4·26	5·16
,, ,,	1288	,,	,,	5·30	4·29	4·91
,, ,,	1293	,,	,,	5·43	4·55	9·49
,, ,,	1337	,,	,,	6·27	4·96	6·66
,, ,,	1562	,,	,,	5·05	5·17	—
,, ,,	1605	,,	,,	6·30	6·02	11·66

In accordance with the present changed state of professional opinion, all Rand batteries lately erected or designed have stamps of a weight not long ago deemed extravagant, and aided by the installation of tube mills stamp duties of 8 tons are as common as of 4 tons but a few years previously.*

Whilst in the United States and in Mexico the relatively light stamp is still generally upheld, there is at least one notable exception in the recently built mill of the Boston Consolidated Copper Company, which is equipped with 312 single Nissen stamps of 1,500 lb. weight each and with a 9-ton duty.†

The accompanying diagram, Fig 1, illustrates a modern heavy gravitation stamp with its mortar-box, as used in the Simmer Deep Jupiter joint mill.

The tabular statement following illustrates the results obtained on a large scale under regular working conditions by the use of heavy stamps, and further shows in the case of the Simmer East that whilst stamps alone can be used with a

---

* See also *Journal of Chemical, Metallurgical and Mining Society of South Africa*, vol. vi., Jan., 1906, p. 215.

† See Lock's "Gold Mining," p. 79.

‡ See *Trans. Institute Mining and Metallurgy*, 1904, vol. xiv., p. 164; also *Journal Chemical, Metallurgical and Mining Society of South Africa*, 1908, vol. viii., p. 374.

§ See *Annual Report of Consolidated Goldfields of South Africa* for 1905, p. 32; also Report for 1906, p. 27, and Report for 1907, p. 28.

* See the *South African Mining Journal*, p. 802, September 19th, 1908; p. 1,016. November 14th, 1908; p. 1,100, December 5th, 1908; and p. 1,348, February 6th, 1909; 1,860-lb. stamps with a 12-ton estimated stamp duty are to be erected on the West Rand Consolidated property (*South African Mining Review*, p. 232, November, 1908.

† See *Engineering and Mining Journal*, 14th September 1907, p. 433.

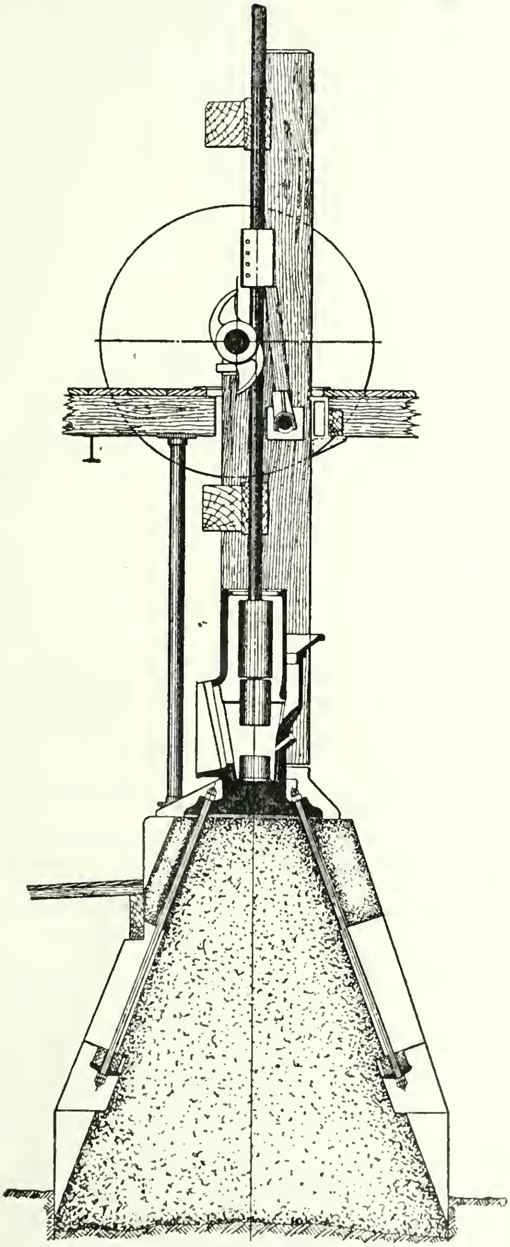


Fig. 1.

Weight of Stamp (New).

Shoe	285 lb.	=	17·066%
Head	410 ,,	=	24·551%
Stem	723 ,,	=	43·293%
Tappet	252 ,,	=	15·090%
	1,670 ,,	=	100·000%

high discharge to produce a fine product, yet the later use of tube-mills as well yielded at the same total cost a product with fewer coarse (+ ·01 in.) particles and hence a still more perfect exposure of the gold.

Somewhat later than the series of battery tests already described, the matter of battery foundations was also investigated. It was recognised that timber foundations were unsuitable for heavy stamps, their liability to decay rendering them less durable than the then recently introduced cast-iron anvil blocks. However, it had never been practically proved since the introduction of these anvil blocks that they were superior to concrete foundations in themselves, and that this point might be elucidated the author in 1906 had a series of 23 comparative trials carried out on two five-stamp batteries side by side in the Simmer East Mill, the one with a concrete block only, and the other with a cast-iron anvil block as well—shown in Figs. 2 and 3.

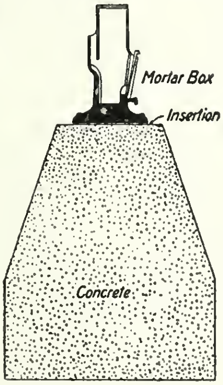


Fig. 2.

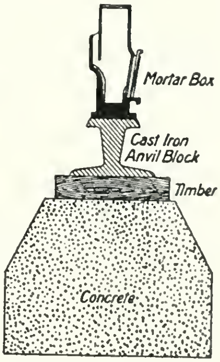


Fig. 3.

The results of the parallel trials were on the average practically identical. These extended from the use of 100-mesh screening to 140-mesh, and the cast-iron anvil block battery showed an average stamp duty of 6·84 tons as against 6·78 tons for the concrete block, the average screen grading analysis of the latter showing, however, only 25·00% of + 60 (0·01 in.) grade as against 26·76% for the cast-iron anvil block.

The question of the life of the concrete block without timber or a cast-iron anvil block has received attention also, and one mortar-box, erected on Mr. H. C. Behr's recommendation at the Simmer East mill, upon a concrete block with only a thin layer of insertion between, has been in use now for three years with no visible sign of cracking or deterioration. This has been so encouraging that at the Simmer Deep and Jupiter mills all the batteries have been erected with the mortar box resting on the concrete blocks.

Crushing is usually performed either by impact or abrasion, and whilst hard ores are dealt with

	LUIPAARDS VLEI.	SIMMER AND JACK EAST.	
	(August, 1908.)	(May, 1907.)	(January, 1909.)
Tonnage Milled during Month...	18,807 tons	35,500 tons	29,600 tons.
Number of Stamps	60	250	130
New Weight of Stamps ...	1,629 lb. (with 18½ ft. stem)	200 at 1,550 lb. 50 at 1,350 lb.	1,550 lb.
Running Weight of Stamps ...	1,520 lb.	200 at 1,450 lb. 50 at 1,250 lb.	80 at 1,450 lb. 50 at 1,550 lb. (with compensating weights)
Average Drops per minute and Set Height of Drop	98·6 at 8½ in.	96 at 8 in.	96 at 8 in.
Duty per Stamp per 24 working hours...	9·667 tons.	5·006 tons.	8·333 tons.
Height of Discharge	4 in.	9 in. (average)	3½ in.
Screen used and Aperture ...	200 (·056 in.) and 250 (·046 in.)	1,600 (·016 in.) and 1,200 (·017 in.)	200 (·057 in.) and 400 (·035 in.)
Ratio of Water to Ore by Weight	8 to 1	8 to 1	6·46 to 1
Percentage of + 60 (0·01 in.) grade in screen pulp... ...	43·7%	10·92%	1·61% (final pulp)
Cost of Crushing per ton of Ore—			
(*a*) Stamps	1s. 3·879d.	1s. 10·424d.	1s. 3·672d.
(*b*) Tube Mills	6·215d.	—	7·070d.
(*c*) Total	1s. 10·094d.	1s. 10·424d.	1s. 10·742d.

by the former method, the latter is applicable to softer material. The relative efficiency of the two methods for hard material is well illustrated by the case of the diamond, which is hardly capable of abrasion at all except by its own dust, whereas, if placed on an anvil, it will be shattered into a thousand fragments by one blow of a hammer.

As has been pointed out by C. de Kalb, the crushing of the larger pieces of ore in stamp milling is done as the result of reaction from the resilient die, the waves of compression transmitted through the ore from the shoe being reflected from the die, until the strains set up from the oncoming and reflected waves exceed the elastic limit of the ore, and rupture results.* The stamp is essentially an impact machine, and hence readily pulverises hard tough ore.

Such ore, if handled in any of the legion of rubbing or shearing appliances, would only be reduced at the expense of serious wear by abrasion of costly steel parts which may have to be discarded when but a fraction of their weight has been worn away. With soft material the case is different, and machines of the latter class may be employed with advantage and without undue abrasion of wearing faces.

It is much to be regretted that no satisfactory machine for testing the capacity of material to resist abrasion, or conversely of testing the abrasive capacity of materials requiring reduction, yet exists, as this property is of prime importance in all crushing operations.

When gravity stamps are used for very fine crushing they pass beyond the economic range, and operate by abrasion rather than by impact, with consequent reduction of efficiency, as may be seen from the following considerations. Impact is essentially dependent on the pressure exerted by the blow, and depends upon the weight of the stamp and the height of the drop. Assume that a 1,750-lb. stamp having a 9·5-in. dia. shoe and with a set height of drop of 8 in. strikes a layer of ore 2 in. thick on the die, and comes to rest 0·5 in. from the die. Then having fallen 7·5 in. and having done work for 1·5 in., it has exerted on the ore a mean total pressure of 8,750 lb.† The shoe having an area of 71 sq. in., the average unit pressure exerted over the whole of its area is only 123 lb. per sq. in., or less than that exerted by a man of average weight supported upon an area of 1 sq. in.

The crushing effect of such a pressure on hard quartz grains is trivial, and a certain amount of reciprocal abrasion of grains of sand re-arranging

---

*See *Mines and Minerals*, p. 135, Oct., 1906; O. H. Howarth in *Mines and Minerals*, p. 441, May, 1906; and *Journal Chem. Met. and Min. Soc. of S.A.*, vol. vi, June, 1906, p. 385.

†Cf. Hiscox, "Compressed Air," p. 437.

themselves under pressure is probably the main effect produced. The case, however, is quite different in the first impact of the stamp on the ore in the box. Assume that only a 2 in. cube resting on the die is first struck; having an area of 4 sq. in., the average unit pressure here is 2,190 lb. per sq. in., or eighteen times as much as under former conditions.

The finer the crushing is carried in the battery the longer sand particles remain before being sufficiently reduced to escape, and the higher the ratio of fine to coarse material in the mortar-box. Hence, under these conditions the liability is for the falling stamp to have its force of impact distributed over so large an area as to produce little effect but abrasion on a bed of shifting compressible sand. These considerations serve also to explain why fine breaking before milling with heavy stamps does little good. A large number of pieces of ore of approximately equal size afford a large area to receive the blow of relatively small pressure upon each. And further, the abrasive action, above referred to, explains why stamp milling with a fine screen and a high discharge, or both combined, is inefficient through the waste of energy in converting fine sand in to slime by abrasion of the particles re-arranging themselves under the stamp.[*]

As regards the desirable size to which ore should be broken before entering the mortar-box, the disadvantage of too fine crushing yielding a uniform bed has already been pointed out. The maximum limit varies with the class of ore, being less for hard ores, but in general it should be such that not more than one blow of a stamp is required to pulverise the largest piece of rock. Hence, the heavier the stamp, the coarser the preliminary breaking admissible, and *vice-versâ*. With unweathered banket ore probably a maximum diam. of $1\frac{3}{4}$ in. is permissible. Larger pieces are more economically reduced by the rock-breaker and the lowering of stamp duty, owing to less actual height of drop with such a feed, is thus avoided.

In passing it may be said that the practice of placing the breakers under the control of the mine captain, who when ore is coming up freely from below avoids congestion in his bins by opening out the breakers wide, is not to be commended. The object sought is certainly achieved, but at the expense of inefficient stamp milling and greater strains on stems when the edges of the shoes fall upon large pieces of hard ore.[†] When the pieces of ore are too large to enter the feed opening of the mortar-box, and feeding consequently ceases, the results are, of course, even worse.

Taking the average running weight of a stamp as with a half-worn shoe and with a stem of less than its original length owing to breakage, it will be found that this weight is some 10% less than the weight of the stamp when new, and that its duty is correspondingly reduced. The obvious remedy is the use, as the shoe wears down, of compensating weights in some form or other, which have been frequently suggested and occasionally used.[*]

Since, however, greater attention has been directed to the weight of stamps through the trials on the Knights Deep referred to, the use of compensating weights has become common on the Rand, and their importance in maintaining a high efficiency and stamp duty is so considerable as to be well worth the small additional trouble and expense involved by their use. Probably the earliest form of this device was placing an old head at the top of the stem or an extra tappet above the one in use.

These devices were, however, crude, and the writer has tried various other methods, including a false head intermediate between the true head and the shoe, on the ground that additional weight is better added near the bottom than near the top of the stem. Probably the most convenient compensating weights, however, are split cast-iron discs about 4 in. high and weighing about 50 lb. or 60 lb. each, which are clamped on the stem by means of two bolts either above or below the tappet. Such a compensating weight is illustrated in Fig. 4, and as many may be gradually added as are needed and can be accommodated.

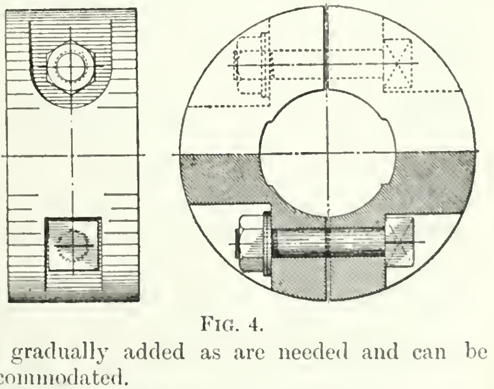


Fig. 4.

With increased weight of stamp, the question of increased shoe and die area arises, but the considerations already advanced, showing the small amount of pressure per sq. in. of total shoe area, indicate that very large shoe and die areas are not needed for hard ores, whilst for soft ore a lower, quicker drop can be used. Some increase, however, is usual owing to the fact that the larger diameter of stems and tappets required with heavier stamps necessitates increasing the distances between stamp centres.

---

[*] See E. A. Hersam's paper on "Economy of Power in Crushing Ore," published in the *Mining and Scientific Press*, of 16th Nov., 1907, p. 624; also C. de Kalb in *Mines and Minerals*, p. 136, Oct., 1906.

[†] See T. A. Rickard's "Stamp Milling of Gold Ores," p. 151.

[*] See Lock's "Gold Milling," p. 109; also *Journal of Chem. Met. and Min. Soc. of S A.* vol. ii, April, 1898, p. 209.

An increase is desirable when the total force of impact of the falling stamp requires a thick layer of ore on the die to avoid excessive shock. Such heavy feeding lessens the duty, because the thicker ore bed reduces the actual height of drop, and the blow is likewise cushioned so that the stamp crushes by abrasion rather than by impact. As already shown, the cushioning caused by fine particles in the mortar-box increases with the height of discharge and the the fineness of the screen.

The greater the force of impact per unit of area the coarser the screen required for maximum efficiency, as otherwise a thick layer of fine ore particles would be needed to avoid pounding. Since even a heavy stamp crushes but a couple of ounces of ore at each blow, and this weight would form but a very thin layer over the surface of the die, it is obvious that the least thickness of ore layer necessary to take the impact of the stamp without pounding is sufficient to supply ample material for pulverisation.

The prevention of irregular wear of dies is as yet an unsolved problem. At first sight it would appear that a slight cavity in the die surface due to soft metal would, when formed, be protected against further excavation by the surrounding portions of the die standing above it, but in practice no such protection seems to be afforded. The cavity grows steadily deeper with use, and even turning the dies regularly is not a cure.

In the writer's opinion the trouble is due to the larger pieces of ore, or pieces of steel such as drill ends, gravitating in the swirl of pulp in the mortar-box to the lowest point, that is into a slight depression originally due to softness of the metal, and thus the impact of the falling stamp is transmitted to gouge out the bottom of the cavity. Such irregular wear is especially liable to start at the back of the centre dies where the ore feed first lodges and is thus struck whilst at its largest size.[*]

Consequent upon the introduction of tube-mills, coarse battery screening has generally replaced the finer mesh, and thanks to the work of this Institution, and of the Chemical, Metallurgical and Mining Society of South Africa, more accurate descriptions are now applied to screening,[†] with the result that the millman can with some certainty obtain a duplication of a previous order.

In accordance with the demands of the modern millman armed with a micrometer gauge, manufacturers' catalogues, in place of giving vague trade terms, now detail diameters of aperture, wire and so forth, with a most laudable desire for extreme accuracy.

Since this development, however, the general introduction of secondary grinding by means of tube-mills has lessened the importance of the battery screen as a criterion of the degree of reduction of an ore, and, in fact, the grading analysis of the final pulp leaving the crushing plant as an overflow from the hydraulic classifiers is now the guide to the metallurgist.

As pointed out previously before this Institution, this change makes the degree to which a particle is reduced dependent on both its diameter and its specific gravity, instead of on the former only, with the result that the pyritic material undergoes that finer comminution and more perfect exposure of gold contents which its greater value renders desirable.[*]

Breakages of cam-shafts as well as stems and other battery parts are still commonly described as being due to "crystallisation" set up by vibrations incidental to the operation of crushing. At the present day, however, such an explanation cannot be accepted, it having been proved that repeated shocks in time develop, in such essentially crystalline aggregates as iron or steel, minute microscopic crevices which gradually extend until a fracture results.[†]

Among the few who have initiated improvements in stamp mill design of recent years is that experienced mill-man, M. P. Boss, whose design of a stamp with long head, thus giving strength of what would otherwise be a weak point in the stem, is worthy of serious investigation.[‡] It is somewhat remarkable that whilst the number of crushing machines invented to supersede stamps is endless, and much space in metallurgical publications is devoted to the description of crushing plants, yet the actual operation of the gravitation stamp and the discussion of its most appropriate design has been but little dealt with either by the skilled engineer or by the mathematician.

In order to employ heavy gravitation stamps to full advantage, it is absolutely necessary that a uniform speed should be maintained by proper governing of the engine. Only under these conditions can the maximum speed and height of drop be safely used without fear of camming; and the damage done in a few moments by sudden increased speed above the normal may prove a costly matter both in loss of time and material. Further, with heavy stamps the use of five-stamp cam-shafts is very desirable, in that as compared with ten or fifteen-stamp cam-shafts both the power to be transmitted and the number of shocks due to lifting stamps are reduced by one-half or two-thirds for each shaft.

---

[*] See *Journal Chem. Met. and Min. Soc. of S.A.*, vol. vii, 1907, pp. 216 and 293; also paper by M. P. Boss on "Crushing Ore" published in *The Mining and Scientific Press* of March 14th, 1901, p. 356.

[†] See *Journal Chem. Met. and Min. Soc. of S.A.*, June, 1906, vol. vi, p. 393; also October, 1907, vol. viii, p. 130.

[*] See *Trans. Inst. M. and M.*, vol. xiv, 1904, p. 55.

[†] See Rosenhain on "The Crystallisation of Iron and Steel" in *The Times Eng. Supplement*, p. 1, 6th Nov., 1907.

[‡] See *Mines and Minerals*, p. 209, Dec., 1908.

Many attempts have been made to supersede the use of gravitation stamps as the standard machine for the crushing of gold-bearing ore, but without success. That, in spite of certain obvious disadvantages, they still hold their own, is probably due to the fact that pneumatic and steam stamps as hitherto designed have proved wasteful of power and liable to derangement under the vibrations and shocks inevitable during crushing, whilst the gravitation stamp has little to go wrong, and power can be cheaply developed by a complex and economical steam-engine removed from unfavourable conditions and tended by a skilled operator without other duties. A serious effort is, however, now being made to utilise a new type of Holman pneumatic stamp at the New Kleinfontein mill, and there is no doubt that the energy and experience of those concerned in this test will, if it is possible, overcome such difficulties as have been encountered in the past with similar crushing appliances.

From the foregoing discussion it will be evident that the day of the stamp as a unit of crushing capacity has passed away. The unit of the present basis is the tonnage crushed per month or per day, and this is dependent on many factors —the relative proportion of stamps to tube-mills, the actual running weight of the stamps, the ratio of water feed, the height of discharge, the height and number of drops per minute, the screening used, and the maintenance of "concert pitch" in the mill engine.

The future limit of stamp weight is difficult to foretell, but it will probably be determined more by the mechanical considerations involved, as in the cam system of lifting, than by any decreased relative efficiency as a device for pulverising ore.

The advantages of heavy stamps, as compared with lighter ones, may be briefly stated as follows:—

1. Reduction of the initial capital expenditure in erecting, say, 200 stamps at 1,750 lb. with accessories, in place of 280 stamps at 1,250 lb. each.
2. Reduction in size of mill building, almost proportionate to the less number of stamps.
3. 30% less shafting, belts and other moving parts to maintain.
4. 30% less labour required for dressing plates, lubricating moving parts, changing screens, and other work incidental to milling operations.

In concluding this somewhat lengthy, yet incomplete discussion of some of the factors affecting present day stamp-milling, it is my pleasant duty to acknowledge my indebtedness to the Consolidated Goldfields of South Africa, whose foresight provided means whereby a departure was rendered possible which, besides benefitting the gold mining industry in general, has enabled them to claim "the heaviest stamps and the highest average stamp duties of any gold mining corporation in the world;"[*] and to Messrs. R. M. Catlin, H. H. Webb, and H. C. Behr and their staffs for greatly appreciated encouragement and suggestions during the progress of the experimental work detailed.

Obituary.

The death of Mr. WALTER HERBERT SMITH, a member of the Society, is recorded with regret.

Mr. SMITH, who was elected a member in June. 1907, was the Battery Manager of the Ayrshire Mine, Rhodesia, and had been suffering from fever for some time. On the journey to the Cape Colony he had a relapse in the train, and was removed to the Bulawayo hospital, where he died early in September.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

MANUFACTURE OF SODIUM PEROXIDE AND SODIUM CYANIDE.—"Two of the chief uses of metallic sodium (made electrolytically by the Castner process) are the manufacture of sodium peroxide and of sodium cyanide. Mr. C. F. Carrier, Jr., calls attention to the fact that the whole sodium recovered as peroxide or cyanide enters into the usual processes as metallic sodium; the reaction is in the first case $Na + 2O = NaO_2$, and the reaction of a certain cyanide process now in use is given by two equations $NH_3 + Na = NaNH_2 + H$ and $NaNH_2 + C = NaCN + H_2$. To reduce the amount of metallic sodium used as starting material, Mr. Carrier uses "a fused bath containing sodium salts, which will be decomposed by sodium, or sodium alloy, in such a manner as to render the combined sodium of the bath available as a source of sodium atoms for the end product; and add sodium, or sodium alloy, to this fused bath." This general principle may be made clearer by application in the two cases above cited.

If sodium be added to a mixed fusion of sodium hydroxide and sodium nitrate in the proper proportions, namely, approximately two parts by weight of the latter to three parts of the former, there are numerous possible reactions, the final state of which may be expressed by the following reactions:

$$NaNO_3 + 3\,NaOH + 8\,Na = 6\,Na_2O + NH_3 \qquad (I)$$

$$6\,Na_2O + 3\,O_2 = 6\,Na_2O_2 \qquad (II)$$

$$NaNO_3 + 3\,NaOH + 2\,Na = 3\,Na_2O_2 + NH_3 \qquad (III)$$

---

[*] See *Annual Report of the Consolidated Goldfields of South Africa, Ltd.*, 1908, p. 30.

Reactions I and III take place at the same time, the preponderance of either reaction depending upon the condition of the operation and the law of mass action. The actual amount of sodium peroxide produced will, therefore, lie between the results of reaction I+II and reaction III. This same principle may be applied to the production of sodium cyanide. If metallic sodium be added to a fused bath of NaOH in which carbon is suspended and NH_3 gas be bubbled through the melted mass at the same time, the following reaction takes place :

$$NaOH + Na + 2NH_3 + 3C = 2NaCN + CO + 7H.$$

Instead of employing pure metallic sodium, Carrier finds it advantageous to use lead-sodium alloy, as it may be produced easily by electrolysis of fused sodium chloride with a fused lead cathode (the old Acker process). But the simple addition of lead-sodium alloy in place of sodium to such a bath as specified above for making oxides of sodium is not practicable because lead would be oxidised as well as sodium and the sodium oxide would be contaminated with sodium plumbate. Likewise the alloy could not be used in the formation of sodium cyanide because the alloy produced in practice has not sufficient reducing power to give the desired result.

The author uses a double compartment cell, like that employed in the Ashcroft process, connection between the two compartments being made by fused lead. The "anode compartment" contains fused sodium chloride so that sodium is deposited onto and alloyed with the fused lead cathode. The sodium-lead alloy is anode in the "cathode compartment" filled with fused sodium hydroxide. By using an iron or nickel cathode in this compartment, sodium would enter into the fused bath from the fused lead alloy and metallic sodium would be deposited onto the cathode. That would be the Ashcroft process. The feature of Carrier's process is that if $NaNO_3$ is added, at intervals in amount proportional to the number of amperes used, the sodium of the lead-sodium alloy will be consumed as fast as liberated according to the following reactions :

$$xNaNO_3 + y\,NaOH + zNa = aNa_2O + bNa_2O_2 + cNH_3.$$

The relative values of the factors a, b, c and x, y, z, will depend upon the condition of the operation. This mixture of $Na_2O + Na_2O_2$ can be entirely converted into Na_2O_2 by solidifying, pulverizing and heating the powder to over 300° C. in air free from CO_2 and H_2O. The lead sodium-alloy, which has been partly depleted of its sodium, is continuously returned to the first compartment.

The production of cyanide of sodium is conducted in the same way, except that a different electrolyte must be used in the second compartment. In this case the fused NaOH holds carbon in suspension. The bath is electrolyzed as before and ammonia gas is bubbled through the electrolyte in place of the $NaNO_3$ used when sodium oxide was desired. The value of the process, if commercially successful, lies in the production of more of the final product than could be produced from the actual quantity of the metal entering into the reaction as metal or alloy only."—*Electro-Chemical and Metallurgical Industry*, March, 1909, p. 123. (A. McA. J.)

SEPARATION OF ANTIMONY AND TIN.—"A method is based upon the difference in solubility of the sulphides of antimony and tin in dilute hydrochloric acid. Thus antimony is quantitatively precipitated by hydrogen sulphide at the ordinary temperature from its solution in 15% hydrochloric acid, whilst tin is not precipitated under those conditions. In the case of mixtures, the precipitate of antimony sulphide obtained at the ordinary temperature invariably contains traces of tin, but if the precipitation be made at 50°-60° C. the precipitate will be free from tin. The solution of the mixed salts in 15% hydrochloric acid is heated to 50°-60° C. in the water-bath, and a current of hydrogen sulphide passed through it for 30 minutes, after which it is cooled to 30° C. and the current of gas continued for 10 minutes. The precipitate is then rapidly filtered off in a Gooch's crucible previously dried at 110° C., and washed with 15% hydrochloric acid saturated with hydrogen sulphide. It is next washed successively with water saturated with hydrogen sulphide, with alcohol, with a mixture of alcohol and carbon bisulphide, with carbon bisulphide alone, with alcohol, and with ether, and is finally dried at 110° C., and weighed. The filtrate containing the tin is partially neutralised with ammonia, diluted with water, and heated, and the tin precipitated with hydrogen sulphide. The results are independent of the proportion of the two metals in solution, and, as is shown by test analyses, are very accurate."—G. PANAJOTOW, *Ber.*, 1909, *42*, 1296-1299. — *Journal of the Society of Chemical Industry*, May 31, 1909, p. 545. (A. W.)

STANDARDISATION OF ACIDS AND ALKALIES.—"The author recommends the use of crystallised borax for the standardisation of mineral acids or alkalies, on account of its unalterable composition. When used for acids, with methyl orange as indicator, it acts like a known weight of sodium oxide and, therefore, represents a known amount of the acid. When mixed with glycerine, half of the boric acid is set free, which, with phenolphtalein as indicator, represents a definite quantity of alkali."—E. RUPP, *Chem. Zeit.*, 1907.—*Australian Mining and Engineering Review*, June 5, 1909, p. 298. (A. R.)

TEST FOR SCHEELITE.—"To test scheelite, place a portion of the powdered mineral the size of a pea in a test tube or a cup and cover with a teaspoonful of hydrochloric acid. Upon boiling a few minutes, or, if heated, allowing to stand a considerable time, tungsten trioxide (WO_3), a bright yellow powder, is formed. Upon the addition of metallic tin or zinc the yellow powder is reduced to a lower oxide, which is of a characteristic beautiful blue colour, and later to still lower oxides of other colours. With zinc the solution first turns blue, then wine coloured and, later, brown. When the reduction takes place very quickly the blue colour of the solution may not be seen at all, the first colour to appear being a reddish lilac, quickly turning to brown. With tin it turns successively blue, lilac, wine coloured and more slowly to brown. The wine colour may last for hours. Tinfoil, which is ordinarily composed largely of tin, with varying amounts of lead, may be used. In the reduction with tin the solution nearest the tin will often have a lilac or wine colour, while a short distance away the solution will be a fine blue. With either zinc or tin the powdered mineral itself will be coloured blue, even though the solution may be of a different colour. When treated with acid several minutes should be allowed to elapse before introducing zinc or tin, otherwise the yellow oxide may not be formed in sufficient amount to give a good reaction, and neither the blue nor the pink colour will be seen, but the solution will be a yellowish brown. However, a blue sediment will form in the bottom of the vessel. If one part of the hydrochloric acid diluted with two parts of water be used tin gives to the solution a blue colour, which lasts a considerable time. Sulphuric acid diluted with two or three parts of water to one of acid may be used instead of hydrochloric acid, and seems to be the more active of the two in changing scheelite to tungsten trioxide. It is slower than hydrochloric acid in reducing the yellow oxide to the blue oxide upon the addition of zinc and needs more heating, but gives the same blue colour, which is more lasting and may still show after standing 24 hours."—*Mining and Scientific Press*, May 27, 1909, p. 403. (K. L. G.)

METALLURGY.

Evolution of the Gravity Stamp Mill.—It is a remarkable circumstance that the ancients who were in many respects first-rate engineers should have overlooked the possibilities of a mechanical device for crushing gold ores. It cannot be that they were lacking in the necessary engineering skill, but rather that they could not see the advantage of combining a water wheel with the pestles already familiar to their workmen and thus save the labour and feed of many men. The sole reason must have been that while slaves were cheap machinery was not necessary, but when slave labour was no longer available or had become expensive, the crude gravity stamp mills were evolved from methods and appliances long known to the ancients.

The system of gold milling practiced in Egypt in ancient times will be described to show that the next step was easily attainable, and required no great amount of adaptability. The quotation is cited from Diodorus Siculus: "When the stone containing the gold is hard, the Egyptians soften it by the application of fire, and when it has been reduced to such a soft state that it yields to moderate labour, several thousand of these unfortunate people treat it with iron picks. Those who are above thirty years of age are employed to pound pieces of the stone of certain dimensions with iron pestles in stone mortars until reduced to the size of a bean. The whole is then transferred to women and old men who put it into mills arranged in a row, two or three persons being employed at the same mill, and it is pounded until reduced to a fine powder. (Here we have a stamp mill operated by three persons lifting a heavy pestle, presumably of iron and probably with a wooden stem.) At length the masters take the stone thus ground to powder and carry it away to undergo the "final process." The final process was "to wash on inclined tables furnished with two cisterns, all built of stone." Here we have what corresponds to a modern canvas plant. Evidently the amount of ore crushed was too great for slaves to pan, so it was run over tables with a stream of water.

Although we have no description of a gravity stamp mill being used by the Romans, it would have been only a short step in evolution to have used mechanical power to lift the heavy pestles used by the Egyptians. From the Hungarian mines there are remains that date from the fifteenth or sixteenth century. The tilt-hammer used in China has been described by Henry Louis.

From the sixteenth century to the time of the discovery of gold in California little improvement was made in the gravity stamp. It was a crude affair consisting of three or four stamps. Each stem consisted of a square wooden beam usually of oak, 12 ft. long and 6 in. square, the shoe of cast iron being inserted in the wooden stem and bound with iron bands and tightly wedged. The mortar was either open, or a wooden box, the bottom of which was composed of hard quartz fragments tightly beaten in with heavy iron rams. Sometimes the screens were of punched iron, but more often the discharge was simply over a side of the box which could be moved up and down to alter the fineness of the crushed pulp. The stamps weighed from 150 to 250 lb., and were dropped from 8 to 12 in. about 70 times per minute. The cams and tappets were of wood, and the stamps were lifted, without turning, in pairs when the battery was composed of four stamps, and alternately when of three stamps. Even the idea of five stamps and lifting in the sequence that gives the best flow of pulp is of late origin. The oldest mills in California had four stamps in a battery all on one cam shaft. Plumas county has many such mills. It is said that six stamps were sometimes put in a battery, and can be seen in Mexico to-day.

The modern stamp mills of the best type as used in different parts of the world are simply the refinement and perfection of these early types, made possible by the improvements in the materials of which the different parts are made. Our stamp now weighs from 1,000 to 1,700 or even 1,800 lb., and drops from 6 to 8 in. at the rate of 100 to 110 drops per minute.

It was the discovery of gold in California and Australia and the high cost of labour, that made necessary improved machinery for crushing ore, and from this necessity the old gravity mill was improved upon until we have the stamp mill in its present state of perfection. The original stamp mills as well as the mortars of the ancients were used simply to break ore that had already been subjected to a preliminary process of comminution. It is a curious fact that the stamp mill is gradually reverting to its original use as a crusher for rock of medium size.

When the cyanide process was unknown or in its infancy stamp mills were in general use as amalgamators, for it was found that a thorough admixture of mercury with gold in the mortar often saved more gold than when the processes of crushing and amalgamating were conducted separately. With the introduction of the cyanide process and the finer grinding of the ores, amalgamation has to a great extent been given up, so that the gravity stamp mill is gradually occupying the place of an intermediate crusher between the rock breaker and the fine grinder. As a coarse or fine crusher it is excelled by other machines, but as occupying the place described above in the process of winning gold from its ores it has no peer.

As the use of the cyanide process for the extraction of gold increases the purpose of the stamp mill is becoming simply a crushing machine. Its simplification from a battery of five stamps to that of one stamp is taking place at the same time, for it has been found that the single stamp crushes more ore per unit or per horsepower than when more stamps are arranged in the same battery. The amount of labour, however, increases with the single batteries, so that, especially where labour is high, a 5-stamp battery with a narrow mortar will in the end be more satisfactory than five single stamps. When amalgamation is necessary inside the battery a roomy 5-stamp battery is used and not the single unit.—Algernon Del Mar.—*Engineering and Mining Journal*, May 1, 1909, p. 890. (A.R.)

The Use of Standards in Reading Gold Pannings.—"In many cases one could not possibly obtain a correct estimate by panning.

He had made many careful experiments. He had taken 1 lb. of moderately fine crushed ore and assayed it very carefully, and then panned down several samples of the ore, and very carefully assayed the amount of gold left in the pan. He always found that he lost nearly half the gold in the panning. So that if they were to judge absolutely by panning he was afraid they would get slipped up very badly sometimes.

He quite understood that one could arrange to pan an ore which was known perfectly well, and could get a fairly correct estimate of that ore. But how could one go in different districts and pan strange ores, and imagine one could get a valuation of those ores which one did not know? One did not know how much was carried away in the slimes. That was where he thought one might go wrong.

With an ore which one knew well a fair result might be obtained, but not with a lot of strange ores in prospects."—A. F. Crosse, *Institution of Mining and Metallurgy, Bulletin No. 57*, June, 1909, p. 32 (H. A. W.)

MINING.

Mine Flushing.—"Mine flushing has been largely adopted in collieries in Germany. Sand, mixed with more or less dirt or clay, is used. This material is dumped on a grid at the mouth of the shaft, where it is thoroughly mixed with water. Cast-iron, wrought-iron, or steel piping, 13 ft. in section, and 6 to 8 in. long, is used for this work. The method as there used allows the filling of 90 per cent. of the volume of space, which so greatly reduces the strata pressure as to materially reduce the cost of timbering, and permit the seam to be completely worked out."—*Queensland Govt. Mining Journal*, Dec. 15, 1908. (C. B. S.)

Concrete for Mine Support.—"A paper on the use of concrete for mine support was read by Prof. W. R. Crane at the 50th general meeting of the Institution of Mining Engineers. The author said that up to comparatively recent times timber had been the principal means of support for underground working; but its use as an economical material for support was rapidly passing. The falling off in the use of timber had been augmented by the application of concrete, which was now recognised as a reliable and satisfactory substitute for timber. Concrete and reinforced concrete were now extensively employed in mining shafts, mine galleries, tunnels, gangways, in support of rooms and stations, in building dams and stoppings underground, and, in fact, in all mining work both within and without the mine. Concrete had the advantage that it could be applied equally well in shafts of any shape and size. Several rectangular concrete shaft linings had been successfully constructed in the United States, but practice favoured the oval and elliptical forms. Only about half the thickness of lining was necessary when curvilinear forms were employed. Many of the deep level shafts of the Lake Superior copper region were lined to the bed rock with concrete. Instances could be quoted of the use of a circular concrete drop-shaft in sinking through loose materials. The relining of timbered shafts with reinforced concrete had been successfully accomplished in a number of instances in Pennsylvania, one of the best illustrations of such work being that of Manville shaft, in the vicinity of Scranton. Stations for underground machinery such as pumps, engines, stables, offices, etc., were now, in many large mines, being lined or re-lined with concrete. Concrete in coal mine workings had an advantage over timber and stone in that, with a little care in surfacing, the resistance of air currents was materially reduced, which circumstance, together with the considerations of greater strength and durability, was rapidly extending its use." *Science and Art of Mining*, June 12, 1909, p. 505. (A. R.)

Temperature at Great Depths in Coal Mines.—"It has been known from early times that the temperature of the earth increases with the depth from the surface, and the text book theory was that, irrespective of what the strata may be, there is a certain rise of temperature for every certain number of feet that you may go down into the earth, commencing at a point below the surface at which the temperature was assumed to be uniformly constant. This point was said to be at a depth of 50 ft., and the rate of increase to be 1° F. for every 60 ft. of depth below that point. This theory has received important modification in recent years as the result of careful and extended observation, and it has been shown that the rate of increase varies considerably in different places, and depends upon the conductivity of the strata, the presence of water, the inclination of the strata, and other considerations.

Within the limited area of the United Kingdom the Royal Commission on Coal Supplies considered that a gradient of 1° in rather less than 64ft. was indicated by the evidence placed before them. This gradient would give an average nominal temperature approaching 110° F. at a depth of 4,000 ft. The following temperatures at different depths are among those which have been placed on record :—

	Depth. Feet.	Tem. Deg. F.
Rosebridge Colliery (Wigan) Arley Seam	2,381	93·5
Pendleton Colliery, Rams Seam	2,088	83
Pendleton Colliery, Lower Level	2,214	86
Moss Colliery (Ashton-under Lyne), Roger Seam ...	2,520	84
New Moss	3,300	86

The maximum temperature consistent with useful work, health, and reasonable comfort appears to be undetermined. But the necessary conditions are generally admitted to be brisk ventilation and dry air. Experience has shown that under these conditions a man can work the usual hours in a temperature as high as 100° F. without a noticeable diminution in his powers of endurance or in the amount of work performed. Professor Heim, referring to the construction of the Simplon Tunnel, says that as soon as the temperature of the air in which men worked reached 77° F. the energy of the work began to decrease, and at 86° F. it was generally only two-thirds, and at 95° F. it was no more than half the normal. Practically men cannot work long continuous hours at a temperature higher than 86° to 95° F. maximum. Every higher temperature must be diminished by ventilation to this point, better still to a temperature of 77° F. In Prussia the working time of persons employed in coal mines where the temperature exceeds 82·4° F. is uniformly limited by law to six hours a day.

Many experiments have been made in order to discover practicable means of reducing the temperature in the workings. As dryness of the air is a necessary condition, cooling the air by means of spraying is practically inadmissible. Where compressed air is used for working coal-cutting machines, the exhaust air has a considerable cooling effect in their immediate neighbourhood, and it is possible that in certain cases the use of compressed air might be extended. It would appear, however, that the best practicable means, according to present knowledge is increased ventilation proportional to the depth."—(Royal Commission on Coal Supplies), *Times Engineering Supplement*, June 16, 1909. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

PRINCIPLES OF MINING. By HERBERT C. HOOVER. Pp. 199, illustrated. 6×9 in. Cloth. $2.50. (New York and London, 1909: Hill Publishing Company.)

"Contents: Valuation of copper, gold, lead, silver, tin and zinc lode mines. Development of mines. Stoping. Methods of supporting excavation. Mechanical equipment. Ratio of output to mine. Administration. The amount of risk in mining investments. The character, training, and obligations of the mining engineering profession.

This book is appropriately entitled the 'Principles of Mining' because it deals with the fundamentals of the art from the economic standpoint, and that, of course, is the only reason why mining is prosecuted at all. The great treatises that discuss in detail the technics of the art concern themselves with the means how. Mr. Hoover treats broadly of the means how, but confines himself chiefly to the reasons why, and so far as we are aware he is the first author to take up that elemental phase of the subject. It has been done, of course, in a fragmentary way in contributions to the technical press and the proceedings of the societies, but no one previously has elaborated a treatise upon it. It is singularly fortunate that Mr. Hoover should be the first to elect to do so, because we can conceive of no one else so thoroughly competent He is so well known in the profession and in the industry at large that it is unnecessary for us to expand respecting his capacity. It is sufficient for us to say, and this merely for the information of the laymen, that he is not only eminent as an engineer, but also as manager of mining enterprises of the first magnitude in many parts of the world. The profession and everyone interested in the mining industry are to be congratulated that he has taken the trouble to write this book."—*The Engineering and Mining Journal*, June 5, 1909, p. 1149. (W. A. C.)

ELECTRO-MAGNETIC ORE SEPARATION. By C. GODFREY GUNTHER. Pp. 193, illustrated. Price, $3. For sale by *The Mining World.* (New York: Hill Publishing Co.)

"The author has gathered together published information on the subject, but includes only such as is of present commercial importance. The theoretical principles are dealt with, and the practical means by which the desired results are obtained, including descriptions of the commercially important magnetic separators, are presented. A large portion of the book is devoted to actual practice in the concentration of magnetic ores in different parts of the world, and to various special separations. The book is bound in cloth and well illustrated."—*The Mining World* (Chicago), June 5, 1909, p. 1081. (W. A. C.)

MILLWRIGHTING. By JAMES F. HOBART. 8vo., pp., 401. Illustrated. Index. Price $3. (New York, 1909: Hill Publishing Co.)

"The millwright is not a book-made man; his skill is the result of knowledge acquired in practice; his is a trade. Hence any millwright will know more than any other millwright will put into a book. But no two men's information is compressed within the same dimensions, and Mr. Hobart has given much that will help his brethren. He has done his work well, and we commend especially the chapters on Laying Out Shafting, Putting Pulleys in Place, Belts and Belting, and Steam and Water Pipe Fitting. The discussion of foundations, erection of structures, and strength of materials is too superficial to yield more than an occasional practical hint."—*Mining and Scientific Press*, June 5, 1909, p. 802. (W. A. C.)

PIONEERING. By FREDERIC SHELFORD, B.Sc. (Lond.), M.Inst.C.E., F.R.G.S., F.R.C.I. 3s. (London: E. and F. N. Spon, Limited.)

"'Pioneering' is a reprint of four articles which appeared in *Engineering* a short time back, now revised and published in book form. It is a handy little volume, invaluable to the beginner about to start on his travels for the first time, and also useful to the 'old stager' as serving to jog his memory when it again falls to his lot to have the fitting out of another expedition. Much sound advice is given, the outcome of personal experience. For instance, it is usually better 'to keep the cash box in a fairly prominent position, but to have only papers in it." Again, he 'saw a man once sit on a snake,' as to which we regret not being told what happened next. All and more than this, presented in brief language, add a charm to the book, which those who are interested can corroborate at their leisure."—*London Mining Journal*, June 5, 1909, p. 702. (A. R.)

ANTIMONY. By CHUNG YU WANG, M.A., B.Sc., etc. (London: Charles Griffin & Co., Ltd., Exeter Street, Strand, W.C.)

"The author remarks in his preface that a metallurgical work in English by a Chinese author is unusual, but, as he explains, long residence in English-speaking countries has familiarised him with the language, and certainly the work bears unimpeachable testimony to his knowledge of our tongue. In any case, no apology is necessary, for the author has added to the list of metallurgical works a book which stands alone, inasmuch as there is not, to our knowledge, any other complete treatise on antimony among all the English text-books. The author deals with the metal's history, chemistry, mineralogy, geology, metallurgy, uses, preparation, analysis, production, and valuation, and, moreover, treats these subjects in a very interesting way. Not the least instructive are the introductory chapters in which the history of the metal is given, and many of the quaint notions which have been held in the past concerning it are instanced. Following up the different phases to the modern treatment and application of antimony naturally often necessitates a statement of facts which are already recorded in other works. For instance, the application of antimony in modern alloys is not dealt with as fully as it is in some other books on alloys, but Mr. Wang has acceptably filled the many gaps that existed between the generally available information on the subject of antimony."—*Iron and Coal Trades Review*, July 9, 1909. (A. R.)

Clerk, Dugald. The Gas, Petrol and Oil Engine. Vol. I., New and Revised Edition. 8vo., pp. 392. *Longmans.* Net 12s. 6d.

Hall, A. D. Fertilisers in Manures. Cr. 8vo., xvi.-384. *J. Murray.* Net 5s.

Jones, A. Clement and Bloemfield, C. H. Elementary Mechanics of Solids and Fluids. Cr. 8vo., pp. 388. *E. Arnold.* With Answers. 4s. 6d.

Matthews, R. Borlase. Electricity for Everybody. A Popular Handbook dealing with the uses of Electricity in Home and Business. 12mo., pp. 324. *Electrical Press.* Net 5s.

Moor, C. G., and Partridge, W. Aids to the Analysis of Foods and Drugs. 3rd Edition. (Students' Aids Series.) 12mo. *Balliere.* Sd. 3s., net 3s. 6d.

Science in Modern Life. A Survey of Scientific Development, Discovery and Invention and their Relations to Human Progress and Industry. Prepared under the Editorship of J. R. Ainsworth Davies. Vol. III., pp. xi.-187. *Gresham Publishing Co.*

Willis, J. C. Agriculture in the Tropics. An Elementary Treatise. Illustrated. (Cambridge Biological Series.) 8vo., pp. 217. *Cambridge University Press.* 7s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 383/09. James Bowmaker Kerr Dodds. Improvements in mortar boxes for stamp mills. 21.8.09.

(C.) 384/09. Georges Schauli. Improvements in and connected with electric batteries. 23.8.09.

(P.) 386/09. John Acheson Jones. An improved tube mill liner. 23.8.09.

(P.) 387/09. Frank Leonard Whitmore. Improvements in skip indicators for winding or hauling plants. 24 8.09.

(P.) 388/09. Charles Rollin (1), The Hedworth Barium Co., Ltd. (2). Improvements in the manufacture of barium compounds. 24 8.09.

(P.) 389/09. Walter Henry Haisman. Detachable cam for gold mining batteries, 27.8 09.

(P.) 390/09. John Acheson Jones. A combined cigarette and match packet. 27.8.09.

(C.) 391/09. Carl Emil Egner (1), Johan Gunnar Holmstrom (2). Improvements in or relating to telephone transmitters. 27.8.09.

(C.) 392/09. Hammersley Heenan. Improvements in apparatus for cooling water or air. 27.8.09.

(C.) 393/09. William Watts Bonson. Dry ore separating machines. 27.8.09.

(C.) 395/09. Georg Ulbrich. Improvements relating to the separation of substances by wet magnetic process. 28.8.09.

(C.) 396/09. Georg Ulbrich. Process and device for dressing magnetic materials. 28.8 09.

(P.) 397/09. James Bowyer. Improvements in means for diverting the pulp discharged from a mortar box on to the amalgamated plate of an adjacent box. 28.8.09.

(P.) 398/09. Ernest Streich (1), Joseph Trelitz (2). Portable destillator constructed that the parts can be put into one another for transport. 28.8.09.

(P.) 399/09) James Connolly. Improvements in metal and ore smelting furnaces. 30.8.09.

(P.) 400/09. William Calder. Improvements relating to feed devices for tube mills. 1.9.09.

(P.) 401/09. James Miners Holman (1), John Leonard Holman (2). Improvements in lubricating devices for rock drilling machines. 1.9.09.

(P.) 402/09. Frederick Busching. Improvements in means of disposing of mine residues on dumps, and for other similar purposes. 2.9.09.

(C.) 404/09. Harry Edward Gresham (1), George Kiernan (2). Improvements in ejectors for vacuum brake apparatus. 3.9.09.

(C.) 405/09. Robert Carlyle Child. Improved centrifugal disintegrator. 3.9.09.

(P.) 406/09. Frederick Walter Cindel. Device for securing stamp heads and the like to stems. 3.9.09.

(C.) 407/09. Percival Thomas White (1), George Robertson (2). Tilting appliance for decanting oils and liquids. 4.9.09.

(C.) 408/09. Charles Irme Bastian. Improvements in the method of transforming electrical energy into heat energy. 4.9.09.

(P.) 409/09. Harry Gumpold. Improvements in explosive compositions. 4.9.09.

(C.) 410/09. John Turns. A compressed air cartridge for blasting and like purposes with air and water receiver. 7.9.09.

(P.) 411/09. Gustavins Adolphus Sheeley. Improvements in linings for tube mills and the like. 8.9.09.

(P.) 412/09. Harold Dunsterville Bain. Bains process of extraction of gold slimes from zinc (being plate zinc, zinc shavings or powdered zinc). 8.9.09.

(P.) 413/09. Charles Edward Hutton. Improvements in amalgamating apparatus. 9.9.09.

(P.) 414/09. Charles Edward Hutton. Improvements relating to mortar boxes and amalgamating tables. 9.9.09.

(P.) 415/09. William Pearce. Improvements in brackets or means for carrying footwall rolls in shafts and the like. 9.9.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BOSQUI, F. L., *l/o* San Francisco; c/o Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg.

DURHAM, B. W., *l/o* Cleveland; Bushtick Mines, Ltd., via Bulawayo.

DYER, S. C., *to* P. O. Box 19, Bulawayo.

GRUMITT, P. H., *to* P. O. Box 1677, Johannesburg.

HILDITCH, C. H., *to* 82, Fourth Street, Boksburg North.

JOHNSON, A. D., *l/o* Denver; Jessie Mine, per Private Bag, Bulawayo.

JONES, H. EWER, *l/o* Bulawayo; The Lodge, Culmington Road, Ealing, London, W.

MCMILLAN, Q. C., *l/o* Germiston; P. O. Box 3663, Johannesburg.

NICHOL, T. T., *l/o* Fordsburg; Springs Mines, Ltd., P. O. Box 54, Springs.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. OCTOBER, 1909. No. 4.

Proceedings
AT
Ordinary General Meeting, October 16, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, October 16th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

63 Members: Dr J. Moir, Messrs. F. F. Alexander, K. L. Graham, E. J. Laschinger, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, R. G. Bevington, A. F. Crosse, Prof. J. Yates, H. A. Adams, G. S. S. Anderson, F. W. Bentley, E. Blume, W. Broom, W. G. Brunton, J. Chilton, F. W. Cindel, C. A. Corder, J. B. K. Dodds, G. G. Ferris, E. A. Furner, J. H. Harris, B. J. Hastings, Dr. J. McC. Henderson, S. A. Herbert, C. B. Hilliard, H. Howard, A. J. Johnson, J. H. Johnson, J. A. Jones, T. W. Jordan, G. A. Lawson, H. Lea, R. Lindsay, R. MacGregor, J. W. Morris, M. T. Murray, S. Newton, F. B. Ogle, F. D. Phillips, W. S. V. Price, J. Pyles, O. D. Ross, H. A. Scarf, W. Sharp, S. Shlom, T. W. Simmons, J. A. Smith, R. Stokes, C. T. Sydenham, J. A. Taylor, W. A. C. Tayler, A. Thomas, K. Tonnesen, C. Toombs, F. W. Watson and L. J. Wilmoth.

13 Associates and Students: Messrs. J. Cronin, D. Dowling, D. W. Greig, W. J. R. Hunter, A. King, N. N. Newland, F. J. Pooler, C. A. Robinson, J. M. Robinson, S. A. Smith, G. G. Thomas, M. O. Tillard, and W. O. Williams.

14 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the September *Journal*, were confirmed.

NEW MEMBERS.

Messrs. R. G. Bevington and F. F. Alexander were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Anderson, Percy, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Surveyor.
Butler, William Mahony, Simmer Deep, Ltd., P. O. Box 50, Germiston. Cyanider.
Doyle, Michael Joseph, Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.
Hesom, William Wade, Sheba G. M. Co., Ltd., Eureka. Amalgamator.
Piele, William, P. O. Box 138, East Rand. Mining Engineer.
Walsh, Peter Arthur, P. O. Box 2473, Johannesburg. Engineer.
Williams, Walter Owen, Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Cyanide Manager. *(Transfer from Associate Roll.)*

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Barrett, Sidney Stephen, Sheba G. M. Co., Ltd., Eureka, Transvaal. Amalgamator.
Bridgers, Marcus Milton, Sheba G. M. Co., Ltd., Eureka, Transvaal. Ore Roaster.
Clarence, John Vaughan Gerald, Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Mining Engineer. *(Transfer from Student Roll.)*
Pasea, Arthur Edward, Sheba G. M. Co., Ltd., Eureka, Transvaal. Vanner Foreman.
Ramsden, Ernest, Sheba G. M. Co., Ltd., "Bellevue," Eureka, Transvaal. Cyanider.
Westwood, William, 4, Central Road, Fordsburg. Cyanider.
Wood, Sydney, Sheba G. M. Co., Ltd., Eureka, Transvaal. Amalgamator.

As Students.—

Bass, Kenneth Christian, Three Sisters Mine, Barberton. Assay Learner.
Craig, Arthur William, Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Learner.
Lotter, Reginald Godfrey, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanide Learner.
McCaffery, Cecil J., Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.
McNiel, William, Wolhuter G. M. Co., Ltd., P. O. Box 1160, Johannesburg. Cyanide Learner.
Robinson, John Martyn, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube Mill Learner.

General Business.

THE SOCIETY'S JOURNAL, VOLUME IX.

The Secretary announced that the binding covers for Vol. IX. of the *Journal* were expected that week, and that members who desired to have their copies bound should send them in to him as soon as possible. The price of binding covers was 3s. 6d., but if the *Journal* were sent in to be bound, the inclusive price would be 6s. per volume, and 1s. extra if necessary to post the volume to members on completion.

He had received from a member in Rhodesia, the following :—

A SUGGESTION RELATING TO THE VALUING OF GOLD MINES.

Mr. R. A. Wade *(Member)*: In order to simplify all calculations of the number of tons of ore in blocks of ground, and also take the dip of the reef into account as affecting the payability of a block of ground, I wish to suggest the measurement of reef widths vertically instead of at right angles to the dip. By this method to calculate the tonnage in a block of ground the horizontal area × the vertical width of the reef is a simpler and more accurate method than the one at present in use. By this method the steeper the stope the greater its width, the assay value remaining the same as before. The question whether a stope is payable or not certainly is affected by its dip in some cases, but this is not considered in the present method of valuing a mine.

LEAVE OF ABSENCE.

The President : I have been fortunate in securing leave of absence, and the Council has given me permission to hand over the affairs of the Society during that time to our Vice-Presidents. I may say that Dr. Moir will take the chair at the next meeting, Mr. Dowling at the meeting following, and Mr. Saner at the third meeting.

THE BARBERTON GOLDFIELD.

By A. Richardson, M.I.M.M. (Member of Council).

Geography.—Barberton is situated at the south-eastern extremity of the Kaap Valley at the foot of the Makongwa mountains, which form the northern boundary of Swaziland, and is by rail 283 miles east of Pretoria and 136 miles west of Lourenço Marques, being the terminus of a short branch from the main line at Kaapmuiden. The Kaap Valley is a circular basin with a diameter of roughly 30 miles, the floor of the basin having a mean altitude of 2,500 ft. above sea level, whilst the almost unbroken rim of mountains rises abruptly in places 3,000 ft. above it. The valley is drained by three rivers, the North Kaap, South Kaap and Queen's, the two last soon uniting and joining the first near Noordkaap Station, proceeding on from there as the Queen's River.

History.—The occurrence of alluvial gold in the Kaap Valley was first reported in 1875, and in 1882 several hundred diggers were at work on the Kantoor and in the creeks in the vicinity, whose numbers, owing to rich finds, rapidly increased until in 1884 there were 2,000 diggers scattered over the De Kaap district, finding gold everywhere and content to leave reef possibilities disregarded. However, some of the more enlightened began to prosecute a search for reefs, with the result that one of the brothers Barber discovered one at the back of what is now the town bearing his name, or as the then Mining Commissioner describes it : "So in February, 1884, in the presence of several diggers, I broke a bottle of gin—champagne being unobtainable—on the rock containing the gold-bearing quartz and named the prospective township Barberton after the discoverers of the reef."[*] The Umvoti Reef was found the next day, and later on followed the discovery by Edwin Bray of what was truly named "The Golden Quarry," the outcrop of the Sheba Reef. The Sheba Gold Mining Company was floated in 1886 with a nominal capital of £15,000 in £1 shares, and the first crushings returned 8 oz. over the plates, the tailings going another 4 oz.[†] On the strength of this the shares rose to £120, and anything in the shape of a reef in the district was readily floated by company promoters for capitals varying from £10,000 to £500,000, and shares in many of these "mines," with the grass roots still undisturbed, ran up to twenty or thirty times their nominal value. Disillusionment quickly followed, and the population of the town, which had risen from 1,000 in 1885 to 10,000 in 1886, rapidly dwindled again, many going to the Rand, then just discovered. Although the history of Barberton district may be said to have begun with the diggers of 1882 there were many evidences discovered by them, such as the remains of properly graded roads, mining works, and old smelting pits containing small quantities of smelted gold, suggesting an Arabic civilization of pre-Mahommedan date, and also old workings probably of kafir origin of a few generations back.[†] It is highly probable that the operations of some earlier race, of which we have only the faintest records to-day, may in some measure account for the comparative poverty of modern South Africa in alluvial auriferous deposits.

---

[*] David Mackay Wilson. "Behind the Scenes in the Transvaal."

[†] T. Rennert "Diamonds and Gold in South Africa."

Geology.—The district lies in what is known as the Swaziland System, a system that extends from the Pietersburg district round the eastern Transvaal to the north of Vryheid, and over much of Rhodesia and Manicaland. It consists of intrusive biotite-granite, schists, slates, shales, quartzites, sandstones and conglomerates, all traversed by numerous diabase dykes. The basin of the Kaap Valley is made up of this granite, while the upturned edges consist of the other rocks mentioned, the schists being next the granite. All these sedimentary and metamorphic rocks dip at steep angles away from the granite, and have a strike tangential to its periphery. The numerous long serrated ridges which break up the floor of the valley are composed of diabase, the most prominent instance forming the Devil's Knuckles. On the west the series is overlaid uncomformably by the flat-lying Kantoor sandstones of the Black Reef series and the lower beds of the Dolomite Series. The strata on the north and south of the valley join together on the east and swing sharply off in that direction to the vicinity of Komati Poort; this bend to the east takes place just west of the Sheba.

The succession of strata is well exemplified along the bridle path from Barberton to Pigg's Peak, and has been worked out by A. L. Hall,[*] from whose account the following description is taken. Leaving the granite the lower slopes above Barberton begin with vertical, soft, probably igneous, schists, containing a thin calcareous band; then follow very hard highly sheared green quartzites; these and grey or pale red phyllitic beds extend as far as the blockhouse, and after them come softer quartzites to the summit of the first and highest range. The higher portions of this range are formed of dark-coloured quartzites with coarse conglomerates made up of cherty quartzitic pebbles; after these come soft reddish ferruginous shaley beds to the Lomati River, beyond which the second range consists of massive quartzites containing pebbly bands, near the Little Lomati River. The ascent to the "Zig Zag" traverses altered shales and schistose quartzites which change into arenaceous rocks right up to "Heights," about half way to Pigg's Peak. Just beyond "Heights" a small granite intrusion comes in, then a conspicuous banded chert bar, followed by banded ironstones with red jasper bands, then coarse conglomerate at the Devil's Bridge; from here to the hill overlooking Pigg's Peak greyish quartzite with bands of sheared green quartzite predominates, then banded ironstone and then a ridge of sheared conglomerates and sheared grits. This succession of strata occupies some 25 miles; the dip is to the south-east, and varies from vertical at each end to 70° in the central portions. Mr. Hall mentions a calcareous band, which is noteworthy, as the formation generally is characterised by the great preponderance of silica and the almost entire absence of lime.

A portion of the field, on Moodies, lying about eight miles west of the town has been the scene from time to time of considerable mining activity, and a section, from north to south, across the formation at this point is here given. Fig. I. (see p. 124). After leaving the granite of the Kaap Valley, schists and shales occupying a width of $2\frac{1}{2}$ miles are traversed; these dip 87 deg. south and form the foot hills of the range rising to an elevation of 4,500 ft. above sea level. The talcose argillaceous schists which make up the greater part of this series contain some narrow beds of chloritic schist, and are separated from the overlying shales and slates by a massive quartzite bar, in places 300 yards wide, which can be traced through the country for many miles. The shales and slates have a width of some 600 yards and a vertical dip. Following on this is a belt of quartzitic sandstone about 900 yards wide, bounded on the south by a mass of conglomerates and hard ribbon quartzites some 600 to 900 yards wide, which forms the backbone of the range, and rises not far west of the point of section to an extreme elevation of 5,800 ft above sea level. This ribbon quartzite is the magnetite-quartzite-slate or "calico-rock," a very characteristic feature of the Swaziland system. South of this is another belt of talcose argillaceous schist, which is the last of the series in which gold has been found in payable quantities, in fact one may say that the succeeding series of sandstones, conglomerates and quartzites, which cover a distance of some four or five miles, are barren.

The granite is seamed with veins of hard white glassy quartz which are rarely auriferous, and though some gold has been taken out of them they may be classified as gash veins of very little economic importance. In the first belt of schists there are several interbedded gold-bearing quartz reefs varying in thickness from a few inches up to several feet, and in colour from pure white through all the shades of grey to black. The gold occurs in irregular patches and lenses more rarely in well defined shoots with a regular dip. This irregularity is characteristic of the district, and adds considerably to the element of chance with which gold mining is there associated. Many of these reefs may be traced for miles, but they only carry gold in short sections, rarely more than 200 ft. in

---

[*] The Geology of Mount Maré, near Pietersburg, and its connection with that of Moodies, near Barberton. Trans. *Geological Society of South Africa*, xii., January to June, 1909.

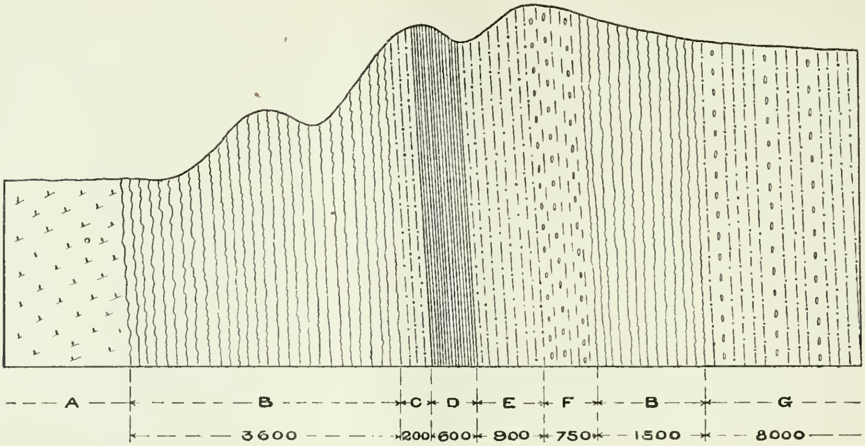


Fig. I.—Section across Moodies.

A.—Granite.
B.—Schists.
C.—Quartzite.
D.—Shales and Slates.
E.—Quartzitic Sandstone.
F.—Conglomerates and Quartzites.
G.—Sandstones, Conglomerates and Quartzites.

Figures show widths in yards.

length, so that the reef may be sampled over a considerable length and yet the rich portions missed. This restriction seriously affects development costs, as the poorer portions between the shoots may not be worth milling. The Pioneer Reef is a white, opaque, granular quartz reef bedded in chloritic schist; it varies in width from a mere parting to 6 ft., and is of a very free-milling nature. A noticeable feature of this reef is that where the country is highly metamorphosed the reef splits up into stringers and rarely carries any values. In the shales next to the quartzite bar the principal gold carrier is the Ivy Reef, a dark grey to black quartz reef with the gold occurring in well defined shoots; the width varies from a few inches to two feet. This reef has been exploited to a depth of 900 ft. below the outcrop, and the shoots are found to maintain their regularity and value. The reefs in the next belt consist of veins of quartz in quartzite, which passes by insensible gradations into soft sandstone; they are never of any size, and have not, in this neighbourhood, been worked to a greater depth than 400 ft. below the outcrop: where, however, a mass of stringers has been found in the sandstone the country rock has carried high values and stopes 15 ft. wide with ore going an ounce to the ton have been taken out. These reefs follow the planes of stratification in the quartzite, and although cross leads are found they are not numerous and only occasionally show local enrichment. The belt of conglomerates and ribbon quartzites which lies to the south of this series nowhere in this section carries gold in payable quantities. From traces up to 2 dwt. have been found in the conglomerate, but nothing to tempt anyone to do any work on it. The first mass of conglomerate is in places 200 ft. thick and composed of large irregular pebbles of chert, jasper, &c., in a coarse matrix. In the second belt of talcose schist, which is about 1,500 yards wide, only one reef of any importance has been found: this is known as the Brighton reef, and has had very little work done on it. It is a reef of grey quartz running from a foot up to several feet in thickness, and carrying from a few dwts. to 3 or 4 ozs. Portions of it are free-milling, but these often give place to large black ferruginous pockets several feet wide and carrying low values.

All this series from the southern schist belt to the granite have been subjected to the intrusion of numerous diabase dykes, and it is noteworthy that these are absent from the barren series of sandstones, quartzites, and conglomerates lying to the south of the second belt of schists. These dykes have a general S.E.—N.W. strike, a vertical dip, and are sometimes as much as 150 ft. thick.

There is a peculiar occurrence of alluvial gold near here, at the head of the valley, above the office of Moodies G. M. & E. Co. The valley terminates in a neck, and just below this there is a diabase intrusion, the result being a shallow basin, roughly, 1,000 ft. in diameter, the bottom of which is filled with a black clay varying in depth from nothing at the sides to 12 ft. in the deepest part. The gold occurs on the bed rock below this clay, and the closest search all over this small area has failed to reveal any source from which the gold could have been derived. A few miles to the west of this section is the Mount Morgan mine, noted throughout the district for the refractory nature of its ore. The reef here consists of a body of dark quartz stringers generally in black shale, sometimes in grey quartzite. The ore is highly pyritic, containing iron and arsenical pyrites and disseminated graphite.

To the east of the section is the Rosetta Sheba, which is now being actively developed: this, from north to south, shows the following succession :—Shale and schist, white reef, 2 ft. to 17 ft. thick, chloritic schist 20 ft. to 40 ft. thick with white quartz stringers of no value, blue reef 15 in. to about 60 ft. wide, then shale and schist again. The White reef contains copper salts and galena, and the Blue reef bands of galena, copper salts and graphite in films, pockets, and grains, but chiefly on the south wall, in veins from 1 in. to 1 ft. thick. These reefs dip with the strata 80° south. Just to the west of the town lie the Moodies Fortuna and Abbott's, the former working a free-milling grey quartz containing some magnetite, and the latter a white quartz which at 50 ft. from surface shows, in order of importance, copper pyrites, limonite, copper glance, and copper carbonate.

At the back of the Market Square is the Thornton-Little working a reef 2 ft. or 3 ft. wide consisting of ramifications of black quartz in quartzite somewhat resembling that of the Sheba in structure. To the east are the Elephant's Kloof, Kidson, and several others, and beyond them the famous Sheba, 12 miles away. As the Sheba has been milling longer than any other mine in South Africa, and as it is undoubtedly one of the most interesting mines in the world, it merits a somewhat extended description. The accompanying cross-section, Fig. 2, explains the formation and the relative positions of the ore bodies. It will be noticed that there are three beds of quartzite known respectively as the Sheba Bar, the Southern Cross Bar, or Zwaartkopje North Bar, and the Zwaartkopje Bar, and all the rich deposits of the mine are associated with one or other of these quartzite bars: it is also noteworthy that all the reefs lie in a zone defined by a series of diabase dykes crossing the strata in a general N.N.E.—S.S.W. direction. Starting at the north of the section, there are in the shales, slates, and schist, a series of mineralised fracture planes striking in no particular direction and of low grade and erratic nature. These are represented by the Eureka, Mamba and Margaret reefs. The Sheba reef is a crushed or sheared quartzite, the planes of fracture being recemented with secondary white and black quartz, the black being the gold carrier while the white is barren.

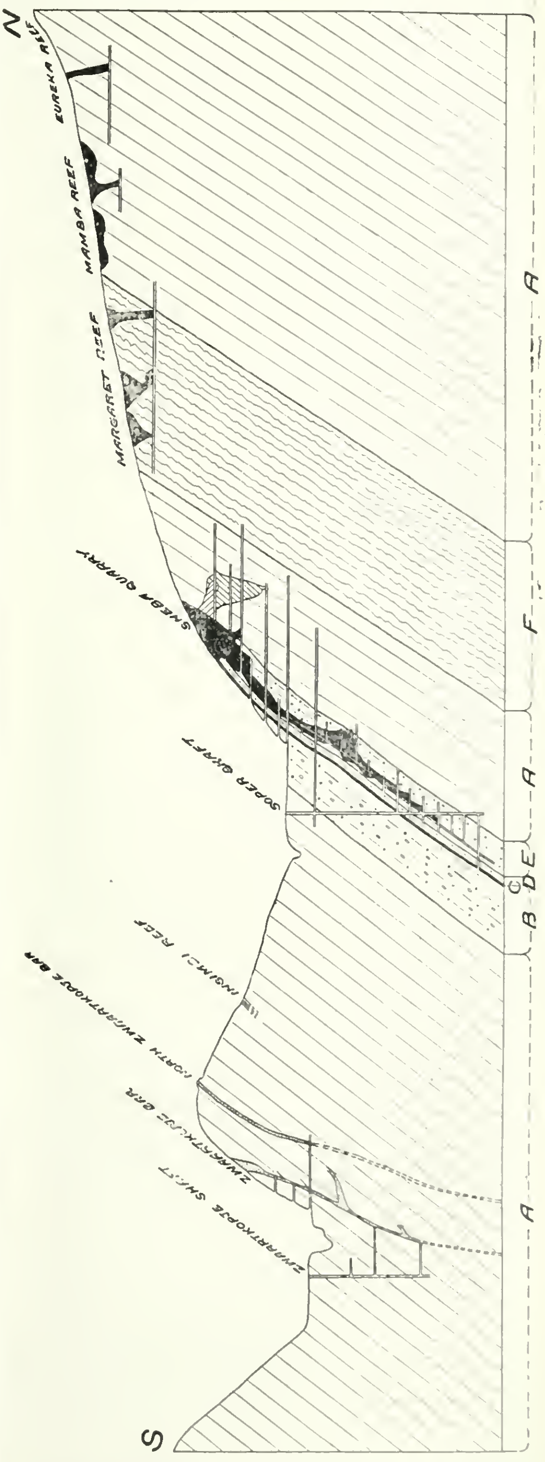


Fig. 2.—Section across the Sheba Mine. Scale 1 in. = 920 ft.

A.—Shales and Slates.
B.—Sandstones and Conglomerates.
C.—Clay Seam.
D.—Sheba Bar, Quartzite.
E.—Sheba Reef, Quartzite.
F.—Sandy Schist.

The dark colour of this quartz has been attributed to the presence of bituminous particles,* but a more probable explanation is that of total internal reflection from strain surfaces. The presence of quartz of two different colours, one rich in gold and the other barren, is due probably to there being two periods of movement and silicious deposition. In the first the auriferous quartz was deposited, and in the second the newer barren white quartz, while at the same time the auriferous quartz was subjected to considerable pressure, leading to the development in it of a multitude of strain surfaces. A similar phenomenon has been observed in the quartz of the Dharwarian series of the Kolar field and the Hutti mines of India, a series the geological features of which closely resemble those of the Barberton formation.* This quartzite of the Sheba reef is continuous on either side but the gold is only in the shoot. The stoping width reaches as much as 100 ft., but is generally about 30, while the length along the strike below the quarry varies from 250 ft. at the upper to 110 ft. at the lower levels. This shoot has been worked for a distance of over 1,300 ft. on a dip of 55° to the 16th level, below which it is unpayable. The Insimbi reef is a quartz body in a fissure striking to the north-west from the N. Z. K. Bar through the shales and slates. The Zwaartkopje ore body is a quartz spur from the footwall of the Z. K. Bar, and though rich at the bar the values gradually disappear as it penetrates into the underlying strata. North of the Sheba there are numerous cross veins, some of which form rich cross leads in Thomas' bar as at Joe's Luck mine. This is one of the few instances where these cross veins have proved payable.

In the North Kaap and Jamestown districts there are several interesting deposits. The Verdite is working a lenticular reef of steatite which is 14 ft. wide in one place, the hanging-wall being diabase and the footwall talc schist: this reef contains some arsenic, antimony, cobalt and nickel, becoming refractory in places. Its dip is 80° and it strikes with the formation. The Verdite gold mine is several miles from the original discovery of Verdite stone, the name being kept on when the present gold claims were acquired. Verdite stone is one of the monoclinic micas, Fuchsite, its beautiful green colour being due to the presence of oxide of chromium to the extent of from 1-4%. The chief mine in this locality, or at the present time in the whole district, is the Worcester. Here the country rock enclosing the reef is composed of chloritic schist and dips 70° in a N. E. direction. The principal ore body in the mine (see Fig. 3) has been opened up on the 500 ft. level over a length of 850 ft., carrying payable values over practically the whole distance, but not over the full width at all points; the width of the reef at the main shaft on this level is 40 ft., and this gradually narrows down to the S.E., until at a distance of 250 ft. from the shaft it is only 6 in. wide. The width at 150 ft. N.W. of the shaft is 60 ft., and beyond this point at 250 ft. the body splits up, stoping being done on the hanging-wall portion only, the footwall portion being unpayable. The width of the reef at the face of the N. W. drive, which is still being driven, is 20 ft. Although payable rock is found throughout the whole length of the body rich shoots occur extending over a length of 100 to 200 ft. and dipping in a N. W. direction.

The following two typical examples show the distribution of values from wall to wall:—

Footwall.—2′ @ 2·0, 2′ @ 3·0, 2′ @ 1·5, 2′ @ 25·0, 2′ @ 4·5, 2′ @ 1·5, 2′ @ 2·5, 2′ @ 18·5, 2′ @ 22·0, 2′ @ 3·5, 2′ @ 9·0, 2′ @ 13·5, 2′ @ 2·0. Hanging-wall.

Footwall.—2′ @ 10·5, 2′ @ 12·0, 2′ @ 16·0, 2′ @ 4·5, 2′ @ 3·0, 2′ @ 3·0, 2′ @ 26·0, 2′ @ 13·0, 2′ @ 1·5, 2′ @ 7·0, 2′ @ 10·0, 2′ @ 3·5, 2′ @ 18·0. Hanging-wall.

About 30 miles east of Barberton, at Louw's Creek, is the Three Sisters mine, one of the most beautifully situated as well as one of the most

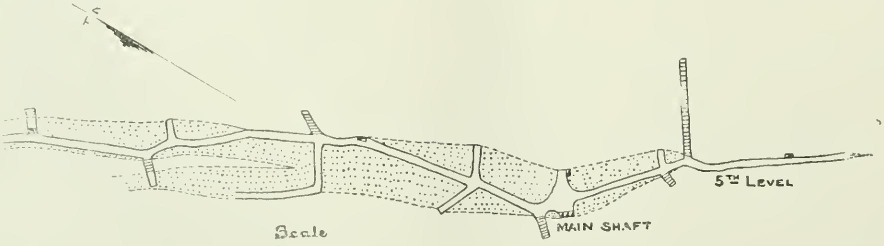


FIG. III.—Sectional Plan of Ore Body at the 500 ft. Level of the Worcester Mine.

---

* [illegible] Bordeaux. Études sur les Champs Aurifères de Lydenburg, De Kaap et du Charterland, *Annales des Mines*, vol. xi., 18[illegible]

* [illegible]olm Maclaren. Gold: Its Geological Occurrence and Geographical Distribution.

interesting geologically. This mine takes its name from three bold conglomerate mountains which hem it in on one side of the creek, and from the top of any one of which magnificent panoramic views can be obtained. This conglomerate, which has received a very large development at this point, on both sides of the valley, is a highly indurated schistose rock showing few signs of weathering even on the highest summits: the matrix is a coarse grit and the pebbles, which are small, are so severely flattened and crushed as to give it the appearance of a breccia. The reefs at present worked are in a foothill which butts up against the mountains at the head of the valley forming a tongue running down into it, the axis of the hill being roughly N.E.—S.W. A section across it from the northern side gives first green slate dipping 80° S. E., then talcose schist, and finally green slate again. The northern slate contains a reef of which a section is shewn in Fig. 4. This reef is a quartz schist with numerous quartz laminæ (several to the inch), sericite partings and scattered iron pyrites. The talcose schist contains several white quartz reefs and leaders. The reefs, which vary in thickness from a few inches to 16 ft., dip generally 80° S. E., conformably with the foliations of the schist. These reefs contain chlorite, calcium and magnesium carbonate, some silicate, and a little strontium. Crossing these at fairly regular intervals are white quartz leaders 1 in. to 8 in. thick, whose strike is slightly inclined to that of the reefs. These leaders dip 45° N. W. across the formation, and have a peculiar wavy appearance in cross section, the distance from wave to wave being 18 ins. and from trough to crest 12 ins. as a general average; where they cut through the white quartz reefs there is no disturbance or faulting and no local enrichment. In the southern slate is a wide reef consisting of grey quartz with iron pyrites in numerous thin bands which give a lamellar appearance to the rock. These reefs in the slate are very pyritic and contain a high percentage of silver. The white quartz reefs are free-milling and the leaders particularly so, the bullion obtained from them being very pure.

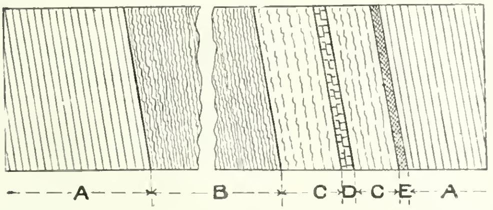


Fig. IV.—Section across N. Slate Reef, Three Sisters Mine.

A.—Country Rock of Green Slate.
B.—Chloritic Schist, 30 ft.
C.—C. Quartz Schist, 2½ ft.
D.—Honeycombed Quartz, 3 in.
E.—Limonite, 2 in.

This slight geological sketch would not be complete without some reference to the mines of Barrett's Berlin and Coetzeestroom on the Kantoor, near Kaapsche Hoop. These are quasi-reef deposits consisting of irregular quartz veins, bunches and stringers, in a soft rock which is the result of the decomposition of the lower beds of dolomite. Sometimes the quartz carries the gold and sometimes the rock above and below to the extent of a few feet. The workings of the former mine are 30 feet above the Kantoor sandstone, on which the dolomite rests, and the workings of the latter are 80 above the same horizon.[*] Alluvial is still worked to a slight extent on the Kantoor in and a few feet under the surface; this gold originated probably from the disintegration of the lower dolomite beds. The "President" nugget, weighing 187 oz., was found here in 1887 six inches below the surface. Generally speaking the alluvial either here, on Moodies, or at Jamestown, only holds out possibilities of wages; the ancient race that worked this alluvial gold probably skimmed most of the cream off, and the diggers of the 80's cleaned up what was left.

Prospecting.—Although most of the properties on which development was done in the early days are now held or being worked by tributors and small syndicates, the district has by no means been adequately prospected, difficulties of transport standing much in the way. During the summer months the grass in the low-lying parts would interfere considerably, as also probably would the rain and the hot relaxing climate, but in the winter months the climate is a perfect one, and the grass fires lay the formation bare. Being a mountainous district with extensive foothills and ravines good sections and exposures can generally be got, and although plenty of chipping and scratching has been done everywhere there is still scope for the man with a good eye for a reef. In prospecting, as indeed in much other work, the ignoring of the obvious has led to the disregard of many significant indications. This has been exemplified in all mining countries, so a few Barberton illustrations will suffice. In the Sheba section (see Fig. 2) the rich portion of the outcrop of the Zwaartkopje Bar, which when found showed nuggety gold freely and stood out above the surrounding surface, and from which a lot of 100 tons went 23 oz. to the ton, lay for nearly ten years with the Sheba battery crushing close to it just over the other side of the hill. Another case was that of a reef discovered on the Swaziland bridle path close to Barberton in 1896. This reef, which crosses the path at right angles, owing to its superior weathering qualities formed an awkward step for horses and a bit of a scramble for human beings, and yet it was ten years before anyone observed that where it crossed the path it carried "visible" freely. Another case was the finding within the last two years of a reef within a few hundred yards of two established mines and an hotel, which from time to time had been the point of call of numerous prospectors. This was discovered by its present owner who while in the course of a Sunday afternoon stroll stopped to admire the scenery, a commendable proceeding in this hilly country, and kicked a projecting knob of quartz off and found it to contain visible gold. One good way of prospecting is to follow up a road party. A reef was discovered recently in this way from which a trial consignment of about 1,500 lbs. yielded 125·35 oz. per ton.

---

[*] A. L. Hall, Transvaal Mines Department: Report of the Geological Survey for the year 1905.

To acquire claims the simplest way is first to consult the Government maps, which are compiled by the Surveyor-General and the Register of Mining Rights: these can be bought, the price ranging from 3s. 6d. to 10s. 6d., or can be consulted at any Government Mining Office; then to find out whether the particular piece of ground is open for pegging or not consult the compilation plan in the Beacon Inspector's office. The next thing is to expend 5s. in a prospectors' permit, which enables one to peg an area not exceeding 2,000 ft. by 2,000 ft.: having pegged the area there is no further fee and three months grace is allowed in which to start work. The work done must be sufficient to satisfy the Mining Commissioner, but exactly what constitutes "work" in this connection has never yet been clearly defined. An old prospector told the writer that two boys kept fossicking and visited once a week would probably meet the case, but some official statement on this point is clearly needed. In the Canadian North-West a somewhat similar performance costs $5, and $100 has to be either paid or expended each year; when $500 has been expended or paid the locator may, on having his ground surveyed and upon complying with other requirements, purchase the land at $1 per acre, paying a royalty of 2½% on sales.

If claims are held under the ordinary 2s. 6d. per claim per month system then it is worth noting that 15 claims, or under, can be exempted from any licence if worked by the owner personally, and to the satisfaction of the Mining Commissioner. If the prospector has any doubt as to the correct procedure in any particular case he need have no hesitation whatever in consulting the Government officials, as they will be only too pleased to put him right. There are plenty of amateur Gold Law experts in the district, but it is much the quickest and cheapest way to go straight to the fountain head.

As the name Moodies often occurs in discussing mining matters relating to Barberton, some information regarding it may be of service: the company publishes no printed schedule, but the following, which is subject to the provisions of the Gold Law, may be taken as fairly correct. Moodies G. M. & E. Co. has full mineral rights over two farms, covering 7,487 morgen and three mynpachts. Since the advent of the small syndicate, terms have been altered and each individual case is treated on its merits. The usual working basis in the case of prospectors is to charge £1 per month for an option over an area equal to a claim area of from 6 to 12 claims. The ground has to be continuously prospected to the Company's satisfaction and the period of option is for six months; this is sometimes, if reasonable cause can be shown, extended for a further period of six months. At the end of this period the prospector is called upon to peg such ground as he requires and pay £1 per claim per month. With ground on which development work has already been done, no cash payment is asked for such work, but an additional 2½% royalty on gold won is charged for all *useful* development, based on a valuation of 10s. per foot run. Here again in case the prospector decides to put up a mill to work certain of his claims but is uncertain about the remainder, the Company may protect these for him for a definite period, so that he may reap the full benefit of his work. A royalty of 2½% is charged on the gross value of the gold won. A mill site is granted with the right to use water for reduction purposes, a charge of £5 per annum being made for this: the right to use water for power is also given, subject to the risk of curtailment, should it be required for reduction purposes. A charge for firewood at the rate of 5s. per month per white habitation is made. Timber, if required, is supplied by the Company at the rate of 1s. per stick 8 ft. long × 6 in. at butt, other sizes *pro rata*: these have to be cut in the Company's gum plantations and transported by the user at his own cost: the timber in the natural bush is not allowed to be cut in order to conserve the water supply as much as possible. Electric power is supplied at a charge of £30 per b.h.p. on motor shaft, the Company erecting and maintaining the high tension side of the plant up to and including transformers; the consumer supplying and maintaining low tension leads, motor switchboard, etc. In the case of a consumer of a small block of power a guarantee has to be given that power will be used for a definite period, long enough to cover the outlay involved in the erection of a branch line to his plant. The Company's plant runs 365 days in the year and power has to be used continuously, no allowance being made for a breakdown, or stoppage, on the low tension side of plant.

Mining.—The mountainous nature of the country, furrowed by deep ravines, enables the reefs to be developed by means of adit levels driven on or to them at considerable depths below the outcrop, which obviates hoisting and pumping expenses and enables the rock to be gravitated to the reduction works with practically no handling. Where the reef cannot be opened from the surface by levels on it cross-cuts are put in from the face of the hill. This method has, however, been greatly overdone in the past; in one case a cross-cut 800 ft. long was put in, which gave only 80 ft. of backs. The excessively high cost of power in the early days doubtless was the cause of this, anything being done to avoid hauling and pumping expenses. Shafts are not common, their sinking and equipment being, as a rule, a little bit beyond the small man; but the increased depth of mining has necessitated them at some of the older companies and at those which are situated in flat or gently undulating country. The walls stand remarkably well, little or no timber being ever required in the stopes, and heavy ground is usually only encountered near the mouths of adits or in the somewhat treacherous talcose schist of the low-lying Jamestown district. All methods of stoping are employed, the governing factor being the run of the gold which, in most cases, is so erratic as to preclude the possibility of laying out development beforehand with any degree of certainty. With these steeply dipping reefs levels 120 ft. apart can be carried without any other trouble than a little water in the winzes as they get near the next level, the practice of rising to connect being rarely followed. Ventilation gives little trouble, though occasionally home-made centrifugal fans worked by hand are installed in unconnected levels. All the mining work is done by hand labour: the rock drills at the Sheba have been put out of commission and there is only one running on development at the Worcester. The following description in conjunction with Figs. 3, 5 and 6 will explain the methods of development and stoping followed at the Worcester.

The drives are usually driven along the richest portion of the reef and cross-cuts about 150 ft. apart are put in from the drives from wall to wall, to prove the width and value of the ore body. The levels are arranged to give backs of about 100 ft., and vertical winzes are sunk in the ore body about 200 ft. apart. The reef is mined by the underhand system, and the levels are protected by leaving a rib of ground 6 ft. above and 10 ft. below them. The hanging-wall is strong, and requires very little protection, although pillars varying in size from 8 ft. to 20 ft. square, according to the width of the stope, are left where considered necessary, generally midway between levels and 50 ft. to 60 ft.

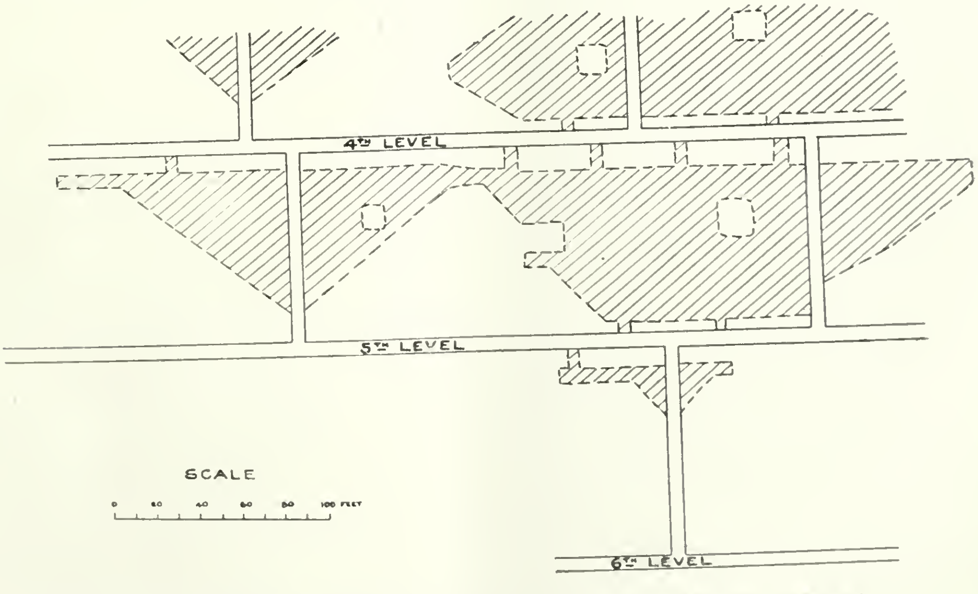


FIG. V.—Longitudinal Section, showing method of Stoping at the Worcester Mine.

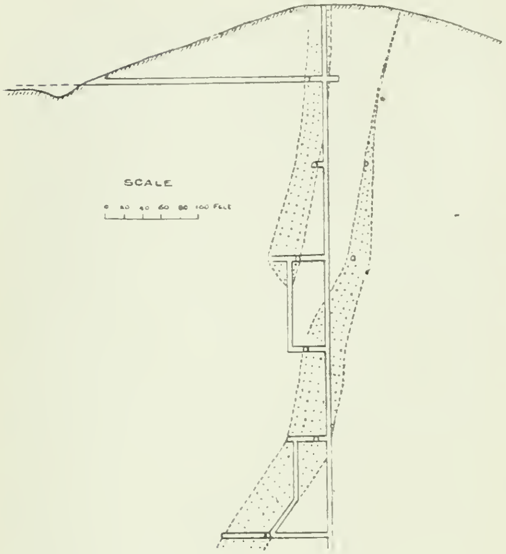


FIG. VI.—Cross Section of Ore Bodies at the Worcester Mine.

apart. Where the main shaft passes through the reef it is protected by a rib of ground of 20 ft. on each side. The reef is harder than that of the Rand, but it breaks better and requires less explosive. All roof holes are put in by double hammer, as it is impossible to drill them single handled. Holes 3 ft. deep constitute a native shift, but a few of the best boys put in holes up to 4 ft. 6 in. 5 ft. holes were tried in the stopes, double hammer, but the numerous fracture planes causes so many fast holes that the practice had to be abandoned. The amount of gelignite used in stoping is on the average 25 cases per month, or slightly under 2 cartridges per ton of ore broken. In development, blasting gelatine is consumed at the rate of one case to every 15 ft. of advance. The average amount of ore broken per hammer boy per shift is 1¾ tons. As there are several connections ventilation is good. The quantity of water amounts to 100,000 gallons per day, and is handled by an 8 in. Cornish pump in two lifts.

Transport.—This will always be one of the minor difficulties in the district, owing to the rugged nature of the country. All the mines on the main range make use of the humble but effective donkey to carry stores from the foot of the hills. The first cost in this case is practically the only one, as the donkey feeds itself and is almost immune from practically all the diseases that affect stock in South Africa. To get heavy machinery to the top of the hills the following method is adopted. It is brought by waggon to the nearest available point at the foot of a ridge whose surface is as free as possible of boulders and rock outcrops, this being a point of more importance than the grade. The front wheels are then taken off the waggon and a rough body bolted on to their axle with two stout sharpened poles as trailers. The machinery is then securely fastened to the frame and, if sufficient natives are available, is pulled direct to the top of the mountain. If there are not enough natives to pull direct, strong poles are fixed at intervals up the ridge and block and tackle employed. Loads up to 4 tons are easily taken up 1,000 ft. of mountain in a day by this method If the load is moving along the edge of a ridge, side ropes are held by natives to prevent it capsizing, and should any unforseen accident occur, such as the breaking of a rope, the pointed trailers take up the load and prevent the outfit from careering down into the valley again.

Reduction.—The small high grade reefs of the district with ore easily distinguishable from the country rock lend themselves readily to sorting, which is done to the extent of 80% in extreme cases. The majority of the reefs are free milling, such mines as the Consort, Mount Morgan, Rosetta, etc., being exceptions, but still it may be said that Barberton ores are refractory when those of the Rand are taken as a standard for comparison. In the more refractory mines plates are dispensed with altogether, practically no gold being recoverable on them, while with others the plate recovery may run up to as much as 80%, indeed, in one instance, the Zwaartkopjes ore body at the surface, the plate extraction reached 90% ; such high extractions are, however, rare.

The fluctuations in recovery by cyanidation are also considerable, and may run from 20% to 80% as a rough estimate.

The mills now running vary in style from the humble 1-stamp of the Tiger Trap, driven by a home-made water wheel to the 40-stamp turbine driven equipment of the Worcester, and as the cyanide plants also show similar differences, all that can be attempted is a description of one or two plants.

The Worcester mill consists of 40 stamps of 1,250 lb., running 96 drops per minute of 8 in.; 900 mesh screening is used, and the duty is 3·75 tons. No amalgamation difficulties are experienced, and the plate recovery is 45% of the screen assay. The cyanide plant consists of 6 circular steel treatment and 3 collecting sands vats, each of 180 tons capacity. The treatment follows closely that generally adopted on the Rand, and occupies from 7 to 8 days. The extraction is 75% of charge value. The slimes vats are 6 in number, of 30 ft. diameter, 8 ft. deep, with a 2 ft. cone. Two are used as collectors, two as treatment vats, one as a residue vat, and one as a spare solution sump. The extraction is at present 75%, and it is anticipated that this will be improved to 80% shortly. The total extraction from all sources is between 80% and 85%. The bulk of the slimes is exceptionally heavy and thick, and there is also a very fine slime (probably talc) which is most difficult to settle. The Adair-Usher process is used. This fine slime also comes down with the sands solutions and passes through the precipitation boxes where, when the ore from the mine happens to be at all acid, it coats the zinc and seriously interferes with precipitation, though with fresh ore it appears to exercise very little deterrent effect. Other mines in the district, as well as in Rhodesia, have also had trouble from the same cause.

The Thornton-Little may be quoted as representative of many of the smaller plants. This has a battery of 5 stamps of 750 lb. weight, running 96 drops of 8½ in., the screen being 900, and the duty 2·5 tons. As is generally the custom with small plants, the ore is fed into the mortar box by hand, a practice that is not conducive to a high duty, as the coloured gentleman who does the feeding likes to fill the box up well so that he may have time for thought before the effort need be repeated. The sands are treated in wooden vats, and the slimes are collected, dried and mixed with them. The plate extraction is 50% and the cyanide 46%, the total from all sources being 75%. In this battery the daily addition of sodium amalgam in the mortar box has proved very beneficial.

The Sheba possesses a fine plant of 120 stamps, frue vanners, roasting furnaces, and cyanide and slimes plant, but as this company has fallen on evil days, which is to be sincerely regretted, and is in a state of transition, present day figures are not of much value. The returns for the year ending June 30, 1908, show 80 stamps running, and 90,650 tons crushed of an average value of 6·518 dwts. The charge values were, concentrates 63·61 dwt., sands 2·76 dwt., and slimes 3·62 dwt. The plate extraction was about 40%, concentrates (roasted and cyanided) 84%, sands 57%, slimes 66%, the total extraction from all sources being 74%. The ore contains iron pyrites, a little arsenical pyrites, and some oxide of tin.

Settling tanks are generally used on the smaller plants and spitskasten very seldom, and the treatment vats are either of wood or corrugated iron. The latter construction stands well up to a capacity of 50 tons of sand; beyond that they are apt to collapse concertina fashion even though buried up to the rim in sand. The ore makes a good deal of slime, but slimes plants are beyond the means of the small man; in a few cases Scoular or Wilfley tables are used for concentrating. The method of discharge is by shovelling over the side, preferably into the bed of a creek. In some instances the plate extraction is so high that cyaniding is somewhat in the nature of a superfluity, and in some others the old hands fight shy of such a mysterious agency, relying on the simple formula of "put your mill up and shove your ore through it," to accomplish the desired result. Refractory arsenical and other sands give good extractions if subjected to years of weathering, but current sands of this nature decidedly do not, and no process of simple application seems to have been yet devised to overcome the difficulty. Roasting would be one solution, but to erect the necessary plant would mean a considerable expenditure, which, with fuel costs, would require the resources of a company. In the case of graphitic ores probably a gentle roast would be sufficient to eliminate the occluded hydro-carbon gas in the graphite which —and not the graphite itself—is the real precipitating agent.* This question of refractory ores has not received the attention in this district that it calls for, and the erection of a central reduction works to treat refractory high grade concentrates would be a great boon to the small worker.

Cost of Plant.—The cost of plant on a 5-stamp basis, including erection, would be about: mill (second-hand) £150 to £180, suction gas engine (15 h.-p.) £370, cyanide plant and accessories £150. In addition to the foregoing cost, a further £800 to £1,000 will be required in most cases to be put into the mine and to provide for contingencies, but £2,000 may be considered a fair estimate in most cases as capital required to reach the producing stage free from debt. Two cases have been brought to the writer's notice which reached this desirable point on a capital expenditure of under £500, and have paid well ever since: these, however, are quite exceptional. There has been in the past no trouble experienced in buying second-hand 5-stamp mills, requiring only new plates, and there is little likelihood that in the future the supply will run short, as there are in the district more silent than producing properties; moreover, there is a tendency among the more experienced tributors to buy their mills on the Rand because of the heavier stamps obtainable there, and with the new mills going up with 2,000 lb. stamps and big amalgamations and consequent reorganisation a scrapping of out-of-date plant, there is likely to be a good supply available on the Rand without drawing on Barberton at all. For new plant a local firm quotes the following prices:— One 5-stamp mill, 850-lb. stamps, with crusher, ore feeder copper plate, A frame, shafting, pulleys, belting and tightener, delivered f.o.r. Barberton, £580; one 5-stamp mill, ditto, but with 1,050-lb. stamps, £610. For driving the 850 lb. stamp mill a suction gas engine, developing 20 b.h.p. at the coast, equivalent to 15 b.h.p. at Barberton, is recommended. One suction gas engine of this size, with magneto electric ignition gear, variable timing device, gas producer, tanks, accessories, pipes and spares, £320 f.o.r. Barberton. For the heavier mill an engine capable of developing 32 b.h.p. at sea level, or 23 b.h.p. at Barberton would be required; this, with the usual accessories as above is quoted £460 f.o.r. Barberton. Another firm quotes for a 3-stamp mill of 750 lb. to 900 lb. falling weight, £200; and for a 3-stamp mill of 250 lb. falling weight, £150. These prices of course do not include the cost of erection; this in the case of a gas engine would mean another £25 to £50. Charcoal costs £4 10s. a ton, and the consumption may be taken as varying from 1 lb. to 1½ lb. per b.h.p. per hour. Mines burning their own charcoal obtain it for 2s. 6d. per bag of 70 lb. weight. Natal anthracite costs £2 5s. a ton f.o.r. Barberton, but is too clinkery and tarry to use alone in the gas producer. A mixture of one-third anthracite and two-thirds charcoal is fairly satisfactory, but charcoal by itself is the best fuel and probably the most economical in the long run.

---

* See this *Journal*, July, 1909, p. 24.

Working Costs.—The vastly different conditions prohibit an average working cost per ton crushed being struck. With a 5-stamp mill crushing from a large, soft reef and employing 2 whites and 25 natives, a return of 2 dwt. over the plates pays well; while in the case of a high grade, hard, narrow reef, sorting up to 65% of rock mined, crushing 250 tons a month, and employing 5 whites and 100 natives, costs will be quite 37s., or even higher. It may be taken as a fair approximation that a 5-stamp proposition, with the normal labour force of 4 whites and 60 natives and sorting 20% of the rock mined should run at about 18s. per ton crushed. Taking the larger mines, the great variability of conditions also prohibits any attempts at averaging. The Sheba costs for the year ending June 1908, were: mining 6/6·13, development 4/-, tramming, hauling and pumping 2/4·72, milling 3/6·88, vanning and roasting 1/3·82, cyaniding 2/9·76, tailings removal 0/6·29, general 2/5·44 total 23/7·04. The costs of the Worcester for the year ending, 1908, amounted to 13/11½, made up as follows:—

	s.	d.
Mining	5	7·11
Transport of Ore ...	0	1·14
Milling	2	3·32
Cyaniding	2	5·24
Slimes...	0	5·33
Development	1	8·16
General	0	11·42
Head Office	0	5·77
	13	11·49

White labour, if no special skill is required, is ample; but skilled men in mining or reduction are not plentiful, and the highly-trained technical man is rarely met with; but it is a mistake to import skilled miners from the Rand for the small mines, as the reef conditions are so totally different and the scale of operations so small that one not used to the district or to a similar class of mining is hopelessly lost for some considerable time. A manager of a small property would receive £40 to £60 a month, and miners and shiftsmen £25, the latter usually working 10, sometimes 12, hours. The Worcester works an 8-hour shift underground and a 9½-hour shift on the surface: all stoping is done on the morning shift from 7 a.m. to 3 p.m: stopers receive 17s. 4d. per shift, trammers 15s. 6d., and developers 35s. per foot on contract, contractors being charged 60s. per case for gelatine, and 2s. 3d. per shift for native labour.

In estimating the cost of stores, it may be taken as a basis that prices are from 15% to 20% higher that those current on the Rand.

Native Labour.—This is nearly all voluntary labour and, as a rule, fairly plentiful. The natives live in huts erected by themselves, and seem a very contented lot. Hammer boys get from 30s. to 50s. a month according to their efficiency; surface boys and trammers 30s. to 40s. The cost of feeding is 10s. per month per boy, in addition to the above. Engagements are generally monthly. Their rations consist of mealie meal, sweet potatoes, and meat on Sunday, an occasional beer drink being added. Swazis are not particularly good mine boys, partly owing to the intermittent periods during which they work, brought about by the counter attractions of home life across the border; there is, however, a sprinkling of coast boys, who leaven the mass somewhat. The average all-round rate of pay may be taken at 40s. per month. In drilling in hard quartz only expert boys can drill two 2-ft. holes on a shift.

Power.—While there is enough water for reduction purposes only rarely is it available for power as well in such cases as the volume is generally small and the head high—700 ft. at the Mount Morgan—the Pelton wheel is used. There is little doubt that the advent of the suction gas engine has made a wonderful difference to the small man, and it may be said to the companies also. At the Worcester the hauling engine and Cornish pump are worked by steam and the rest by water power. In dry seasons, however, the amount of water is inadequate, consequently a 120 b.h.p. suction gas engine is to be erected to make up the deficiency. Two turbines are in use at this mine, both obtaining their power from the North Kaap River. The larger, which is 400 yards from the mill, receives its water through a race $1\frac{1}{2}$ miles long and has a head of 55 ft. This is a vortex twin discharge machine with a 4 ft. wheel running at 168 revs. and developing 175 h.-p. From this turbine power is transmitted to the mill by means of an endless wire rope of $\frac{3}{4}$ in. diameter, 6/19 construction, running at a speed of 5,000 ft. per min. The distance apart of supports is 300 ft. and the life of the rope four months. The cost of running this turbine is 2·66d. per ton crushed or 5/9 per h.-p. per month The smaller turbine is a single discharge machine with a 28 in. wheel and develops 75 h.-p. It is situated two miles from the mill and has a head of 45 ft., the water coming through a race $\frac{3}{4}$ miles long; the transmission is by electricity at 500 volts. The Sheba ran its mill with steam power, but there will be sufficient power available at its generating station on the Queen's River to run the reduced plant now operating.

Health Conditions.—It is necessary to discriminate between the town by itself and the district as a whole, as the former enjoys the distinction of being one of the healthiest towns in the world, the white death rate for the last three years averaging only 7·5 per thousand, or half that of the United Kingdom. The prevalence of malaria, however, renders portions of the district unhealthy, and the deaths from this cause form 38% of the total; but even with malaria the general death rate will compare favourably with other countries. The rare occurrence of enteric or pneumonia is a factor of great importance, and miners' phthisis is unknown. In the winter months the climate is delightful, but during the summer is very hot and enervating. The altitude of Barberton is 2,830 ft., and that of the highest point in the district 6,250 ft.; and the rainfall for the last two years averaged 33·49 in. During the period June, 1908 – June, 1909, the highest absolute maximum temperature registered was 96·7° F. in the month of September and the lowest absolute minimum 38·4 in July, the highest average maximum being for the month of December 82·8 and the lowest average minimum 47·3 for July.

General.—The conditions topographical, geological and metallurgical vary so surprisingly in mines situated close to one another that generalisations, such as we are used to on the Rand, are not possible. One noticeable feature is the atmosphere of reserve with which the tributor surrounds his operations, an attitude which is a contributory cause to much of the neglect from which the district suffers. Publication of returns is essential if a country wishes to go ahead in small mining, as it is the only way of bringing its potentialities to the notice of the outer world. The Rhodesian Chamber of Mines and the papers of that country publish every month a full return giving district, name of mine, name of operator, number of stamps, tons treated and fine gold won of every mine in the country, small and large, and in a recent one these particulars covered the operations of 219 mines, 138 of which had five stamps or under. This sound policy of publicity has attracted much attention to Rhodesia, and has been largely the cause of the rapid progress of that country from a mining point of view. Again, in Barberton there is no association of tributors, and they are therefore quite unable, as a body, to make representations to the Government on points affecting their operations, a very serious disability especially in times of labour shortage.

Though the irregularity of auriferous deposition and the uncertainty of reef continuance render operations on a large scale generally undesirable, many chances offer for the small syndicate: but even with these latter considerable caution must be exercised if disaster is to be avoided. The chief reasons for the failure of small mines may be conveniently collected under the following headings:—

1. Bad selection of ground.
2. Inefficient methods.
3. Lack of capital.
4. Bad management.

In the selection of ground it is essential that the reef itself as well as the related geological conditions should be studied with extreme care, and that nothing whatever be taken for granted; information given by interested parties, though often reliable, occasionally suffers from a low standard of authenticity, and the ten ounces of the seller are apt to materialize as the two dwts. of the buyer. The breaking qualities of the reef, stability of walls, contour of surface, water supply, transport facilities, and particularly health conditions, are only a few of the many points which must be investigated. To pass a sound judgment on all these requires special qualities in the man who does the investigating a point sometimes overlooked by Rand syndicates, who more often than not are averse to spending money in these preliminaries. It is much better to spend a hundred or two in a thorough examination, and then, if unsuitable, turn the proposition down and cut the loss than to spend only a few pounds and then take the risk of losing thousands.

The manager should be a good all-round man with some business sense, no false pride, and capable of turning his hand to anything. A plant may be hung up for days because the miner is on the "bust," and nobody on the property knows how to blast the holes, or the gas engine may jib, and so on; contretemps like these frequently happen and test the adaptability of the manager. The lack of capital operates principally through shortage of development and inability to keep the mill running full time, and also necessitates the purchase of cheap plant not up to the work it is called upon to do. Inefficient methods are largely a corollary of the above; but with these ores metallurgy is of such importance that the application of wrong methods of extraction, may have very serious results.

In concluding, the writer would like to thank Messrs. A. B. Inglis, C. B. Kingston, G. W. Lowdon and R. Wormald, members of the Society, for their kindness in furnishing him with much information about the district, and for helping to make the few weeks he spent there pass so pleasantly.

The President: I am sure we are greatly indebted to Mr. Richardson for his extremely interesting paper. It deals with a district which many of us know well, and those who have spent a few months or even years in the Barberton district will appreciate the facts which the author has brought forward. There are many interesting points in the paper which we will be able to emphasise at a future meeting, and the Society will be indebted to anyone who has anything to say. I should like, therefore, to propose formally a vote of thanks to Mr. Richardson for the interesting paper he has given us to-night.

The vote of thanks was agreed to.

Mr. A. Richardson (*Member of Council*): I omitted to mention in the paper that I have brought with me to-night for examination specimens of the various reefs mentioned. These are not brought because they may contain visible gold, but on account of their being typical of the ore being put through the mills on the different properties.

THE ASSAY OF CYANIDE SOLUTIONS AND SLIME RESIDUE CARRYING DISSOLVED GOLD.

By A. Whitby (Member of Council).

At the outset it must appear that I have entered on an exposition of methods which are, to some extent, classical. There is a legitimate argument, however, to explain this late appearance.

At the time (1902) when the first experiments were made, I was imbued with the idea that, simple as the process was, the ground had been sufficiently covered by Virgoe, and I therefore contented myself with a discussion of Virgoe's original process embodying therein an idea garnered from Prof. de Wilde's patented process for the recovery of gold from cyanide solutions.* It will be remembered that de Wilde's patent covered the precipitation of gold from a cyanide solution by means of a cuprous salt as auro-cuprous cyanide. It was, therefore, but a short step to supplement Virgoe's method by adding a reducing agent to complete the precipitation of gold shown to be defective in his process. This should have been sufficient, one would say, to guide assayers to a correct appreciation of my final method. Unfortunately, however, I have recorded instances of sulphate of soda having been used as a reducing agent, of ferri-cyanide instead of ferro-cyanide added as an adjunct, and lastly of solid lumps of cyanide used to strengthen the solution. It may be well to point out, at this stage, what every industrial worker in the sciences never ceases to reiterate—keep to your working conditions till you prove them at fault.

Of late, I have received many enquiries for a description of this method, and perhaps, owing to not having placed the details on record in specific form, have even found it identically described in an oversea journal as a new and original process devised by another chemist.†

These facts have determined the writing of this short note, more especially as the process has now had time to settle into working grooves and, when carried out properly, never fails to give complete satisfaction. I take it, that members will not accuse me of self-advertisement if I desire to be credited in full with the initiation of a method which compares favourably with any other existing one, and during seven years has never been assailed or condemned by any competent authority. May it be noted, however, that originality is not claimed, but merely service adaptation of other men's ideas.

In describing the process it will be well to detail the reagents ordinarily used:—

1. Solution of copper sulphate (cryst. 15%).
2. Solution of sulphuric acid (1 – 5).
3. Solution of sodium sulphite (cryst. 10%).

Occasionally used:—

a. Solution of potassium cyanide (20%).
b. Solution of potassium ferro-cyanide (10%).
c. Solution of sodium hydrate (5%).

In practice 20 assay tons of solution or 583·3 c c. are placed in a capacious flask and 15 to 20 c.c. of No. 1 solution added. When the cyanide strength is much above ·2%, more of the copper solution is required. After shaking, 7 or 8 c.c. of No. 2 are added. This produces a marked change in colour with an acid reaction. If the solution, however, from some cause or another was originally very alkaline, it is possible more acid will be required. After again shaking add 20 to 30 c.c. of No. 3 solution and shake vigorously. The mixture should now smell strongly of SO_2. Repeat the shaking occasionally during five to ten minutes, settle, decant on to a large ribbed filter paper, finally transferring the whole of thé precipitate to the filter. Rinse the flask once or twice with water on to the filter, drain and transfer filter to a fire-clay crucible. Flux with about two of borax and one of litharge using sufficient reducing agent for a 25 gm. button. See that the entire filter is well covered with the flux fuse, for twenty minutes, and if filter is not entirely burnt off, finish the fusion with a wash of litharge. Ordinary cyanide solutions carry sufficient cyanide and ferro-cyanide, but the operator must be careful to see that there is a slight excess of copper in his filtrate.

---

* *Vide* vol. iii., pp. 6-12.

† "Easy Method of Assaying Solutions," Walter Seymour Duprey, *Mining Science*, July 15, 1909. Reproduced from *Mexican Mining Journal*, July, 1909.

In the case of waters, slime solutions or others carrying little or no cyanides or ferro-cyanides, first render faintly alkaline then add sufficient of occasional solution (*a*) to make up to not more than ·1% KCy, five or six drops of (*b*) and proceed as usual.

Auric chloride solutions, however, do not yield to this treatment, on account probably of the formation of auri-potassic cyanide and difficulties attached to its reduction. There is here a point I have long desired to investigate, but have never had any opportunity. What is the precise province of the sulphite of soda, if not to reduce the gold from some higher state of oxidation? If the process is conducted in stages, *i.e.*, if the filtrate after treatment with the copper and sulphuric acid solutions be precipitated with the sulphite, the residual gold is recoverable from this further precipitate. On the other hand, why is not a solution of auric chloride in cyanide, forming, according to the books, auri-potassic cyanide amenable to the process without prolonged boiling with a reducing agent before treatment?

Slimes Pulps.—It would be invidious on my part to dilate on the history of slimes residue assays on these fields. It is sufficient to say, that under ordinary conditions of drying, the bulk of the dissolved gold fails to make its appearance in the assay. This has puzzled metallurgists and led to considerable controversy. The fact, however, remains.

Coincident with my experiments on solutions I conceived the notion of treating slimes pulps as solutions, adding to them such quantities of the reagents as were necessary for the ratio of solution to dry slime, bearing in mind that such solution carried but little cyanide and practically no ferro-cyanide. I found that such a pulp could then be dried without loss, and the reagents added were so infinitesimal that they could not influence the weighings. Usually slimes residues were conveyed to me in screw-cap jam jars, and the necessary reagents added, the cap screwed on, the jar well shaken and the pulp afterwards transferred to the drying vessel, either a porcelain dish or enamelled pan. With this treatment the total gold, dissolved and undissolved, was recoverable.

As independent testimony of the efficacy of this method, I will quote from a communication received from an assayer, who was initially a sceptic.

Sample of partially treated slimes pulp in three portions:—

		Assay Value dwts.
No. 1.	Treated as above described ...	5·02
No. 2.	Dried without treatment ...	3·75
No. 3.	Calculated value made up as follows: Solution 63·7% A.V. 1·6 dwts., washed residue 2·2 dwts. ...	5·01

This, of course, is by no means typical of current slime residue pulps, but it shows a loss of 45% of the dissolved gold by simple drying.

I scarcely like to say how high I have found the difference in my own practice on ·3 to ·4 dwt. slimes residues.

Before concluding this short note I wish to affirm that as many as 30 to 40 solutions can be and are, comfortably precipitated, fluxed and cupelled alongside of current work daily in assay offices on the Rand, and the results so far have never been contested. I should also like to refer to my esteemed Mexican confrère who has curiously, and I hope unconsciously, advocated the use of ferro-cyanide as an adjunct to the preparation of his solutions. The addition of ferro-cyanide was originally suggested to me by Mr. Caldecott, and while its efficacy as well as the entire chemistry of the process remains obscure, it is certainly remarkable that another assayer should have so precisely imitated or originated the conditions of a test in constant use on these fields since 1902. In discussing this note, I hope members who have any suggestions to make regarding the chemical reactions involved will come forward, because discussion on these points will surely add to our knowledge of the cyanogen compounds. This is a country of strenuous workers who seldom have time or opportunity for investigations into theory and, as in this case, prefer to leave it alone so long as results are correctly obtainable.

Mr. H. A. White (*Member of Council*): I should like to propose a vote of thanks to Mr. Whitby for this excellent little paper, which deals fully but concisely with matters he has already brought before this Society some time ago.

I would point out that the basis of this method is Prof. de Wilde's method for dealing with cyanide solutions, brought before this Society in 1898 (vol. ii., p. 337). Dr. Loevy, who was then President, described the method as a perfect and complete precipitation for analytic purposes. Later on (vol. ii., p. 407) Mr. J. R. Williams supported this as the result of his own experiments. Since that time I have used this method very largely, but as my procedure was somewhat different from that described by Mr. Whitby, I will briefly describe it. I use nothing but a saturated solution of cuprous chloride in hydrochloric acid and a 5% solution of potassium ferrocyanide. The cuprous chloride may either be bought as such or the solution prepared by saturating strong hydrochloric acid with copper sulphate and reducing in a well stoppered bottle with excess of copper in the form of wire or strips. I do not find it necessary to add further cyanide even to mill water samples.

Suppose a sample of solution leaving the slime boxes is to be assayed, the method is as follows:

A very large beaker is taken, and in it are placed 2,333 c.c's. (80 A.T.) of solution and a drop of silver nitrate solution, sufficient to part 0·10 dwt. A few drops of solution of the ferrocyanide are then added, followed by sufficient acid cuprous chloride solution to bleach the first formed precipitate of cupric ferrocyanide so that the precipitate remains nearly white when vigorously stirred. Then in order to flocculate this somewhat unsatisfactory white precipitate a further addition of solution of ferrocyanide is made until on stirring the resulting brownish precipitate settles well.

It will be noticed that I use the ferrocyanide as an indicator of complete reduction of the solution and then as a flocculating agent in order to secure rapid but complete filtration. Its use in the latter capacity was mentioned by Mr. G. W. Williams (vol. iv., p. 504), and suggested itself at once to any experimenter who noticed how much more satisfactory was the filtration when ferrocyanides were present, as is usually the case with certain solutions sent for assay.

I agree with Mr. Whitby that an addition of cuprous chloride should be made to samples of slime residue before drying down, and in accordance with Prof. de Wilde's latest solution method (vol. viii., p. 327) only a few drops of a saturated solution are required, and there is no risk of loss of gold from copper in the lead buttons. We made several comparative experiments at the May Consolidated G. M. Co., and found a difference of 0·05 dwt. in favour of Mr. Whitby's method. This has not, however, I have been informed, the usual experience on at least one other mine, but I always recommend the adoption of this simple precaution which we certainly owe to Mr. Whitby. I call the author's attention to the correspondence in the May *Journal* on this question, as he seems to have overlooked Mr. S. H. Pearce's error in referring to a paper read before this Society by Mr. Rusden (vol. ix., p. 401).

EXPERIMENTS ON THE ASSAY OF ACID WASHES RESULTING FROM THE CYANIDE "CLEAN UP" BY THE USE OF BISULPHATE.

By L. J. Wilmoth (Member).

It is a common practice on many mines on the Rand to run the waste acid washes after clean up into large tanks for subsequent treatment with zinc fume for the recovery of their gold values. Mr. Donaldson's paper in the last *Journal* has prompted me to bring forward the results of a few comparative tests recently made on the assay of these acid washes, and I trust that other assayers will come forward with their experiences in the assay of this type of sample, so that we may arrive at a more or less standard method of assay. On these fields with the great uniformity of sample to be dealt with—both routine and bye-product work—there is little excuse for the diversity of methods in vogue at present, and it is time we assayers settled down to some uniformity of method. This diversity of method is probably more marked in our routine work than in anything else, and is to be greatly deplored. A standardisation of our methods would tend to much greater accuracy, which after all is what constitutes our value to this great industry.

This experimental work was done some two or three months ago, and I trust the results will be of some interest to members.

In all cases the amount of solution taken for actual assay was taken by weight and not by measurement, on account of the varying specific gravity of the different samples of solutions. In practice this point is of great importance, as any error at this stage may introduce a considerable error in the results. In most samples of these solutions an amount of about 20 A.T. should be taken, though with the richer ones 10 A.T. will give quite reliable results. The main object of these investigations was to ascertain if the method in use (spoken of here as the standard method) is accurate, and whether failing accuracy it could be improved upon by any other method.

The standard method is as follows, 20 A.T. of the solution to be assayed is acidified with 7 c.c. of a 10% sulphuric acid solution and then 25 c.c. of a 10% copper sulphate solution is added. The sample is brought to the boil and about 2 gm. of zinc fume is added. As the action starts, and in fact during the whole time of evolution of hydrogen the zinc fume shows a tendency to agglomerate, and care has to be taken that this is prevented by frequent stirring. Agglomeration of the zinc fume has a tendency to interfere with the accuracy of the results. After complete solution of the zinc the precipitated copper is caught on a filter paper which after filtering is fluxed in the usual way having soda in excess in the charge. The proportion of soda to borax is usually 2 : 1. The slags from this fusion are always very coppery, and the cupels after cupellation very heavily stained. Results from this method always agree very well.

The first step in these few tests was a comparison between the standard method and (1) evaporation with litharge, (2) boiling with an alkaline sulphide in an acid solution, (3) a slight modification.

The method No. 2 was as follows, 20 A.T. of sample were taken and acidified with strong hydrochloric acid. The sample was brought to the boil and 30 c.c. of 10% sodium sulphide added. The boiling was then continued for a further 15 minutes. A few cubic centimetres of a 10% solution of lead acetate were now added. This, on coming into contact with the solution, was immediately converted into sulphide, which acted as a collecting agent. The sample was now filtered and the filter paper and its contents fused with a little borax and litharge. The idea of this was to precipitate the gold as a sulphide and also to collect it free from any zinc and copper. Separating out the zinc first in any form would probably be unsatisfactory, as its bulkiness would probably lead to the retention of the gold to a considerable extent. The method No. 3 was merely a modification of the standard method. Thinking the copper in the slag and the lead button might have a tendency to give low results the following modification was tried: After collecting the copper precipitate on the filter paper it was treated with warm dilute nitric acid to dissolve it out. After complete solution of the copper (which is very rapid) the filter paper was run down with a little borax and litharge.

In fusing the residue from the evaporation with litharge a flux consisting of merely litharge and borax was used to demonstrate the effect of unsuitable fluxing. The following is a summary of the results of these four tests:—

Evaporation with litharge flux unsuitable	Standard method	Standard Cu dissolved	Alkaline sulphide
0·63 dwt.	1·55 dwt.	1·53 dwt.	0·90 dwt.

The low results by the evaporation method are accounted for altogether by the unsuitability of the fluxing. The only difficulty to be overcome is the presence of the zinc. Further samples were taken and another test run between the standard method and evaporation with litharge, this time using a flux suitable for the charge to be handled. The proportions of litharge and borax were altered and soda and a small portion of silica added. The results in this case were as follows:—

Standard method.	Evaporation with litharge.
0·26 dwt.	0·23 dwt.

Looking back at these methods it will be seen the results undoubtedly point to the superiority of the standard method over the alkaline sulphide method. The lowness of the results in this case can only be accounted for by the interference of the zinc contents of the solution. The evaporation method with suitable fluxing of the residue gives good results though slightly lower than the standard, here again the zinc may account for the difference. The lowness of the results with modification (3) is difficult to explain but is probably due to solution of gold as well as copper when the latter is being dissolved.

Another modification of the standard method was tried. This consisted of adding a small quantity of sodium sulphide to the solution after complete solution of the zinc, and continuing boiling for a further 10 minutes before filtering when the precipitate was treated in the usual manner. The results, however, showed no improvement.

Standard method.	Standard method and alkaline sulphide.
0·28 dwt.	0·26 dwt.

In all these comparisons the standard method gave the highest results, but the presence of the metallic copper on the filter paper undoubtedly is the weakest part of the process, as from there it passes to the crucible to be fluxed off as well as possible. This method relies upon the evolution of hydrogen by a zinc copper couple, and so a test was tried in which the copper was replaced by lead, thus giving an efficient and at the same an unobjectionable material to handle. The method tried was one described by Chiddy in the *Engineering and Mining Journal* some time back. In this test the standard method was run as usual, and the following was the lead method. 10 A.T. of solution to be assayed were taken and first 10 c.c. of a 10% lead acetate solution were added then 15 c.c. of strong hydrochloric acid. The solution is brought to the boil and 2 gm. of zinc shavings added. The boiling is continued until all the lead sulphate (due to sulphates present in solution) is decomposed and a clear solution obtained. The precipitate is filtered and fused under borax. There are several little points of importance to be noted. It is essential to decompose all the sulphate of lead present or mechanical losses will ensue, also the acid must be added before the zinc shavings.

The results by these two methods are as follows:—

	Standard method.	Chiddy's method.
	2·02 dwt.	2·14 dwt.
	·18 „	·22 „
	·49 „	·45 „
	·12 „	·13 „
	1·27 „	1·44 „
Av.	0·816 dwt.	0·876 dwt

In the above comparison it will be seen that Chiddy's method gives a result higher by ·06 dwt. In only one of the above tests, which are single assays on different samples does the standard method give the higher result. It would seem that the advantage lies with the use of lead acetate instead of copper sulphate in supplying the material for the couple.

A comparison of the cupels used in the different methods is most striking, and one of the chief causes of the advantage lies in the cleanness of the lead button for cupellation. A comparison of the slags by the two different methods also suggests a further cause for the difference.

A test was run to ascertain the effect of the sediment so often noticed in the sample bottles. A bottle was taken and shaken thoroughly and then a portion taken for assay while the sediment was still in suspension. The bottle was now placed on one side for a couple of hours, and when the sediment had settled a sample of clean solution taken for assay by Chiddy's method. The results were as follows:—

With shaking.	Without shaking.
1·27 dwt.	0·23 dwt.

This illustrates the effect of the sediment on the assay value. In reviewing these few experiments it would appear that to obtain the highest results the sample bottle should be thoroughly shaken up and Chiddy's method of assay used. The use of either the standard method or the evaporation with litharge if carried out properly with care as to the use of a suitable flux will yield good results, but slightly lower. Of these two, the standard method gives the better results as the copper exerts less injurious effects in the crucible than the zinc.

From the reduction works manager's point of view this last test is of importance as it is he who has to put the assay values to their true use either in guiding his work or in working out his reports. The assayer is merely interested in getting the true assay value of the samples submitted. The question naturally crops up what chances are there that the sample submitted truly represents the material sampled, but that comes outside the scope of this short paper, and is trespassing on the domain of the cyanide works. In any case, it is a question well worth investigating by those interested, and is one that should well repay the time devoted to it.

The President: This paper is exceedingly interesting to the assayers who are present to-night, and we are indebted to Mr. Wilmoth for bringing it forward. I should like to remark on one or two points as I have done a considerable amount of work in this connection. The solution for assay should be a weighed quantity as I have found this run as high as 1·12 in specific gravity. For the last five months, I have been carrying out some tests to check the standard method. During that time our acid solutions have been treated with zinc fume, in each lot assays being taken before and after treatment. According to the assay we ought to have got between 76 oz. and 77 oz. of fine gold, but when we had specially run it down in pots we got 86 oz. of gold bullion, which ran about 900 fine. I reckon that is about as close as anyone can expect to get with this class of solution, especially as the difficulty of sampling is great. I would propose a hearty vote of thanks to Mr. Wilmoth for bringing forward this paper to-night.

Mr. A. F. Crosse (*Past-President*): I should like to ask one question. In what condition do you think the gold is in these solutions?

Mr L. J. Wilmoth (*Member*): I think you will find the bulk of it is in suspension. There is probably very little in the solution. The high specific gravity of the solution makes it very difficult to get it all down.

Mr. A. Whitby (*Member of Council*): I have much pleasure in seconding the vote of thanks to Mr. Wilmoth for these notes, especially as the subject presents difficulties, and the author has given us the record of work done. I do not in any way wish to criticise at this juncture, but I notice that the comparison instituted between the author's method and Chiddy's appears to show that the latter is to be preferred. On general grounds the comparatively large amount of copper which would necessarily find its way into Mr. Wilmoth's lead buttons, would always tend to give low cupellation results, which will doubtless explain the comparative failure of the so-called standard method. I am quite in agreement with the author on the general ground of standardisation, but the term appears a little unhappy in this case.

Mr. L. J. Wilmoth: I would like to point out that the method which in this paper I call the standard method is not an original one of my own. I believe it originated in the Goldfields Laboratory.

Mr. H. A. White (*Member of Council*): Mr. President, I join issue with both yourself and the author of this paper in regard to the weighing of these solutions. You have just now pointed out that from a certain quantity you have recovered very closely the amount called for by assay; I am, however, fairly safe in saying that you estimated that quantity, not by weighing but by measuring the volume. I consider that, as is usual on these fields, a measured quantity should be taken for assay purposes where solutions are dealt with by volume, and that the assay returns should show the result as value per fluid ton of 32 cub. ft. This method has been in use for ordinary cyanide solutions for a very long time, and disturbing it in the method proposed only provides extra labour without increase of accuracy.

Prof. G. H. Stanley (*Member of Council*): I agree with Mr. White. If actual weight is required, it is an easy matter to get it by taking specific gravity with a hydrometer, but, as he says, it is volume which is always taken.

The President: Mr. White's suggestion tends to alter assay methods, and until this is general I am afraid we cannot accept it. The assayer reports as per ton of 2,000 lb. and not per fluid ton. The calculation of this value to the measured volume is easily accomplished by the cyanide manager.

Mr. F. W. Watson (*Member*): Why does Mr. Wilmoth add alkaline sulphide to a strongly acid solution, if H_2S in the nascent state is required? The author should have stated this, since it seems to me that it would be as efficient to pass H_2S slowly through the solution.

The Secretary read the following reply:—

THE DISTRIBUTION OF THE GOLD PRODUCED ON THE RAND.

(Read at February Meeting, 1909.)

By T. Kirke Rose, D.Sc., A.R.S.M., M.I.M.M., F.C.S. (Hon. Member).

REPLY TO DISCUSSION.

Dr. T. Kirke Rose (*Hon. Member*): I appreciate very much the kind observations made on this paper, but have not much to say in reply to the points that have been raised.

In answer to Mr. Caldecott, *(a)* I have no opinion to express on this point. *(b)* The gold bullion imported into the Royal Mint is for the most part refined by sulphuric acid. *(c)* I have no information as to the purchasers of gold exported from the United Kingdom. War chests are expensive luxuries, tending to defeat their own objects. There is nothing to show that the new supplies of gold are being hoarded except by private individuals in Egypt and India. They seem to be used generally in the Arts, for currency, and as a basis for currency.

In reply to Mr. Bevington, I see on referring to the tables that the chief changes in the imports of gold bullion into the United Kingdom from the Rest of World were as follows:—

	1905	1906	1907
	oz.	oz.	oz.
Germany ...	150	171,058	367,023
United States ...	6,363	18,576	170,006

These imports no doubt consisted of refined bullion of high grade, although the values are not given in the tables. The amounts are quite enough to account for the improvement in the mean fineness of the imported bullion, and show that the President's (Mr. Bevington) surmise is correct.

THE PILGRIMS REST GOLD FIELDS AND MINING METHODS.

(Read at March Meeting, 1909.)

By J. Moyle-Phillips (Member).

REPLY TO DISCUSSION.

Mr. J. Moyle-Phillips (*Member*): See Fig. I., "Vertical Section along the strike," should read "Vertical Section parallel to outcrop." As the arrow indicates that the general dip of the country is W, it is obvious that the section was not taken along the strike. The true bearings of the section are E.N.E. to W.S.W.

As regards Mr. Price-Griffith's contention that the Theta reef is "one well defined interbedded sheet," and not a coalescence of two or more reefs as quoted by me, it would be more convincing if he showed what became of the Bevets reef —which lies above the Theta reef and between it and the shales—when the Theta reef comes into immediate contact with the shales as it does in several places in the Clewer mine.

In quoting three reefs as lying in the dolomites viz., Bevets, Theta and Beta, throughout my paper, I am giving the general idea prevalent in this district, and also suggested by Mr. Hall in his report on the Geological Survey of this district in 1906. My own idea, after three years' observations, is that there is only one or possibly two true reefs, viz., Bevets *or* Theta or Bevets *and* Theta. Nowhere in the district as far as I have been able to ascertain have the so-called three distinct reefs been found in the same locality, and where two of them have been found only one has ever proved payable. Of the two so-called distinct reefs in the Beta mine —Bevets (or Theta) and Beta only, the so-called Beta has been worked, the other being unpayable. Again, at Frankfort, where they have the Bevets and the Theta (or Beta) reefs only, the Bevets reef has proved payable.

I cannot agree with Mr. Price-Griffiths that the term "indicator" as applied to that particular quartz leader lying immediately above the Theta reef in the Clewer, is a misnomer, as, where the reef does not outcrop, in sinking for the reef when one strikes this leader surely it may be called an indicator from the fact that the true reef is in close proximity.

As regards the manner in which the reefs were formed, I gather from the reports of the meetings of the Geological Society that that body is not itself prepared to state in what manner these reefs were formed, hence I fear to step in where they fear to tread.

I am sorry to have to tell Mr. Mance that the particulars he asks for are not available.

I tender my thanks to Messrs. Caldecott, Richardson, Mance and Saner, for their interesting additions to the paper, which have greatly enhanced its value, and I also have pleasure in acknowledging the assistance rendered me by Mr. Hall's report of his Geological Survey of 1906, and although many of his remarks are open to argument yet the report, as a whole, gives a very lucid and descriptive account of the district.

I also tender my thanks to Mr. Leggett-Neale —General Manager—and Mr. R. Sutherland, and members of the Survey Department of the T.G.M.E., Ltd., for their kind assistance, more especially the former, as without his approval and help the paper could not have been written, and, lastly, to Mr. A. Richardson, who has so ably chaperoned the paper through all its various stages, and to whom the idea of writing it is primarily due.

SMALL MINES OF RHODESIA.

(Read at September Meeting, 1908.)

By B. I. Collings (Member).

REPLY TO DISCUSSION.

Mr. B. I. Collings (*Member*): I am afraid Mr. Phillips misunderstood my remarks on the subject of the age of the ancient workings in Rhodesia. I consider that the more recent ones may be 200 years old, or possibly less, but there is no doubt that many of the ancient workings are very much older than this.

The tables supplied by Mr. Phillips, showing the length of the payable chutes, give results which are typical of the smaller quartz mines of the country.

Mr. Kingston points out that the present day tributor does not work under the same advantages as those who started the movement some four or five years ago. It is a very different business crushing ore from a developed property to opening up one from grass roots, and paying one's way while developing.

I trust that further Rhodesian papers may be contributed to the Society, as its mines afford innumerable problems of interest.

My thanks are due to Mr. A. Richardson for many suggestions while writing the paper, and for his kindness in reading it before the Society.

STOPE MEASUREMENTS.

(Read at May Meeting, 1909.)

By Otto S. Tonnesen (Member).

DISCUSSION.

Mr. K. Tonnesen (*Member*): In the *Journal* for July I find that Dr. Moir has offered some criticism on this paper. As you are probably aware, the author of this paper is at the present on a trip to England and the Continent, and his reply can therefore not be expected for some time. I consider that Dr. Moir has been very unfortunate in his criticism, and should therefore like to make a few remarks in connection with it. I regret that I could not do this earlier owing to temporary absence from town.

Anyone reading Dr. Moir's criticism without being familiar with the mathematical problems in question would think that the deductions in the original paper were wrong and misleading. That such is not the case has already been demonstrated by Mr. Coaton, who in his example shows a difference of 0·158 in 278·428 between the method of calculation adopted by the author and that advocated by Dr. Moir, or about 0·057%. But says Dr. Moir, the author should have pointed out that his method was an approximation. The author, no doubt, took it for granted that his audience knew that *any* application of mathematics to practical problems involves approximations. Dr. Moir's *own* method is an approximation, and is based on certain assumptions with regard to the properties of the tape line, which are not mathematically correct.

The assumption used by the author is generally adopted on similar problems in engineering calculations, not because we do not know the theory of a catenary curve, but because the equation of the parabola offers so much quicker and easier solutions, with an accuracy generally quite sufficient for all practical purposes.

The catenary curve would be the line of equilibrium for a *perfectly flexible* tape, which is *loaded* or weighted *uniformly* along its length, while the same tape would take the shape of a parabola if it was loaded uniformly along the *horizontal projection* of its length. When we consider how very small the difference between these two conditions is in a stretched tape, we can without further calculations realise that it is immaterial whether one or the other assumption is adopted. There are other conditions which will influence the shape of the stretched tape far more than this, for instance, the tape is never, mathematically speaking, uniform in weight even when new, much less when it gets muddy or wet; further, the tape will be more or less twisted, and the portions of it which stand on edge are less flexible than when it is on the flat. Again, if we want to be hairsplitting we should take into consideration that the tension is not constant but greater towards the end, so that the tape here will stretch more, and consequently weigh less per linear unit.

The author has in his paper on Stope Measurements figured out the area of the segment enclosed by the tape in order to prove how defective the old methods of stope measurement sometimes are. Now it is clear that if there is any appreciable difference at all in this area whether the curve is calculated as a parabola or a catenary, this difference would be in the author's favour as it would increase the area of the segment and consequently increase the errors of stope measurement with offset. If the full drawn line on the figure represents a parabola, then a catenary with the same sag would resemble the dotted line. Even if we are not familiar with the geometrical properties of the catenary, we can easily realise this when we remember, as I have stated before, that a tape which takes the shape of a catenary is one which is slightly heavier loaded towards the end than one which would form a parabola. But when we increase the load of a tape at a certain point, we also increase the curvature at the same point.

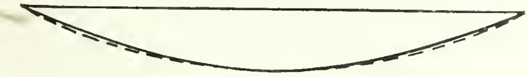


Fig. I.

I shall now deal with a couple of points in Dr. Moir's criticism, where he apparently lays himself open to counter attacks. The author in his paper is dealing with an example, taking the length of the offset line A—B as 316 ft. and the sag, projected on the reef plane, as 2 ft., and he says that the area of the parabolic segment equals two-thirds of the circumscribed parallelogram. No reader should be in doubt as to which parallelogram this refers, as the sentence is also expressed in figures: $\frac{2}{3} \times 2 \times 316$, the parallelogram therefore must have an area of 2×316, which is the circumscribed parallelogram I show in dotted lines on the figure.

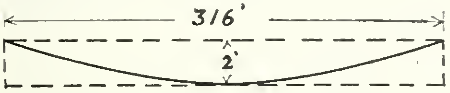


Fig. II.

Dr. Moir seems to think that the author is wrong in stating that the parabolic segment is two-thirds of this parallelogram, and assumes that the author has been mixed up with another mathematical sentence which Dr. Moir brings forward but which, however, has no direct bearing on the point in question

Dr. Moir has then gone to some trouble in working out the segment area by means of an integration of the catenary as expressed by hyperbolic functions. Had these mathematical deductions been correct, they would not have been without a certain theoretical interest, if nothing else just to show that we have also on the Rand men capable of dealing with higher mathematical problems. Unfortunately, however, the result appears to be wrong on the face of it, and therefore discourages any attempt to follow Dr. Moir's deductions. Dr. Moir finds the segment area to be:—

$A = \frac{Sl}{3} + \frac{Sl^2 \sin \alpha}{6a}$, when S = sag, l = length of tape, and α slope of same. If we, however, apply this equation to the case that $\alpha = 0$, that is to a tape with its two ends, both held on the same level, we get $\sin \alpha = 0$, and the equation takes the form $A = \frac{Sl}{3}$ or less than a triangle with a base = length of tape and height = sag, which is absurd. Dr. Moir's tape would have to hang something like the dotted line shown on the figure.

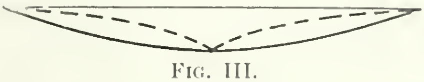


Fig. III.

THE EFFECT ON AMALGAMATION, OF DIFFERENT INTERVALS OF TIME BETWEEN THE DRESSINGS OF PLATES.

(*Read at June Meeting, 1909.*)

By G. O. Smart (Member of Council).

DISCUSSION.

The President: I may say we have adopted this method of twelve hourly dressing on the

Simmer Deep and Jupiter mill, and we find that it answers very well indeed. The plates, after seven weeks' running, are in first class condition, and we are getting the same proportionate quantity of gold as we had in the eight hours' dressing. We have also followed Mr. Smart's lead with the tube mill plates, and have adopted the eight hourly dressing with equally good results.

THE CONTINUOUS COLLECTION OF SAND FOR CYANIDING.

(*Read at October Meeting, 1909.*)

By W. A. Caldecott (Past-President).

DISCUSSION.

Mr. H. A. White (*Member of Council*): Although I notice no extravagant claims, references to "New Metallurgy" or the like, I consider that this able paper chronicles not only a departure, but a distinct advance in Rand metallurgical practice There are two features upon which the author may be especially congratulated: not only is the plant described actually running upon a working scale, but full details are supplied of costs and results in place of the more usual sanguine anticipations with which we are all familiar. So full and complete a description demands a careful consideration and detailed criticism of the data supplied. Any alteration in ore treatment to win acceptance must show either decreased capital cost, decreased working costs, or increased extraction, and the method advocated seems to display improvement, in all three directions, but the greatest stress is laid upon smaller installation costs. I cannot, however, congratulate the author upon the clearness of the estimates of capital expenditure quoted, and no proper comparison is possible, as the items are differently grouped and moreover mutually inconsistent. For example, the ordinary system quotes 9 items totalling £63,784 for treatment vats, structure, etc., against which I only find one item "treatment vats and structure" in the new scheme, and this amounts to £23,156. Notwithstanding this great difference, it is only claimed on page 51 that 26 vats are required in place of 48, whereas the above proportion would be about 26 to 71. Tracks under vats are not mentioned for the new system nor sumps for the old one. We may no doubt have greater confidence in the difference shown for the two systems which amounts roughly to £30,000, but it would be interesting to know if the method in use on the new plants of the East Rand Mines is not cheaper still. There the difficulty of treating in the collecting vats is evaded so far that no other treatment vats are provided. It is even possible that if perfectly slime free sand could be collected that the method would be economically sound. With regard to working costs, a figure of roughly one penny per ton is quoted, and I think it should be made quite clear that this refers to total collection cost, and not to extra cost for the new method. If such were the case the reduction in capital cost of £30,000 on 5,350 tons per day would mean the loss in working cost of £8,000 per annum. I have mentioned this point because I know that doubt has been expressed, although it seems quite clear to me that total costs are meant, and even that these are a little lower than with the usual system. With reference to extraction, the Simmer and Jack figures are most satisfactory, and though the actual assay differences are but small the tonnages and the strictly comparative nature of the test warrant the conclusion that more gold can be got in less time and that we are not dealing with a slight bias of the personal equation. Such a conclusion is confirmed by the assay values of the last drainings and an examination of the valuable curve diagrams. The figures below these diagrams show one point of interest which deserves discussion. The double treatment final drainings are steady at 0·08 dwt., and the single treatment come down to 0·20 dwt. only. Yet the dissolved gold, by difference from figures quoted, is practically the same in both cases. I cannot help thinking that a direct estimate of the dissolved gold would show very different results. This is of course quite a side issue, and does not affect the inference Mr. Caldecott has drawn, but it is possible that an apparent discrepancy may conceal some fact of real importance, as was the case in the discovery of argon in the air. A general review of the subject matter of this paper leads me to the conclusion that the success arrived at is mainly dependent upon the practically complete elimination of slime from the sand, and that this matter has not elsewhere in this country received quite all the attention its importance demands. I gather that the moisture in the collected sand will average about 16% and that in the discharged residue about 7% less. If 5,000 tons a day are treated on this basis, there will be an accumulation of weak solution of about 10,000 tons per month, enough to be embarrassing, unless Prof. de Wilde's or some similar method be employed for depriving it of gold and cyanide. Possibly, however, a better result might be obtained on the filter table by using a deeper bed with slower travel or by some other suitable method. I would suggest that further figures are required to demonstrate the gain in treating 8% of fine sand as slime, and should include account of the dissolved gold carried out by slime residue as compared with that associated with sand, the undissolved gold in the two cases, and increased cost due to extra wear of pumps (if any). In conclusion, I have to congratulate this Society upon a paper so valuable to us and to the world of metallurgy.

The President: I expect that Mr. Caldecott will deal with Mr. White's remarks personally, but I should like to say that during the past week we have been able to run with one vacuum pump for one table, and the result has been very satisfactory, as the 15% which Mr. Caldecott speaks of is rather high. Our present results shows 12% moisture in our sands leaving the table. With regard to the point that some 10,000 tons of solution will be brought into the plant monthly, I can only say that in our plant at the Simmer Deep it has not proved to be the case. A considerable loss of water must occur through evaporation, both of the solution in itself and of that in contact with the sands during transferring. Our last four results from slimes residues gave undissolved gold values at 0·08 dwt., 0·10 dwt., 0·09 dwt., 0·08 dwt. per ton of slimes, thus showing a very satisfactory dissolution of the gold in the slimes.

A METHOD FOR THE RECOVERY OF ZINC FROM SOLUTIONS OF SULPHATE.

(*Read at September Meeting, 1909.*)

By Wm. Cullen (Past-President) and G. F. Ayers (Associate).

DISCUSSION.

Prof. G. H. Stanley (*Member of Council*): In connection with this paper I mentioned last month that I had some two years ago deposited zinc from sulphate solution. I had some specimens by me, and I have brought them here for your inspection to-night. With certain strengths of solution and current conditions the zinc comes down in branching crystals, under other conditions in spongy masses more or less as are indicated on the blackboard (illustrated by means of diagrams). Some of the crystals are mounted for microscopic examination and ready for inspection under the microscopes before you, at the close of the meeting.

Mr. A. F. Crosse (*Past-President*): What would it cost to precipitate a ton of zinc? I think experiments have been made in Germany but it was found to be too expensive.

Prof. G. H. Stanley: After making these experiments and talking to one or two Cyanide Managers, my opinion was something like that of Mr. Crosse. I scarcely think the quantity would repay the cost and trouble, except on very large plants.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution:—

Mr. Thos. Donaldson (*Member*): Reference is made in this paper to the fact that magnesite is used for the precipitation of zinc as zinc hydrate. I would like to mention another purpose to which magnesite has been successfully applied. The contact substance used in the "oleum" plant at the Modderfontein Dynamite Factory for the purpose of converting sulphur dioxide and oxygen into sulphur trioxide, is magnesium sulphate which has been impregnated with platinum chloride, and dried at a high temperature. After being in use for some time this substance loses its power to "convert" on account of its being contaminated with impurities taken up from the gases. The principal impurity is iron, chiefly in the form of sulphate, but lead sulphate is also found, although in small quantity. These impurities are objectionable, and a means for their removal was sought, and found in the use of *magnesia*. In the course of the recovery of impure contact substance, the magnesium sulphate has to be dissolved in water, in order to separate the metallic platinum from it. This solution, which contains also the iron and lead salts, is heated up by steam, and stirred by compressed air, in order to oxidise the ferrous salts. Some calcined magnesite made into an emulsion with water is then added, and the agitation is continued. Sufficient magnesite is used to neutralise any free acid present, and convert the iron into hydrate. Even if insufficiently oxidised beforehand, the iron is soon converted into ferric hydrate, and precipitated, carrying down with it any lead present in solution. With regard to the manner in which the lead is brought down, I would refer any one sufficiently interested to Mr. Wilkie's paper on this subject published in the *Journal of the Society of Chemical Industry*, June 30th, 1909. After the precipitate has been allowed to settle, the solution, free from lead and iron salts, can be decanted off, and the magnesium obtained as crystals by evaporation of the water. The working out of this short process has been chiefly in the hands of my colleague, Mr. Campbell.

THE SMELTING OF GOLD PRECIPITATES AND BULLION WITH OIL FUEL.

(*Read at June Meeting, 1909.*)

By Arthur Yates, M.I.M.M (Corresponding Member of Council).

DISCUSSION.

Mr. A. Thomas (*Member*): It is always a difficult not to say a hazardous matter to attempt to criticise or discuss a paper dealing with the methods of work and appliances used in another part of the world, such as the one we have been favoured with by Mr. Yates, from the Redjang Lebong Gold and Silver Mine, Sumatra, for so important a part do local conditions play in the estimation of the most economical and efficient means for adoption that what may be very good practice and quite justified in one case may be altogether unsuitable in another. And so it is not my intention to enter into a discussion on the comparative merits of oil as against coal fuel in smelting, and I therefore leave this aspect of the question to those better qualified to express an opinion. However, I can quite understand that oil does possess certain advantages over coal as a fuel as mentioned by the author in respect to the ease and cleanliness of manipulation, and the ready way in which it lends itself to the more perfect control and maintenance of the heat applied. But as the author freely admits, it is solely on account of its relative cheapness in the district of the mine that he was lead to use it in preference to coke and charcoal on which he had previously relied, and the Society has reason to be extremely grateful to the author for bringing to our notice the practicability and expediency of utilising oil fuel for smelting purposes.

However, it is rather on another point raised by the author that I would like to make a few remarks. In reading over his most interesting paper I was somewhat surprised at the short life of the retort he is using, which I think works out at 4·57 charges per retort. I am not quite sure as to the exact nature of the fluxes he is using in smelting the precipitate, but it is evident from the analysis given of the precipitate and retort that its character would be basic, due to the large amount of metallic oxides present, and this would have a tendency to corrode the plumbago retort; and in order to prevent or minimise the risk of this happening I am inclined to think that a clay liner inserted into the retort would amply repay for the extra cost entailed by prolonging the life of the retort. Also, such a liner would have a tendency to eliminate a great deal of the matte which is so troublesome while smelting such products. My experience has been that during the process of smelting gold precipitates in a plumbago pot with a liner in every case when a liner breaks through and the slag coming in contact with the crucible a matte will invariably be found on the bullion button. This fact, which I have repeatedly observed to happen may, it seems to me, be accounted for by the action of the carbon of which the crucible is principally composed on the basic sulphates which on breaking through the liner becomes reduced to sulphides which go to make up the composition of the matte.

I offer this explanation of the possible cause and mode of formation of matte on bullion buttons, and suggest a simple means whereby the chance of its occurence may be reduced or entirely obviated.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(*Read at May Meeting, 1909.*)

By Jas. Gray, F.I.C. (Member).

DISCUSSION.

Mr. W. R. Dowling (*Vice-President*): At the August meeting* I made the statement that the lime burners protected themselves in the sale of their lime.

The following figures confirm the statement. Nine trucks weighed during July, August and September by the Railway Department, with bags, 203,850 lb. Deducting 3 lb. per bag for 1,098 bags leaves 200,556 lb. as the nett weight of lime. The total weight as per consignment notes was 199,910 lb. The difference of 646 lb. equals about $\frac{1}{3}\%$ and might be considered less than the experimental error.

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

(*Reprinted in this Journal, September, 1909, from the Bulletin of the Institution of Mining and Metallurgy, August 12, 1909.*)

By W. A. Caldecott (Member).

DISCUSSION.

The President: I should strongly urge upon all members who have had any experience in the crushing of ore by these heavier stamps to bring forward their views on the matter. We have taken advantage of the generosity of the Institute of Mining and Metallurgy in allowing us to reprint this paper. It has been done not only for the information of members, but to bring forward discussion of the paper by men who are not members of the Institute of Mining and Metallurgy.

---

* See this *Journal*, Vol. x., August, 1909, p. 63.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THERMAL PROPERTIES OF CUPELS.—"In seeking an explanation of the differences known to exist between bone-ash and patent (magnesite base) cupels when in actual use, the thermal properties of cupels have been investigated by the authors. By means of a special apparatus which is described, the relative diffusivities of heat were determined for various brands of cupel, both at 100° C., and during the actual cupellation process. The patent cupels were found to have a higher diffusivity and a greater specific heat than the bone-ash cupels, and it was noticed that during cupellation at the same muffle temperature, the actual temperature of the cupelling button was much lower in the former than in the latter case. During experiments in the cupellation of silver with lead, considerable differences were also noted in the rate of cooling and in the time required for the solidification of the silver beads. It was thus found that, after being withdrawn from the same muffle temperature, silver beads take longer to solidify and 'spit' on patent than on bone-ash cupels; and that spitting is more liable to occur when the latter kind of cupel is used."—C. O. BANNISTER and W. N. STANLEY, *Institute of Mining and Metallurgy*, May 20, 1909. [Advance proof].—*Journal of the Society of Chemical Industry*, June 15, 1909, p. 606. (A. W.)

FUSIBILITY OF MIXTURES OF GOLD AND TELLURIUM.—"Gold is readily soluble in molten tellurium, a pasty solution being obtained if the proportion of gold exceed 60%. The fusion curve indicates the existence of a compound, Au_2Te_4 (43·59% of gold), solidifying at 472° C., but no other compound appears to be formed between these elements. On strongly heating the pasty mixtures, rich in gold, tellurium is driven off and finally, at 1,065° C., pure liquid gold remains."—H. PELABON, *Compt, rend.*, 1909, *148*, 1176-1177.—*Journal of the Society of Chemical Industry*, June 15, 1909, p. 606. (A. W.)

CRUCIBLE ASSAYS.—*Directions for Fluxing Charges.*—(1) Examine the material submitted for assay and estimate its approximate mineralogical composition. Then write down the corresponding weight of each mineral in the quantity to be melted (usually 0·5 A.T., called 15 gm.)

(2) *Silica.* Add, if necessary, enough silica to make with that already present in the ore 2 gm of SiO_2 for each gm of $CaCO_3$, $MgCO_3$, or $BaSO_4$, (calcite, dolomite or barite) present, and 1½ gm for each gm of Fe_2O_3, ZnO, clay or other infusible silicate (hematite, zinc mineral, kaolin, etc. Limonite is equivalent to ⅞ its weight Fe_2O_3 and pyrite to ⅔.) Consider 1¼ gm of powdered glass equivalent to 1 gm of silica.

(3) *Borax.* Add ⅓ gm of borax glass for each gm of silica or infusible silicate (clay or talc) present in the ore and the fluxes. For cupel assays add four gm of borax for each gm of bone ash.

(4) *Soda.* Add 1⅓ gm sodium bicarbonate for each gm of SiO_2 present in the ore or fluxes. Consider fusible silicates as glass, etc., equivalent to 50 per cent. silica. For pure lead ores add at least ⅔ the weight of the ore. For clay, talc, etc., use a mixture of soda and potash.

(5) *Litharge.* Add enough litharge (0—20 gm) to make with the lead already in the ore an 18-gm button and in addition for the slag, 2 gm of PbO for each gm of silica or infusible silicate in the ore. (Consider glass as equivalent to 50 per cent. silica). For cupels add 2 gm of PbO for each gm of bone ash. Increase the amount of PbO largely to keep copper, zinc, antimony, arsenic, and tellurium out of the button if they are present in the ore. For ease in correction for silver in the PbO use litharge in even multiples of 10 gm only.

(6) *Argol or nitre.* Add enough argol or nitre to make an 18-gm button. One gm of Fe_2O_3 may be assumed to prevent the reduction of about 1½ gm of lead, but the reducing power of sulphides should be determined by a preliminary assay if they are present. When more nitre than the weight of the ore is required do not use this method. Roast ores containing much pyrite, antimony or arsenic, adding charcoal for antimony and arsenic. If but little nitre is required use the nitre method in preference to roasting. For galena, etc., nails may be used with charges in which the PbO for the slag is replaced by $NaHCO_3$ in excess. When nitre is added in the nail method, be careful to form no FeO which will not fuse in the soda slag.

(7) Buttons brittle from antimony or zinc or showing scales of copper must be scorified.

This system of fluxing forms a bisilicate of Na_2O and PbO or $Na_2Pb(SiO_3)_2$. This is ample flux for a pure quartz gangue and in it the other bases dissolve forming, of course, slags of lower silicate degree. As given, the charge for pure limestone yields a slag of oxygen ratio of 1 in the base to 1·54 in the acid not counting borax. $BaSO_4$ gives a ratio of 1 to 1·7; $MgCO_3$, 1 to 1·47; Fe_2O_3 (as FeO), 1 to 1·33; clay, 1 to 1·42; etc.

Effect of the Borax.—Since borax is added to form a borosilicate it is reasonable to figure it according to the amount of SiO_2 present in the slag. The proportion given has not been finally settled upon as the best. It is sufficient to greatly lower the temperature of slag formation, to make the slag viscous during the reduction period, and to lower the temperature of final complete fluidity. It is also sufficient to serve as a general safeguard to flux a slight excess of either acid or base. On the other hand it is expensive and makes the buttons more difficult to clean.

Our charges are a little higher in PbO than usual and for some basic ores it will be more economical to substitute soda for litharge but this makes the charge more bulky. For ease in remembering it was thought better to have a constant ratio of soda and litharge to silica rather than to use the form 1 to 1½ gm of soda and 1·5 to 2 gm of PbO for each gm of silica, which is more nearly what would be used by an experienced assayer.

The amount of fluxes as given by this table are ample and without any great excess. In estimating weights to be fluxed even gm are taken, using care to see that the weight of refractory mineral is over-estimated rather than under-estimated. The minimum amount of the sodium lead silicate required to dissolve the various bases has not yet been accurately determined. If I ever conduct a class in assaying again, this will be worked out for the more common minerals by making up charges of the pure oxides, etc. It is also desired to work out the amount of excess litharge needed for each gm of copper, antimony, etc., in the charge to make clean buttons. This PbO method has been found to work well with up to 25 per cent. of antimony and 60 per cent. of zinc as sulphides but is not so successful for copper.

Dissolving Action of Litharge.—A large excess of PbO severely attacks the crucibles and the risk of ruining the muffle by spilled charges becomes important. Therefore it is thought best to scorify the coppery buttons as is done with the buttons obtained in the standard scorification method for copper mattes.

On account of this dissolving action of litharge upon the crucibles more litharge, as given in the directions, is of advantage in assaying clay and talc ores. The charges without excess litharge have very little effect upon Denver crucibles and attack the Battersea crucibles but slightly. The borax seems to protect the crucibles from being drilled by PbO probably by coating them with a viscous aluminium borosilicate.

We have found that by long fusion cupels can be entirely dissolved to a fairly fluid slag by much borax with some PbO and no soda or silica. This forms calcium borate and sodium phosphate as in the tests on platinum wire. The amount of bone ash in the charge is determined by weighing the cupel with the litharge and subtracting the standard weight of that size cupel when new, to give the PbO to be deducted from the weight of the stained part of the cupel. Apology is asked for the incomplete state of these data, but I thought it better to publish this now, rather than postpone it indefinitely in awaiting complete data.—A. A. Steel, *Engineering and Mining Journal*, June, 1909, p. 1243. (A.R.)

Cupellation Losses.—"At present there is little evidence as to the comparative variations in the absorption of gold and silver by bone-ash and magnesia respectively. It would be desirable in such experiments to state what bone-ash cupels are used. The chemical composition of commercial bone-ash varies considerably, and the screen analysis varies still more In the present paper the authors speak vaguely of 'bone-ash' without further specification. This is not sufficiently explicit.

The authors quote some measurements of muffle temperatures, but none of them are of much use (I include my own in this sweeping condemnation), for two reasons. In the first place the air current is variable and is in no case specified.

Secondly, the temperature of the air above the cupel cannot be taken by a thermo-couple on account of the radiation of heat from the cupel and from the walls of the muffle. The radiated heat is of course absorbed by the thermo-couple much more than by the air passing through, with the result that the thermo-couple always shows a temperature far above that of the air immediately surrounding it. These are among the causes of the great discrepancies between the different observations of muffle temperature.

The freezing point of litharge, given by Doeltz and Mostowitsch as 906° and by Mostowitsch alone later as 884°, is, in truth, a most difficult point to determine. A cooling curve gives no definite point of arrest. There is no doubt that at 906°, litharge is completely molten. At 640°, it is completely solid. Between these temperatures there is a long pasty stage. It needs some courage to announce the exact point above which the pasty litharge is to be called liquid, whilst below the point it is to be called solid.

A little light is thrown on the matter by my own observations that litharge is not absorbed by a bone-ash cupel maintained at 750°, but is absorbed by one kept at 825°; and by Fulton's observations that absorption does not take place at 815°, but is rapid at 883°. If absorption is to be taken as the criterion of liquefaction, then the melting point of litharge must apparently be placed at about 825°. In cupellation, therefore, the cupel must be above this, but the temperature of the air in the muffle near the cupel may be almost anything, and for the reasons given above must remain indeterminate.—Dr. T. Kirk Rose, *Institution of Mining and Metallurgy, Bulletin No. 57*, June, 1909 (H.A.W.)

Behaviour of Tellurium in Assaying.—The effect of litharge on tellurium was investigated in various ways. Lead buttons containing, lead 6 parts, gold 1 part, tellurium 1·5 parts were fused or soaked under litharge at a temperature of about 800° C. for 15 minutes and then cupelled. The litharge was black, lustrous and opaque. The lead button when cupelled yielded 96 of the gold. Duplicates of these fusions were left in the furnace, and the temperature raised to 1,000 to 1,100° C. for another 15 minutes, and on cupelling the recovery was only 94%. The slag was dark, red and glassy: evidently some of the tellurium had been reduced as indicated by the gold recovery. Again, equal portions of pure lead were fused separately with samples of the opaque and the glassy slags containing equal quantities of tellurium, with the result that the lead from the fusion with the opaque slag showed far more tellurium than the other. This would seem to indicate that a portion of the oxidised tellurium is reduced again when the temperature is raised.

A lead button of the same composition as previously described was dropped into litharge to which a little sand had been added, and which had been kept at a temperature of 1,000 to 1,100° C. for 30 minutes and allowed to remain in the furnace for 10 or 15 minutes more. The weight of the button was unchanged, and it was completely absorbed on cupellation. The slag was straw colour, and gave no indications of tellurium except for a slight reddish cloud, near the button. A repetition of the experiment, except that after the formation of the silicates the temperature was lowered and maintained at a low point after the introduction of the button, gave similar results. After the formation of silicates there is practically very little tendency for the tellurium to oxidise. An examination of the slags seemed to indicate that the reaction between the litharge and tellurium is represented by the equation:

$$2PbO + Te = Pb_2O + TeO.$$

and that the reverse of this reaction takes place at higher temperatures.

The effect of scorification, unless the operation is carried out at a moderately low temperature, is to concentrate the tellurium in the button. The most effective way to remove tellurium is to "soak" the button under litharge either in a scorifier or crucible at low temperature.

The effect of tellurium in a lead button containing gold and silver is to weaken the surface tension of the molten metal during cupellation. This effect of tellurium, assisting the general tendency of the film of litharge in the same direction, results in the tendency of the molten metal to "wet" the sides of the cupel. If the amount of lead is small in proportion to the silver, gold and tellurium, small portions of metal become detached, and either pass into the cupel or leave minute beads.

Lead is more readily oxidisable than tellurium, especially at high temperatures; if sufficient tellurium remains toward the end of the cupellation, complete absorption may take place, or in other words, complete absorption and subdivision are the result of insufficient lead. The tendency to become "wetted" is greater with bone-ash cupels than with patent cupels.

The presence of silver protects gold by acting as a diluent during the cupellation of buttons containing tellurium. The absorption loss due to the presence of tellurium is a function of the temperature of cupellation. An excellent bibliography is given in connection with the paper. — S. W. Smith. *Inst. Min. and Met.*, Bull. 44.—*Engineering and Mining Journal*, June 12, 1909, p. 1,176. (A.R.)

Recent Discoveries of Graphite in British African Colonies. — East Africa Protectorate.— Some specimens of graphite gneiss from the right bank of the Tsavo river, five miles below the Tsavo station, on the Uganda railway, associated with carbonate of copper were sent, but examination proved they were of no commercial value.

Uganda.—A specimen from the slopes of Ruwenzori consisted of dull greyish black graphitic schist containing disseminated flakes of graphite. The percentage of carbon amounted to 16·7 only. Another contained 31·2% carbon and 6·4% volatile matter other than water. Another only 32·68% carbon

Nyassaland.—In the north-west of the upper Shiré district the mineral survey observed seven different parallel bands of graphitic gneiss in which the graphite was interfoliated with other minerals in large flakes. Samples contained 70·6—91·6% carbon, valued if carefully cleaned at £40 per ton. A specimen from a deposit near the 'Mkoma Mission, in the Angoniland district consisted of practically pure graphite in the form of coarse flakes up to 2 in. in length. It was described as soft and flaky, and classed as ordinary lump of superior quality, worth about £32 to £34 per ton.

Rhodesia.—Samples from the north-east showed 16·85% of ash, mainly felspathic material which would detract seriously from the value of the mineral.

Natal and Northern Nigeria.—Samples from these districts showed no results of economic value.— *Bulletin of the Imperial Institute*, Vol. vii., No. 2, 1909, p. 166. (J. A. W.)

New Method for Determining Antimony.— "A method for the determination of antimony, in which the decomposition of the ore is accomplished with ammonium persulphate, is described by M. F. Coolbaugh and J. O. Betterton (*Western Chemist and Metallurgist*, iv., pp. 229-232). The use of this reagent avoids the danger of loss of antimony by volatilisation during fusion.

The portion taken for analysis is fused with 8 to 10 times its weight of ammonium persulphate for 15 minutes at a low temperature. A small Erlenmeyer flask can be used for the fusion. The flask is placed on an asbestos mat on a hot plate and after the first decomposition has taken place, the heat is gradually raised until quiet fusion is attained and allowed to remain in that condition for 10 minutes. The fusion is extracted with 20 c.c. of hot hydrochloric acid (sp. gr. 1·10) and then diluted to 100 c.c., heated, and a rapid stream of hydrogen sulphide gas passed through it for 15 minutes.

The precipitate is filtered off and dissolved through the filter paper with 40 c.c. of concentrated hydrochloric acid. A small amount of potassium chlorate is added, the solution heated and any separated sulphur filtered off. Two drops of a ferric chloride solution are added to the solution, which should be at a temperature of 90° C.; stannous chloride is then added drop by drop until the yellow colour of the ferric chloride just disappears. The whole solution is then poured into a beaker containing 10 c.c. of a saturated solution of mercuric chloride, diluted to 500 c.c. to 600 c.c. and titrated with potassium permanganate to the first pink colour. The stannous chloride reduces antimony in a hydrochloric acid solution before it will reduce arsenic. The mercuric chloride takes up the excess of stannous chloride.

The successful use of this method demands a close adherence to the details. The precipitation of the antimony should take place within 15 minutes, if the bulk of the solution is not over 75 c.c., and the acid between 1·1 and 1·15 sp. gr. The potassium chlorate oxidises the arsenic to the higher form. The reduction by stannous chloride is best made with a strong and a weak solution, using the strong solution to accomplish most of the reduction and the weak to finish with.

The amount and strength of the ferric chloride should be known in terms of the permanganate. The reduced solution should be poured immediately into the mercuric chloride to prevent the reduction of the arsenic. Silver, lead, bismuth, arsenic and tin do not interfere. Copper should be removed. The time consumed for a single determination is not more than 1½ hours. The method has given good results in comparison with other well known methods."—*Engineering and Mining Journal*, July 31, 1909, p. 209. (A. R.)

Rapid Analysis of Aluminium.—"A process for rapid analysis of aluminium was described at the meeting of the French Academy of Sciences, Aug. 2.

The inventor, Mr. Kohn Abrecht, reduces the metal to the form of very fine filings. Then, after placing it in a tube, hydrogen is run through, which expels the air; and, finally, hydrochloric acid, which forms aluminium chloride. The latter is passed into a flask for analysis."—*London Mining Journal*, Aug. 28, 1909, p. 276. (A. R.)

Metallic Tellurides.—"1. Sodium telluride, Na_2Te, is a colourless crystalline salt and is highly hydrated.

2\. Tellurium dissolves in sodium telluride solution with the formation of sodium polytelluride. Beyond the proportions represented by Na_4Te_3, tellurium is not soluble in sodium telluride. Na_4Te_3 does not crystallise as such, but breaks down upon evaporation into Na_2Te and tellurium.

3\. Potassium telluride, K_2Te does not crystallise from aqueous solution, but is precipitated from strong solution by alcohol.

4 Telluride of zinc is yellow when freshly precipitated and brown when dry. Its composition is $ZnTe.H_2O$. The anhydrous telluride is dark red in colour.

5. Cadmium telluride, $CdTe$, is precipitated as a maroon coloured substance, almost black when dry.

6. Nickel forms a telluride of the composition $Ni_2Te_3.4H_2O$, black. When heated in hydrogen it yields $NiTe$.

7. Cobalt forms a telluride of analogous composition and similar properties to that of nickel.

8. Lead forms the telluride $Pb_2Te_3.4H_2O$, which is decomposed by heat into $PbTe$, water and tellurium.

9. Ag_2Te is precipitated by sodium telluride from silver acetate solutions.

10. Copper forms the two tellurides $CuTe$ and Cu_2Te_3 by precipitation with Na_2Te and Na_4Te_3, respectively.

11. A telluride of arsenic is formed by precipitation, and is soluble in excess of Na_4Te_3 solution. This indicates the existence of a telluro salt of arsenic analogous to the sulpho salts.

12. Mercuric telluride is precipitated from mercuric chloride solution, but reacts rapidly with excess of mercuric chloride, forming mercurous chloride and tellurium chloride.

13. Sodium telluride precipitates gold from auric chloride solution and platinum from platinic chloride solution, the tellurium dissolving as the tetrachloride."—C. A. Tibbals —*American Chemical Society*, Aug., 1909, p. 912. (A. McA. J.)

METALLURGY.

Electro-Chemical Plate Amalgamation.—"The electro-chemical system of amalgamation is designed to extract from the sands or pulp all gold values not encased. It is entirely a new system of amalgamation, designed to do the work of the ordinary mill plate, including the treatment of slimes, and, at the same time, to extract values which cannot be saved by the usual mill practice, and which otherwise would require cyanide treatment ; to do all this in one single, inexpensive and continuous operation, requiring practically no more outlay for installation and maintenance than the usual mill plate. The system also furnishes a method by which the Alaska beach deposits, black sands, hydraulic, placer and dredge tailings, mill tailings, desert deposits and many low grade propositions can all be profitably worked.

'It may be sometimes necessary to grind ore or sands to 100, 150 or even 200 mesh before the values are released, but after this has been done the pulp is simply passed over the amalgamating device, and the electro-chemical forces automatically extract the values to such an extent that usually no further treatment is profitable. In ordinary cases values are recovered at 80 mesh. In certain cases where the ore is unusually refractory, base or complex, it may be be necessary to treat the tailings for hour or two by the continuous electro-cyanide process, which is an exceedingly simple and inexpensive operation, but withal very efficient.

'The electrolytic method of amalgamating calls for a sluice box of suitable width, the length being 40 ft. or 50 ft., or even longer, according to conditions. In the bottom of the box is an ordinary copper plate acting as a cathode. Graphite anodes of suitable construction are arranged to conduct the low voltage current into the water (electrolyte) passing over the device. Pulp of proper fineness, screened gravel or sand, is passed into the apparatus by suitable means. Certain solutions are also added to the water. A critical low voltage current of proper amperage is passed into the anodes ; the water, carrying pulp, etc., acts as an electrolyte, and makes the electric connection with the copper plate (cathode). In operation, the water is at all times electrified, and the copper plate is also constantly under electrical action. The device can be made of any desired capacity."

The current density used is about five-tenths ampere per square foot of cathode surface ; the pressure at the generator is from 5 to 10 volts. The solutions added to the water contain mercury and sodium salts and some salts from which hydrogen is easily liberated by electrolytic action. The mercury solution is added to the water passing into the apparatus, the amount depending on the values in the pulp. About the same amount of metallic mercury as is used in ordinary amalgam practice is necessary ; practically all of it is recovered in the amalgam.

Several chemical, electrical and electro-chemical processes are in constant operation in the amalgamating box, the reactions which would ordinarily occur being greatly intensified by the presence of the electric current. The most important part of the process is the even, constant and uniform deposition of mercury on the cathode plate by the electrolysis of the mercuric solution. The coating of mercury is chemically pure, it adheres tenaciously to the plate and has no tendency to 'flour.' As the mercury is liberated in its nascent state, its affinity for metals and gases is much greater than in ordinary practice ; its amalgam-forming powers are intensified to such a degree that rusty, greasy or pyritic gold is readily amalgamated. 'Fouling' or 'sickening' of the plate is impossible.

Fouling of the plates is prevented by the liberation of nascent hydrogen, the cleansing and reducing powers of which are used to great advantage in electrochemical amalgamation. It destroys instantly organic, greasy and foul conditions, counteracts all impurities in the water, and cleans rusty gold. Large quantities of hydrogen are occluded in the mercury as the latter is deposited and exercise continuously a powerful cleaning and reducing effect. Metallic sodium, liberated from the sodium salt, unites with the mercury to form sodium amalgam. The conditions of its formation ensure its purity. With the hydrogen occluded in the mercury it forms hydrogen-sodium amalgam, the most powerful amalgamating agent known. Hydrogen-sodium amalgam besides holding free gold in any condition, will also hold platinum. It will amalgamate all metals not encased, which are in a comminuted state. Amalgamation takes place as soon as a particle of gold touches the quickened mercury surface; the larger particles will come in contact with the mercury very soon after entering the sluice box ; microscopic values will require possibly 40, 50 or even 60 ft. of cathode surface. Experiment has demonstrated that electrolytic amalgamation will recover from 95 to 99% of all gold not encased, and in many cases the extraction reaches 100%. Usually when the pulp of refractory ores is ground to 100 to 200 mesh, the tailings, after passing the electric amalgamator, do not contain enough values to make supplementary gold-saving devices necessary."—Elmer E. Carry, *Mining Journal*, June 12.—*Engineering Magazine*, July, 1909, p. 839. (J. A. W.)

Electrolytic Refining of Gold.—"The separation of silver and other impurities from gold by electrolysis in a solution of hydrochloric acid was suggested by Charles Watt of Sydney in 1863. He proposed to apply the method at the Sydney Mint, and claimed to have discovered means of preventing the evolution of chlorine from the anode, but no steps were taken to test his process. In 1878, Wohlwill of

Hamburg introduced his process at the Norddeutsche Affinerie, and it has since been applied elsewhere in Germany. It displaced the sulphuric acid process at the Philadelphia Mint in 1902, and at the Denver and San Francisco Mints in subsequent years, and is in process of being installed at the Ottawa Mint. It has made little progress in other countries.

In the Wohlwill process the electrolyte is a solution of gold chloride containing from 25 to 60 gm. of gold per litre, and from 20 to 100 c.c. of concentrated HCl per litre. It is heated to a temperature of about 60 to 70°. The anodes consist of impure gold, containing not more than 60 or 70 parts of silver per 1,000, and the cathodes of pure sheet gold. The anodes dissolve partly as trivalent gold, forming $HAuCl_4$, and partly as monovalent gold. The AuCl formed at the anode is partly decomposed into $AuCl_3$, and metallic gold, which falls to the bottom, but some of the monovalent gold ions reach the cathodes. The higher the current density the more trivalent gold is dissolved and the less AuCl is formed, so that there is less gold in the slime at the bottom. The current employed in the United States is from 300 to 600 amperes per square metre of anode-surface. The deposition of gold at the cathodes is greater than the amount dissolved at the anodes, owing to the presence of impurities in the unrefined gold, so that it is necessary to add chloride of gold to the solution at intervals. Silver is converted into AgCl at the anodes, and falls to the bottom of the vessel, or is in part dissolved. The anodes may contain up to 150 per 1,000 of silver, but if such a large proportion is present it is necessary to scrape the silver chloride from the surface of the plates. Platinum is dissolved with the gold and accumulates in the solution, from which it is recovered in the ordinary way. The deposited gold is nearly pure.

The electrolyte is circulated by mechanical means, and is renewed when it has been made foul by the accumulation in it of copper, palladium, etc. Lead makes the deposited gold impure, but may be rendered insoluble by the addition of sulphuric acid.

Apart from observations on the technical details of this process, the points on which information is particularly required are: (1) the time required to complete the process. This is most important, because of the loss of interest on the gold under treatment. The chief objection to the application of this process in England is the fear that the process is too slow. (2) The cost for labour, power, etc. (3) Whether, if speed is unduly increased, the gold is likely to be of bad quality, contaminated with lead or other elements which would make it brittle."—T. K. ROSE. —*Mining and Scientific Press*, June 26, 1909, p. 890. (A. McA. J.)

THEORY OF THE SETTLEMENT OF SLIME.—"A paper on the settlement of slime, by H. S. Nichols, appeared in the *Mining and Scientific Press*, July 11, 1908. From the study of clay-working slips I can add information from a different view-point. In most respects what I submit is supplementary.

In that article no reference was made to the viscosity of the liquid medium. Viscosity is defined as the measure of the internal friction of a fluid. The viscosity of water is commonly taken at a given temperature as unity. In Fig. A is reproduced graphically the variation of the viscosity of water with the temperature. The viscosity decreases as the temperature rises. As the data given in Landolt and Börnstein's tables extend only to 70° C., it was necessary to extrapolate to get the value for 88° C., the highest temperature used by Mr. Nichols. The settling curves for various temperatures are practically straight lines, so that the intersections of the ordinate for any number of millimetres settled should be in about the same proportions. Taking the 140 mm. ordinate, the following table has been prepared :—

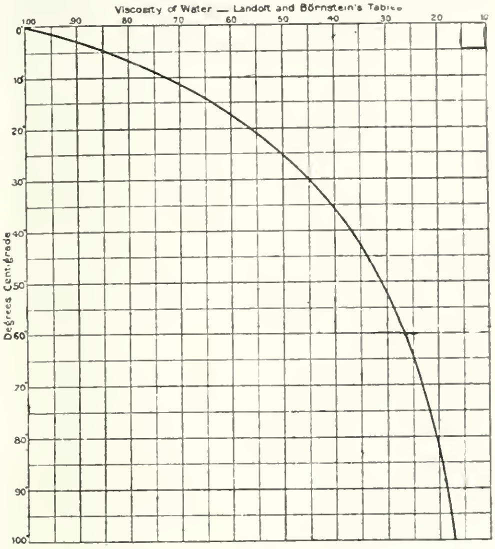


Fig. A.

Temperature, degrees Farenheit ...	60	100	140	190
Temperature, degrees Centigrade ...	16	38	60	88
Viscosity of Water from Fig. A ...	62	38·4	26·9	18·4
Viscosity of Water multiplied by 3..	186	115·2	80·7	55·2
Minutes settling 140 mm. curves for heated liquids ...	194^1	110^2	82^3	50^4
Minutes settling 140 mm. curves for heated and cooled liquids	185^7	140^6	80^5	

Superior figures indicate numbers of curves.

From this table it appears that, within the limits of experimental error, the rate of settling is directly proportional to the viscosity of the water, whatever the temperature corresponding. According to Fig. 1, 5, and 6 of Mr. Nichols' paper, the higher the percentage of solids in the pulp the slower the rate of settling. In Fig. B., which gives the results of experiments by A. V. Bleininger (Trans. American Ceramic Society, 10, 394, 1908), it is shown that the viscosity of a clay-slip increases more and more rapidly as the clay-content increases; consequently the diminution in speed of settling shown in Fig. 1, 5 and 6 of Mr. Nichols' paper can be attributed to increase of viscosity with concentration. As clays differ in viscosity at equal concentration of slips, different slimes should also be expected to vary. Consequently, a treatment adapted to one slime will require its proportions to be altered somewhat to apply to another slime.

To explain the other figures, it is necessary first to discuss the various conditions of matter that may be present in a slime. The coarser particles are crystalloid, and a relatively dense and hard as compared with the finer matter. Upon and surrounding the crystal grains is an enveloping film of matter in the 'gel' form of the colloid condition. It resembles gelatine in properties, absorbs water, and shrinks on drying, becoming hard and cementing the grains together. It is less dense than the crystal grains, and will settle slower than them if separate.

If a large excess of pure water be used, or a proper proportion of a suitable solvent, like ammonia, soda ash, or caustic soda, the gel is dissolved and forms a turbid suspension which will not settle clear. This is called the 'sol' form. In most cases it is the soluble sodium, ammonium, or potassium compound of a very large and complex polymerised molecule, in which the sodium or other part is an exceedingly small part of the whole. This is illustrated by cases 1a, 1b, 2a, and 2b of Fig. 4 (from Mr. Nichols' paper) If to such a sol, sulphuric acid be added, the acid takes away from it the base that made it soluble, and the gel form of the colloid is precipitated and will settle. If barium chloride or lime is added the reaction is something as follows:

$$2\text{Na with sol} + BaCl_2 = \text{Ba with gel} + 2NaCl.$$

The insoluble barium or calcium compound is formed, and will settle readily. If salt (NaCl) is added, it merely 'salts out' the sol; that is, when salt comes into the solution the sol has to go out of the solution until a solubility-equilibrium is reached. In Mr. Nichols' experiments, as he has indicated, nearly equal results were obtained by the following:

	H_2SO_4.	$BaCl_2$.	CaO.	NaCl.
Mg per 200 c.c. pulp containing 2·5% solids ...	16·68	83·32	55·54	166·90
Chemically equivalent proportions used ...	0·34	0·77	1·98	2·89

On figuring to a basis of chemically equivalent proportions, it is seen that salt is much less effective than the other chemicals as a cause of coagulation. The lime does not show up very well on the equivalent basis, for a reason to be explained further on.

The fourth condition of matter is that of true solution, into which form the various soluble salts must be brought in order to act as precipitants. Now matter in all four of these conditions is acted upon both by gravity and by electrostatic repulsion. The repulsion is between particles having charges of the same sign. In the case of true solutions, the size of the dissolved molecules is so small that the electrostatic repulsion far outweighs the effect of gravitation, and the particles are uniformly diffused throughout the liquid. The sol is the first step in aggregation; the particles are large enough to reflect light, and while electrostatic repulsion still predominates, the effect of gravity is to make the upper layers of the liquid somewhat less concentrated than the lower. The gel and the crystal are the irregular (or amorphous) and the regular forms, respectively, of the solid state. The opposing effects of gravity and viscoscity largely predominate, but electrostatic repulsion is partly responsible for the slow settlement of the finest grains.

It was shown above that electrolytes may promote sedimentation by forming insoluble compounds. A second way in which they may accomplish the same object is by decreasing the amount of OH ions present in the liquids. Both OH ions and all forms of clay particles in water are negatively charged. The OH ions appear, with most clays, to have a marked effect in causing the clay colloid to assume the sol form; while a depression of the number of OH ions tends to coagulate the sol and form the gel. The reason for the comparatively weaker action of lime (CaO) as a settling agent is that, while it forms an insoluble compound, it also increases the OH concentration of the liquid, two opposite influences on sedimentation. Sulphuric acid, on the other hand, not only destroys the soluble compound, but it also decreases the OH concentration. With 4% H_2SO_4 the viscosity is about 6% greater than that of water, but these other effects predominate. Sulphuric acid is therefore a most effective coagulating agent.

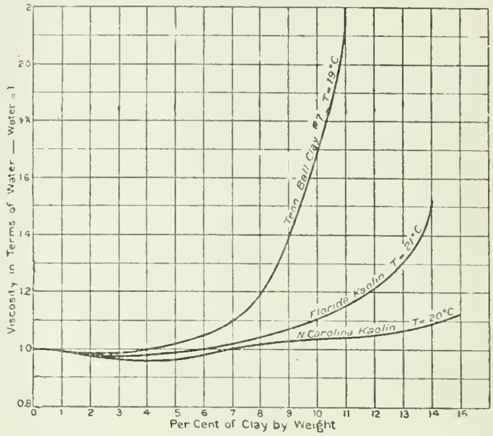


Fig. B.

Mr. Nichols' fourth deduction, that close packing of the clay particles will render the electrolyte inefficient and finally of no avail, is not exactly in accord with clay-working experience. Carbonate of soda (Na_2CO_3) is commonly added to clay slips, with the result that they require about half as much water to be capable of pouring as would be the case without the addition. Addition of acid sufficient to neutralise the soda stiffens such a slip again. This example is, however, with more dense slips than in the cases considered by Mr. Nichols."—H. E. Ashley.—*Mining and Scientific Press*, June 12, 1909, p. 831. (W. A. C.)

The Chile Mill.—"The comparison between the two styles of mill, Chile v. stamps and tube mills, is shown in the following table.

Comparison of Chile Mills and Stamps and Tube Mills.

	5 Chile Mills.	25 Stamps, 2 Tube Mills.
Capacity to 150 mesh ...	100 tons	100 tons
Cost of mills and power plant	$24,000	$17,000
Weight of mills and power plant	195 tons	127 tons
Power required	46 h.p.	95 h.p.
Wearing parts renewed...	every 2 years	every 3 mons

The figures given in the table are based on cost of ironwork without framing, as the latter would be about the same in either case. The tubes figured were 3·5 × 14 ft., and the stamps 1,050 lb. Among the points not covered in the table is head room, the loss of of which is much less with the Chile mill, thus giving greater latitude in choice of sites. Less actual ground space is also required. Supporting framework, building and erection costs, including a simple hand crane for the Chile mills are about the same in either case, while depreciation, attendance (skilled as well as unskilled), accessory equipment, breakages and probable delays are all decidedly in favour of the Chile mill. The best quality of the stamp mill, its ability to withstand rough handling, is possessed by the Chile mill. Neither over-feeding nor under-feeding has any effect on its continuous operations, these conditions merely reducing capacity temporarily."—M. R. LAMB.—*Engineering and Mining Journal*, June 12, 1909, p. 1182. (A. R.)

TUBE-MILL LINERS.—"The following points have to be considered in connection with liners for tube mills:

(1). They should be comparatively thin, so as not to take up much room in the mill.

(2). They should be so arranged that there is no slip of the pebbles on their surfaces.

(3). They must be easily put into the mill, require no attention, and be readily renewed.

(4). They must show economy as regards wear.

The 'ribbed liner' here described has all these features.

This method of lining tube mills has been well tested at the Komata Reefs and Waihi Grand Junction batteries, and has given great satisfaction.

The quartz at the Komata mine is very hard, and very coarse screens are used on the battery (wove-wire screens, with three holes per linear inch), so the tube mills are running under very severe conditions. At present two mills, each 16 ft. × 4 ft., lined with 'ribbed liners,' are handling 100 tons per day of the 3-mesh product, and grinding it down to a very fine condition. The grades before and after grinding were respectively :—

On 6-mesh.	10	20	40	60	90	120	200	Passed 200
%	%	%	%	%	%	%	%	%
6	14·5	18	13	10·5	13	4·5	4·5	16
—	—	—	—	1·5	16	15·	8·5	59

This high efficiency of the mills is chiefly due to the system of liners, as the ribs are so arranged that there is no slip of the pebbles on the liners, and the cascade of pebbles, due to the revolving of the mill, is well developed. This is of the utmost importance in running tube mills. The mills at Komata are run with a 'returning box' in connection with them. This box is of the 'spitzkasten' type, and automatically returns the unground particles to the mills, the furnished product overflowing and going to the treatment plant.

The wear of pebbles and liners in the two Komata mills is 3 cwt. to 4 cwt. per week for each 16 ft. × 4 ft. mill, and totals 1·56d. per ton of sand.

A comparison between 'ribbed liners' and silex liners of the most improved type was made at the Waihi Grand Junction mine. The ore at this mine is not nearly as hard as at Komata. During fifteen weeks with the silex liners, 8,430 tons of sand were ground in the mill. The cost for liners was 0·66d., and for flints (13 tons 17 cwts., valued at £38 1s. 9d.), 1·08d. = 3·6lb., making a total of 1·74d. for flints and liners per ton of sand ground.

During the fifteen weeks with the 'ribbed liners,' 8,680 tons of sand were ground. Very careful measurements showed that the life of a set of liners, allowing one set of angle shoes for renewals, would be eighty weeks. The liners are of hard cast iron, and the estimated cost per ton of sand works out at 0·42d. The flints used during the fifteen weeks came to 5 tons 12 cwts., which worked out at 0·42d. = 1·44 lb. per ton of sand, making a total of 0·84d. for flints and liners per ton of sand.

The above tests show about the same tonnage in each case, as this was the total ore being crushed by the stampers during the test, but during the run with the 'ribbed liners' the grinding was very much finer than with the silex liners, the result being a reduction of 1s. 6d. per ton in the value of the sand tailings.

Owing to the 'ribbed liners' being comparatively thin, the average working-area of a 19 ft. × 4 ft. 10 in. mill is about 20 per cent. larger than with silex liners, and this, combined with the lifting action of the ribs, increases the grinding efficiency of a mill to a remarkable degree.

The arrangement suitable for a mill about 4 ft. diameter is eight ribs. It is of great importance that the ribs be correctly spaced and of correct size; roughly, they should be about 18 ins. apart, and a 4 ft. mill requires eight ribs, and a 5 ft. mill ten ribs. The ribs for a 4 ft. mill are 3 ins. × 2½ ins., and for a 5 ft. mill 3½ ins. × 3 ins. Another very important feature with these liners is the speed of the mill, as this should be slower than with the thick, smooth-surface liners, such as silex or concrete.—F. C. BROWN, *New Zealand Mines Record*, *London Mining Journal*, June 26, 1909, p. 795. (A. R.)

SCALING OF BATTERY PLATES.—"The high extraction recorded by Mr. Goddard was possibly due to the fact that the plates had been previously silver-plated, which prevents, to a certain extent, the absorption of the gold by the copper; and the amalgam was simply a cake on the surface and might have been cleaned-up almost as completely by the use of steel scrapers, without hammering the plates. On the other hand, had the plates been originally plain annealed copper, there would probably have been, at the time of scaling them, a much larger quantity of gold absorbed deeply into the copper, and a very small percentage would have been recovered by the method described. Consequently, if it is desired to recover the gold from annealed plates and avoid the expense of shipping them to a smelter, it becomes necessary to treat them by a more effective process. Below I give details of one by means of which practically all the gold is extracted from the plates without materially affecting them; in fact, in comparison with new ones the plates are perhaps improved, as they become more porous and therefore more absorbent to mercury, than before treatment. I give the following minute details of procedure because I have never seen the method referred to in any book :—

No. 1. After cleaning up in the ordinary way by means of a squeegee and scrapers, etc., the plate is taken up and well washed, after which it is heated all over evenly at a temperature just sufficiently high to eliminate all the mercury.

No. 2. While still warm, coat the upper surface with a mixture of finely powdered sal-ammoniac and hydrochloric acid (the mixture should be almost of the consistency of a paste and may be applied conveniently with a 3 in. paint brush).

No. 3. Then expose the plate to a moderately high temperature until evenly red hot all over.

No. 4. While red hot plunge suddenly into a tank of cold water, when nearly all the gold comes away in the form of scales, ranging from small particles up to pieces of 2 in. diameter. About 10 % adheres to the surface of the plate, but is easily detached by chipping with a suitable edged tool.

The process is very simple, and, with convenient means of heating the plates and a suitable tank, I believe a plate could be taken up, scaled and laid for re-amalgamation in about an hour.

To avoid the necessity of bending the plate the tank should be of a greater area than the plate and about 1 ft. in depth, and should have a smooth bottom to facilitate the collecting of the scales.

The scales are black in colour and contain possibly about 25 % copper, which is removed by boiling in dilute nitric acid in an enamelled cast-iron mercury bucket. The liquid is decanted, and the operation is repeated three or four times until the acid shows only faint signs of copper in solution, after which the scales are washed, dried and smelted.

Some years ago I superintended the scaling of eight large battery plates from a 20-stamp mill in the vicinity of Boksburg, Transvaal. The operation was conducted under considerable difficulties on a wood fire in the open, and with a tank which required the bending of the plates to receive them. In spite of these inconveniences the work was accomplished within 24 hours, and most of the time was taken up between straightening the plates out again and annealing them.

I do not happen to have kept notes recording the exact extraction of the gold corresponding with those given by Mr. Goddard in his paper, but the result obtained was very satisfactory, and the fineness of the gold was found to be about 5 % higher than the average of that which had been yielded from the regular clean-ups extending over a few years, and I believe this is invariably the case when the scaling is done by this method.—J. E. Breakell, *Institution of Mining and Metallurgy, Bulletin No. 57*, June, 1909, p. 39. (H. A. W.)

MINING.

Acetylene Lamps Underground.—"Acetylene lamps, according to W. F. Slaughter (*Proc.*, Lake Superior Mining Institute, vol. xiii., p. 206), consume about one-fifth as much oxygen as candles do when giving the same amount of light. On that account, when it is necessary to work in badly ventilated stopes or headings, acetylene lamps should be used, not only because they consume less oxygen, but also because they give a bright light in air in which it is difficult to make a candle burn. Acetylene lamps are now made small enough to be worn on a miner's hat; they are used at the Saginaw mine on the Menominee range, Michigan. While the lamps require some attention in order to keep the water feeding properly to the carbide, the miners soon learn how to care for them. The lamps then give little trouble. It is said that it costs about two-thirds as much for light when acetylene lamps are used as against candles. At the Michigan mine the cost of the carbide required to last a 10-hour shift is about 2 cents."—*Engineering and Mining Journal*, Jan. 16, 1909, p. 177. (C. B. S.)

Loading Blast Holes and Driving Small Drifts.—"Contributors to recent issues of the *Journal* appear to hold widely divergent views as to the proper method of loading up the drill holes. As to the proper place for the detonating cap, I am certain that Mr. Adkinson is correct in stating that a hole shows more shattering in the vicinity of the detonating cap; this is no reason, however, for placing the cap in the bottom of the hole.

As everyone knows, dynamite is detonated from a solid into a gas by pressure. When the detonating cap lets go it compresses about 2 cub. in. of the surrounding mass of dynamite sufficiently to cause dissociation; the transition from solid to gas must occupy a definite interval of time, about one-twenty thousandth of a second. The initial explosion of 1 or 2 cub. in. of dynamite in the immediate vicinity of the cap would generate sufficient force to compress the entire charge to the dissociating pressure.

Detonation of a mass of black powder is effected by a uniformly expanding zone of flame starting from a spark. With dynamite detonation is accomplished with a succession of jolts separated by imperceptible but definite intervals of time. This explains why a strong cap and a well packed charge give definitely better results. The strong cap detonates enough of the surrounding charge to make a strong primary explosion. When the charge is well tamped the primary explosion would compress 2 ft. or 3 ft. of powder to the pressure of dissociation. With a weak cap and poorly compacted dynamite, it is conceivable that it might take three or four jolts to explode the whole charge.

The reason the ground around the primer is badly shattered is that the primary explosion of an ounce or so of dynamite has not power enough to spring the rock burdened on the hole, and as the tamping is tightly up against the explosive, the pressure crushes and shatters. In the interval that precedes the detonation of the main portion of the charge the gas from the primary explosion blows the tamping back an inch or so and cushions the main explosion to a certain extent. For these reasons I believe a cap should be in the top of the charge, because it is evident that the main or secondary explosion tears the rock out. With a cap on top a small part of the upper end of the charge is rendered useless for rock breaking purposes; with the cap in the bottom a small part of the bottom of the charge is consumed in shattering the bottom of the hole. With a lifter or back hole it would not make a particle of difference, as far as breaking ground went, where the cap was; with a cut hole, where the power is needed on the toe of the hole, a cap on top of the charge will tend to put the locus of the principal concussion a fraction of an inch nearer the bottom of the hole.

Tamping.—In breaking rocks with mud-caps, I find it makes a very material difference whether or not the detonating cap is placed on top of the dynamite. The most effective way to break a rock is to pick out a little depression and form a wall around it with plastic clay; fill this crater with dynamite, lay the cap on the centre and cover the whole with soft clay. If the cap is shoved down into the dynamite the breaking power of the mud cap is lessened. The only possible explanation of this phenomenon is that when the detonator is near the bottom of the dynamite, the detonation lifts a portion of the explosive bodily from the rock before it bursts into gas. High explosives must have a good contact with rock which it is desired to break. When a mud cap is placed on top of a stone, the explosive is usually plastered down tight against it, and the stone is broken into several fragments; if the same quantity of explosive were poked under the stone it would have poor contact and do very little execution. This is the reason why dynamite is supposed to strike down.

In regard to tamping a hole, I think that ordinarily 8 in. or 10 in. of clay tamping is enough. Strong cut holes require more tamping to get the full power of the explosive. Take the case of a 6 ft. incut hole in the face of a drift, the hole being $1\frac{3}{4}$ in. in diameter and loaded with 32 in. of 60% explosive. The centre of the charge would be burdened 4 ft. and in hard granite or trap it would take one two-thousandth of a second for the concussion to travel from the explosion to the free face. This means that for a period of one two-thousandth of a second the gases of the explosion must be confined. At the end of the period, the momentum imparted to the rock tears it away from the solid. During this period the tamping will be subjected to a pressure of 450,000 lb., sufficient to force 18 in. of clay or 3 ft. of water tamping 6 in. This shows that 18 in. of clay tamping is none too much. A 6 ft. hole burdened 30 in. would kick 8 in. of clay tamping back 6 in. before the rock cleaved from the solid.

A quick way to tamp up a round of holes is to have one of the muckers make a candle box full of clay slugs 1½ in. in diameter and 4 in. or 5 in. long. In loading twist a piece of paper around one end of the slug and they are easily shoved in and rammed home with the loading stick. I think the best results are obtained by putting in 18 in. of tamping for the incut holes and 14 in. for the other holes. Using a 2 in. time interval on the fuses, the fire in the fuse will be several inches in the tamping before an adjacent hole goes off. It is no uncommon thing for a hole to kick off 2 ft. or 3 ft. of another hole; as long as the fuse was burned down into the tamping and firmly bedded in a foot or so of clay, the spent fuse would break in case a collar blew off; otherwise the fuse might be pulled out of the hole. I noted a case in which one hole cut another hole off in the middle of the powder charge; the hole had pulled back to a slip and the dynamite was cut so smoothly that there was no sign of a hole on the slip until one of the men stuck his pick into it.

This cutting off of holes is a prolific source of accident. If the holes are overloaded as they often are, powder is thrown in the muck. Of course, there is no method of blasting that will eliminate accidents altogether, because things are liable to happen in blasting a round of holes that no man can foresee.

Drifting and Crosscutting.—In driving drifts and crosscuts where there are no good slips or floors to break to, the 'V' or pyramid incut will be found to be the most satisfactory. In narrow drifts four holes will be enough and they should be drilled so that the points nearly or quite intersect. In drifting, I think the double-screw column is much handier than a cross-bar, although it takes longer to rig up on a column than it does on a bar. In uneven ground, the machine often runs up or down on a hole; with a column and arm it is possible quickly to adjust the machine for alignment and often prevent the steel from fitchering and running tight. In very tight, hard ground, it pays to set the column as close to the face as the machine will work; drill the holes about 5½ ft., and give them as much lift as possible. Of course, when the cut holes are drilled with the machine close in, the column has to be rigged back about 18 in. to drill the back holes and lifters.

If the ground is only moderately hard it pays to drill the cut holes so as to bottom 1¾ in; then 4 lb. of 60% or 75% dynamite can be put in the bottom 30 in. of each hole. If the ground is extremely hard, time can be made by drilling the holes smaller, bottoming them 1¼ or 1⅜ in. Not only do small steels cut faster and hold their gauge better, but it is also far easier to collar a hole on a slanting face with a small starter.

When small holes are used it will be necessary to spring the four cut holes with about four 1¼ in. sticks of 60%, then blow them out well with a blowpipe; this will chamber the holes so that they will take enough powder to throw out the incut. It takes about 40 minutes to spring four holes, (15 minutes to tamp up, 15 minutes to get the smoke out of the drift, and 10 minutes with the blowpipe to clean and cool the holes.)

Order of Fire and Misfires.—In most cases it is a decided avantage to blast the incut with a battery, although it takes more time than if the holes are all loaded and fired in one round. When fired with a battery the cut generally kicks the collars of a few of the surrounding holes, so that after the incut round there is not the same danger of holes cutting one another off and causing misfires and scattering powder in the muck. Again if the cut does not come out well the bottoms can be reloaded and split with the squaring holes. When the incut is blasted separately, there is not much danger of the machinemen lighting holes out of their proper order and perhaps spoiling the whole round.

If a hole that is solidly tamped with about a foot or so of clay tamping misses fire, it is certainly no joke to dig the tamping out. If the caps are always soaped, and waterproof fuse is used, such mishaps are bound to be extremely rare. One method of drawing tamping that is a little safer than picking it out with a scraper, is to burn it out by repeatedly loading up whatever collar there is to the hole; after two or three shots the concussion will jar through the tamping and set off the main charge. Another method, which I believe is safer, is to make a little tank by capping both ends of a 6 in. or 8 in. pipe about 3 ft. long, tapping in a couple of inch nipples on opposite sides of the pipe. This tank would hold two or three pails of water. To blow tamping use a small tip on the blowpipe, wedge a plank or board across the drift, and after inserting the blowpipe in the hole, lash it fast to the board so that it cannot kick back. Now run the blowpipe hose to the water tank and run the machine hose from the air pipe to the upper nipple of the tank. Use a valve 200 ft. or 300 ft. back from the face for turning on the air pressure. This method works well in an upper or flat hole; it will often wash out a foot of tamping without shifting the blowpipe."—G. C. McFarlane.—*The Engineering and Mining Journal*, Jan. 23, 1909, pp. 225-6. (C. B. S.)

A Selective Electric Fuse Spitting Device.—"Spitting holes in shafts, especially in wet ground, is hazardous work, and the danger has been greatly increased by the introduction of electric hoisting. With electric-power systems the loss of power is sudden and without warning, while in the case of air or steam there is a gradual dying away of the power. Hence, there is danger of the failure of the electric power between the blasting signal warning the engineer to be ready to hoist and the final signal for hoisting the miners away from the blast.

Since even the best of electric-power systems is liable to fail, especially when the transmission line is long, the solving of the problem of blasting without the attendant danger of being caught when the holes are fired, has been given much study. Of course, simultaneous blasting by electric current is possible, but that is not desirable in shaft work where timbers generally have to be kept close to the bottom and where the cut holes must be fired first to get the best results.

The device described in the following paragraphs was perfected at the Hecla mine, in the Cœur d'Alene district, Idaho, for selective firing of holes from a distance by means of electric current. In sinking a big three-compartment shaft at that mine, the air plant being taxed to its full capacity, the hoisting had to be done from the main plant, which was operated by electric power generated at Spokane

Falls, Washington, over 100 miles distant. The frequent temporary interruptions in the transmission of the power got on the nerves of the shaftmen, for they feared that the power might fail when they were spitting heavy blasts in the shaft. The result was that the company electrician devised a plan of spitting the fuse at a safe distance so as to eliminate this danger.

One Miss in a Thousand Shots.—The first device used was not satisfactory, but by rebuilding it and using a higher voltage, P. C. Schools, electrician at the mine, has succeeded in bringing the machine to such a state of perfection that the misses amount to only 1 in 1,000.

The perfected system consists of a firing board, where the operator tests his circuits and 'spits' his holes in the order desired; a reel, on which is wound the cable carrying the wire used in spitting; and firing blocks attached to the end of the cable. The holes are charged and primed in the usual way, and the spitting wire is inserted in a slit cut in the fuse near its end. The fuse is wrapped tightly with electrician's tape, and thoroughly coated with axle grease, so that the juncture is practically waterproof, and the spitting can be done successfully under water.

The spitting ends are all prepared before going into the shaft, and the fuses are all cut the same length, as the operator gives the time interval between holes when he inserts the plug at the firing board. Each fuse has two leads of a spitting wire projecting from its end. The cable containing the wires with the attached firing blocks, which is kept on a reel in the station on the next level above the shaft bottom, is now lowered to the bottom, and the two No. 16 annunciator wires projecting from the fuse of the first hole to be fired are securely wrapped around the two heavy copper leads of block No. 1. This gives hole No. 1 direct connection with the firing board on the level above. Holes Nos. 2, 3, 4, etc., are then attached to the numbered blocks in the order in which they are desired to explode.

The system is designed for a 24-hole round, which, of course, can be used for fewer holes if desired; the number of holes can easily be increased, but that rarely would be necessary.

Tests Insure Ignition of the Fuse.—When all the holes are ready to be fired, the men are hoisted to the firing station, and the circuits are tested out. To test the circuits, the main line switch is closed and care taken that the single-pole firing switch is open, for it is impossible to spit a fuse unless the firing switch is closed. This firing switch is kept in a box under lock and key, and only one man on each shift has a key to open it. The flexible cable and plug is then inserted into each of the holes in the firing board numbered to correspond to the holes below to be fired. If the circuits are closed, and ready to be fired, the lamps at the top of the board will light. If the lamps should not light, then there is something the matter with the circuit that must be remedied. If all the circuits tests closed, then the shots are ready to be fired.

To fire the shots, the main line switch is closed, the firing-switch box is unlocked, and plug inserted into No. 1, the lamp lights, the firing-switch is closed. This short circuits the lighted lamps causing them to go out, and at the same time applies 440 volts directly across the No. 26 tinned iron wire in the fuse at the bottom of the shaft. This wire melts with a blinding flash, spits the fuse, and burns itself free. The firing switch is then opened immediately, so that if an arc is maintained at the fuse it will be smothered by the cutting in of the lamp resistance. With the plug still in No. 1, and the firing switch open, failure of the lamps to light indicates that the spitting wire at the bottom did its work, and the fuse is now burned; but if the lamps again light up brightly, it indicates that the fuse did not spit, and that the firing switch must again be closed. It is seldom, if ever, that the firing switch has to be reclosed.

Little Additional Time Required.—The operator then allows his time interval—a few seconds between the holes—which in most cases is simply time enough to change his plug to the next hole. He then proceeds with second hole as above described, retesting the circuit-firing resistence to see if the operation was successful, and continuing until all the holes are spit. The melting of the fuse wire leading to each hole disconnects the firing blocks so that the lower end of the cable is free, the upper end is then detached from the firing board, the cable wound on to the reel, and set aside until the next round The fuses are all ignited and the shots go in the order desired without any attendant danger.

While seemingly complicated in description, this device can be cheaply installed where the current is available. In making this device nothing is required besides the ordinary material and apparatus kept at a mine where electric current is used for power. The original device was patented by one of the miners who was working in the shaft when it was used, but as neither of the inventors have taken out a patent, the perfected form is not covered by the stolen patent.

This device is as simple to operate as a telephone switch board, while the attaching of the firing blocks to the fuse takes little more time than would be required in spitting a fuse with a torch, and is quicker than spitting with a hot iron; but, of course, it is not speed that is important, but safety."—Robt. N. Bell, *Engineering and Mining Journal.—Queensland Government Mining Journal*, March 15, 1909, p. 129. (C. B. S.)

Monazite.—"The extensive application of the oxide of thorium in the manufacture of mantles for incandescent gas lights creates a large demand for minerals containing thorium. Of the latter, monazite is found to be far the most abundant, and is therefore the principal source of thoria. Dr. Charles Baskerville mentions eighty minerals in which thorium has been found. Some of these would possess high commercial value as a source of thorium if found in sufficient quantity. Among these may be mentioned thorianite with 58 to 89% of thoria, auerlite with 70%, calciothorite with 59 to 60%, thorite with 48 to 72%, uranothorite with 48 to 52%, and thorogummite with 41 to 42%. Auerlite has been found in small quantities in the zircon mines of Henderson County, N.C. Frank L. Hess describes the occurrence of several thorium-bearing minerals along with the other rare earth minerals in the famous Baringer Hill locality, Texas. These are mackintoshite, thorogummite, nivenite, and yttrialite; they are of scientific interest rather than of commercial value as a source of thoria. Thorianite and thorite, especially the former, are found in quantities of commercial value in Ceylon.

Monazite is a phosphate of cerium, lanthanum, praseodidymium, and neodidymium with a variable percentage of silica and thoria. The proportion of thoria in monazite ranges from less than 1% to more than 20%. The average quantity in monazite obtained for commercial purposes varies between 3·

and 9 per cent. The colour of the monazite ranges from grayish to yellow, to reddish, to brownish, and greenish. The lustre is resinous, and is especially brilliant on cleavage faces. Monazite is opaque to translucent, and subtransparent. The specific gravity ranges from 4·9 to 5·3, and is generally over 5, so that the mineral is readily concentrated by ordinary methods of washing. The mineral is brittle, with a hardness of 5 to 5·5. Monazite generally occurs in small crystals with brilliant faces in the original rock matrix. These crystals are rounded by attrition when set free from the rocks and deposited in gravel beds by streams.

The bulk of the world's supply of monazite for the preparation of thorium nitrate used in the manufacture of incandescent gas mantles comes from Brazil. The Brazilian monazite is exported to foreign countries, principally Germany and Great Britain, for manufacture. The plants manufacturing thorium nitrate from monazite in the United States are supplied entirely by the domestic production. Up to the present time all the monazite produced for commercial purposes in the United States has come from North Carolina and South Carolina. Monazite occurs in a number of other States, and apparently promising deposits have been tested in Idaho and Georgia. The monazite region of North Carolina and South Carolina is a north-east—south-west belt twenty to thirty miles wide and over 150 miles long. This belt lies wholly within the Piedmont Plateau, and borders closely on the Blue Ridge, to whose general course it is roughly parallel.

Monazite occurs in quantities of commercial value in the Carolinas in the form of sand in alluvial gravel deposits. These deposits are situated in the bottom lands along creeks and streams, in the stream beds, and in some cases in terrace formations. The gravels of these deposits vary in richness, and probably do not often run over 1% monazite. The gravel beds range from one foot to several feet in thickness, and they cover areas varying from a few square yards to many acres. Occasionally, the rock formations similar to those from which the monazite is derived carry an appreciable amount of monazite. In no cases, however, have these rocks been found rich enough for profitable working on a large scale.

The usual method of mining monazite is by concentration in sluice boxes. In some cases Wilfley or other concentrating tables, operated by gasoline motors, are used to separate the monazite from the gravel. The crude sand thus obtained ranges from 15% to 60 or 70% monazite. By treatment with electro-magnetic machinery, the grade of the sand is raised to about 90% monazite with an available content of 4½ to 5% of thoria. The monazite is shipped in this form to the manufacturers, and there turned into nitrate with a thoria content of 48 to 50%."—DOUGLAS B. STERRETT—*Mineral Resources, U.S.A.*, 1908, *London Mining Journal*, June 5, 1909, p. 700. (A.R.)

COST OF UNDERGROUND WORK.—"The general manager of the Golden Horseshoe Mine has furnished the subjoined statement relating to the underground department, and containing some interesting particulars of the operations of the mine during the year ended 31st December, 1908.

(Tons wherever shown are of 2,240 lb.)

Work done by 3⅝ in. Rock Drill Machines.—The total average number of machines in constant use throughout the mine was 30·81. The following are particulars of work done in stopes and on development:—

Stoping.—

Machines in use per shift	24·81
Number of holes bored	72.338
Average depth per hole bored	6·92 ft.
Average number of feet bored per drill per shift	22·53 ft.
Tonnage of ore broken per drill per shift	10·74 tons

Explosives.—

Stoping: Explosives used per ton of ore broken	0·745 lb.
Drives: Explosives used per foot driven	14·09 lb.
Crosscuts: Explosives used per foot crosscut	9·66 lb.
Passes, Rises and Winzes: Explosives used per ton 'risen' or sunk	9·38 lb.
Shaft Sinking: Explosives used per foot sunk	16·16 lb.

Ore Broken and Raised.—

Average tonnage per man per shift (based on total number of men engaged in the underground department, viz., 495) = 1·6825 tons.

Average tonnage per man per shift (based on total number of men actually engaged in stoping and filling ore into shoots, viz., 275 men for 298 1-3 days, breaking 238,552 tons) = 2·9077 tons."—*Monthly Journal*, Chamber of Mines, W.A.—*Australian Mining and Engineering Review*, June 5, 1909, p. 294. (A. R.)

MISCELLANEOUS.

AN IMPROVED TAPE REEL FOR ENGINEERS AND SURVEYORS.—"This new design is intended to overcome the many objections, difficulties and annoyances incident to the use of the present designs of reels—objections which, in many cases, even a novice in their use has easily realized, but which somehow have escaped the attention of the reel makers.

Among these defects are (1) slow winding; (2) difficulty in preventing the unwinding of the tape; (3) easy clogging along friction surface with dirt and grit. Smaller defects may be found in the large friction surface, short winding crank, etc. The makers of this new reel, which they call the "Ideal," claim to overcome all these and other defects. Thus, where it takes—with the present tape-reel—about 130 turns of the winding crank to wind up a 100-ft. tape, consuming from thirty-five seconds to a minute of time, by means of an arrangement of gearing, giving three revolutions of the tape drum to each revolution of the winding crank, only 40 revolutions are necessary to wind up the 100-ft. tape. This can be done with ease in ten seconds. This feature of quick winding is particularly desirable in working on city streets and other places where interference from traffic is constant, and many broken tapes would be avoided by such a rapid winding device. The second defect in the present reels mentioned above is obviated by the introduction of a 'tape arrester.' This is merely a rod working in a tube attached to the reel-plate. Normally, the inner end of this piston is free from the gear wheel, but on pressing it with the forefinger the inner end engages the teeth of the larger gear wheel and instantly arrests the unwinding of the tape. This feature, taken in combination with a leather handle attached to the reel, permits the operator to make accurate measurements without the necessity of laying the reel aside or otherwise disposing of it—it will save cut fingers, broken tapes, and inaccurate operations, to say nothing of the temper of the operator.

The third defect mentioned, that is, the clogging of the friction surface, as well as the other minor defects, are overcome by the form and method of construction. All the gearing is well protected and the form of the plates has been made to fit the hand and thus prevent slipping during the operation of winding. The drum is light, strong and free from friction surface, revolving about an axle attached to the smaller gear; there is nothing to be clogged, and any grit that may collect about the axle can be easily removed. Altogether the advantages claimed for the new reel are so striking that every tape user should investigate its merits."—*The Engineering Digest*, Feb., 1909. (C. B. S.)

Bonding New to Old Concrete.—"For connections made after a lapse of 24 hours or more, break back the surface concrete to firm material, and clean the fresh surface with steam, air blast or forceful water streams so as to remove all fine loose material. Saturate well, but not so that water stands on the surface or oozes from the material. Paint completely with neat cement grout, mixed to the consistency of thin cream, just before new concrete is deposited, and see that the latter is of proper mixture, containing a proper proportion of mortar, which should be worked against the joint so as to be certain that no voids exist in its vicinity.

For connections made after long intervals, so that the old cement has set hard, and where the expense of rough-pointing the whole surface is greater than is required because of the nature of the desired bond, use commercial muriatic acid, diluted with clear water, 1 to 5, or the commercial bonding powders, dissolved in clear water at the rate of 5 lb. of powder to 10 gal. of water. First wet the old concrete surface with so much water that a fresh wetting is not immediately absorbed. Remove any excess of moisture, and, when the surface appears as if commencing to dry, paint on the old surface three successive coats of acid one after the other. Let this remain for about 30 min., after which carefully clean the surface of unspent acid, soluble salts, and fine material, with plenty of water, finally cleaning with a steam jet or air blast if obtainable. Just before the fresh concrete is to be deposited, and while the old material is still very damp, apply a thin coat of neat cement grout mixed to the consistency of thin cream just before the new concrete is deposited, and see that the latter is of proper mixture containing a proper proportion of mortar, which should be worked against the joint so as to be certain no voids exist in its vicinity.

Generally, if proper bonding methods are adopted, with acid and grout, work may be stopped along 45° planes at any point in a beam, at pleasure, between the ends and the third points; such planes would better be diagonal compression planes. The writer prefers such joints to vertical ones at the centers of beams, where direct tensile and compressive stresses are larger. Further, it is to be remarked that nothing but ample reinforcement, consisting of stirrups or numerous diagonal rods, can compensate for the detrimental effect of a horizontal joint left along or near the central axis of a beam. The writer prefers numerous stirrups, well connected to the tension rods, and amply anchored in the top concrete."—From a paper read by Mr. E. P. Goodrich before the American Society of Civil Engineers, March 3, 1909. —*The Engineering Digest.* (C. B. S.)

The Gas Engine and Low-grade Fuels.—"The value of a combustible as a fuel for power generation depends not only on its thermal efficiency, but on local conditions affecting cost. While coal and petroleum are the most efficient and concentrated fuels available for power purposes, they may not always be the most economical.

There is an increasing effort to make available the lower grade of fuels at the places where they are found and tranportation costs do not have to be considered. Efforts in this direction have met with considerable success especially in the adaptation of the gas producer to the generation of gas for gas engines, from materials that have usually been considered as having little fuel value. The power being generated by exploding the gas with air, the attainment of high temperatures required in steam generation is not necessary, or desirable, sufficient heat only need be generated in the producer to distill the volatile matter from the fuel and to burn the solid carbon to the combustible carbon monoxide. The completion of the combustion takes place in the cylinder of the engine when the mixture of gas and air is exploded.

Producer gas through its use in gas engines is becoming an important factor in solving the power problem in those regions where water is too scarce or contains too many impurities for satisfactory use in steam boilers."—*The Mining World*, June 26, 1909. (C. B. S.)

Meaning of the Kilowatt Hour.—"Many are puzzled to understand the peculiar significance of some of the terms used in denoting the measurement of electric current. The words 'amperes,' 'volts' and 'watts' slip easily off the tongue of those who are familiar with the language of electrical science, but it is not very well appreciated how many are unable to comprehend their precise meaning. The 'kilowatt hour,' however, gives more trouble than any of the other terms.

To explain this term needs first a clear definition, and then a comparison. Everyone will understand that a certain amount of force must be used to drive electric currents through a circuit. This force is measured by volts; thus we have 110-volt currents and 220-volt currents, the one expressing just twice the force of the other. But the quantity of current passing through a circuit depends upon the force and the resistance, and so the quantity is expressed by a different term, viz., 'amperes.' Now the efficiency of the current depends upon both force and quantity, and to express this efficiency or united action we multiply the force by the quantity—that is, the volts by the amperes—and express the result in watts. Thus 110 volts multiplied by five amperes is 550 watts, or 220 volts multiplied by five amperes is 1,100 watts.

A kilowatt is, of course, 1,000 watts, which is equivalent to about $1\frac{1}{3}$ h.-p. In charging a battery the lighting company's bill for the use of so many watts for so many hours—thus 1,000 watts for 10 hours—would be charged as 10 kilowatt-hours, which at 5 cts. a kilowatt-hour would be 50 cts., a charge that seems little enough for 10 hours' use of $1\frac{1}{3}$ h.-p." —*Mining World*, June 26, 1909, p. 1228. (C. B. S.)

Increase in World's Gold Production.—The London *Statist* gives the world's production of gold in 1908 as the largest in history, or \$409,000,000, against \$400,000,000 in 1907 and \$390,000,000 in 1906. The outlook is for a further expansion of gold production, because of its stable value, more economical mining with modern machinery, and the thorough extraction, compared to the loss of one-third of the gold in milling a generation ago.—*United States Consular Report.—Mines and Minerals* August, 1909, p. 44. (A. R.)

Test of Large Reinforced Concrete Beams.—"A pamphlet issued by the University of Illinois gives particulars concerning an interesting series of tests conducted by Professor A. N. Talbot with the object of determining the actual properties of beams made under practical conditions of construction. The members were taken from a number of beams prepared for use on the Illinois Central Railroad, and were in the form of slabs 25 ft. long by 6 ft. 3 in. wide by 2 ft. 10 in. deep at the middle, the average weight being 33 tons each, the weight being probably expressed in American tons of 2,000 lb. It is satisfactory to observe that the behaviour of the slabs was similar to that of beams made and tested in laboratories, a result which should inspire confidence in the quality and soundness of reinforced concrete work of judicious design and constructed with care. The tests show very clearly the efficiency of vertical U-shaped stirrups and bars bent up towards the ends in withstanding diagonal tension. Ample precaution against incipient failure by diagonal tension is specially important because of the risk of sudden failure and the impossibility of detecting signs of weakness in places where the sides of beams are not open to inspection."—*Mining, Building and Engineering Journal*, July 10, 1909. (C. B. S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

The Copper Handbook for 1908. Compiled and Published by H. J. Stevens, Houghton, Mich., U.S.A. $5.

"The present is the eighth annual edition of this work which has grown since 1901 from 328 pages to 1,500 pages of text. No less than 6,767 copper mines and companies are listed and described, the descriptions varying from a few lines to several pages according to importance. There are also 23 chapters dealing with the history, geology, geography, mineralogy, chemistry, metallurgy and statistics of copper. To those persons who are interested in copper mines the work is invaluable."—*Mining Engineering*, July, 1909. (A. R.)

Handbook for Field Geologists. By C. W. Hayes, Ph.D., Chief Geologist, U.S. Geological Survey. viii. +159, (New York: John Wiley & Sons. London: Chapman & Hall, Ltd.). 6s. 6d.

"One of the most useful geological handbooks that we have seen for a long time is a handbook for Field Geologists which has recently been received for review. In order to make the handbook as complete as possible within reasonable limits some 84 pages of general instructions are given with regard to preparation for field work, outfit, field observations and the method of making various horizontal and vertical measurements, the use and manner of recording notes, and the collecting of specimens, together with a deal of information with regard to the various details of field operations. Excellent and concise as these instructions are, the geologist who has occasion to do much work in the open will doubtless find the instructions for special investigations which are given in Part II. a most useful and thoroughly well arranged series. A number of schedules have been prepared with a view to securing system and completeness in making and recording observations. Most of them are not intended to be exhaustive or to cover all possible points on which the specialist will make observations. They are rather for the guidance and help of the specialist when he is working outside his speciality These objects have been clearly borne in mind by Dr. Hayes, who deals separately with the different classes of rocks, igneous and sedimentary, and points out the special matters which are to be investigated and considered in each. At the end of the chapters in which these things are discussed, a summary of the instructions is given, together with notes regarding the significance of the various characteristics which are necessary to be noted. Following these there are given a collection of schedules containing every important heading under which facts should be noted, regarding the geology and structure of rock masses generally, the geological details of glaciers and glacial deposits, metalliferous ores, placer deposits, stone, road materials, cement materials and lime, clay, sand, gravel, coal and gas, and the rest. In all there are fourteen separate schedules, and it would be difficult to think of any rock characteristic or point, however small, about which inquiry should be made, which is not in one way or another mentioned or suggested in the schedules. The handbook is of convenient size, well bound, and in every way a *vade mecum* which should be in the hands of every geologist who takes his work seriously."—*S.A. Mining Journal*, Aug. 21, 1909, p. 702. (A. R.)

Igneous Rocks. Vol. I. By Joseph P. Iddings. (New York City: John Wiley & Sons. $5.

"Vol. I. is devoted to composition, texture, and classification of igneous rocks. It is, naturally, highly specialised and interests chiefly the petrographers. The book is well illustrated and contains copious references to the general literature on the subject."—*Mines and Minerals*, Aug., 1909. (A. R.)

Handbook on Tacheometrical Surveying. By C. Xydis, C.E. 6s. (London: E. & F. N. Spon, Limited.)

"A useful book, which comes to fill a gap in our technical literature, as hitherto very little has appeared in print on the subject handled so thoroughly in this work. We remember only a paper by the late Mr. Brough and another by Mr. Niel Kennedy. The pros and cons of the subject we need not discuss here, as they are well known to those of our readers who are interested in such work. Unlike the majority of its kind, this work is anything but voluminous, and on that ground, in this busy age, it appeals to us; moreover, its information is couched in plain and concise terms, with many practical suggestions for methods of working, serviceable alike to surveyors and their pupils."—*London Mining Journal*, Aug. 21, 1909. (A. R.)

Prehistoric Rhodesia. An Examination of the Ethnological and Archaeological Evidences as to the Origin and Age of the Rock Mines and Stone Buildings, with a Gazetteer of Mediæval South-East Africa. By R. N. Hall, author of 'The Ancient Ruins of Rhodesia.' With Illustrations, Maps and Plans. 12s. 6d. (London: T. Fisher Unwin.)

"This book describes the ruins and ancient mine workings of Rhodesia. It is published in an attractive form, while the descriptions are aided by a number of most excellent photographs.

Mr. Hall does not believe in Prof. MacIver's conclusions — viz., that the ruins are comparatively modern. He endeavours to prove their antiquity, and to upset the statement made—that Nankin china was found below the walls of the oldest buildings.

Investigations of this nature present great difficulties, for we note the ruins appear to be almost characterised by their irregular design, while the absence of squares and circles in their constructive details is a feature worth noting; the apparent absence of inscriptions increases the difficulties of research work. Mr. Hall is to be congratulated upon the way he has massed and pieced together the scattered records, involving, as it must have done, a vast amount of patient labour.

It seems doubtful if the true age of these Rhodesian remains will ever be known. The extensive ancient mine workings which abound all over that territory are evidence of large quantities of gold having been mined. We think that the estimates of tonnages and values of ground treated are too problematical to be considered.

The conclusions generally favour the antiquity of these remains, and there is also evidence that the mine workings have been undertaken from ancient even up to modern times.

It would appear that Rhodesia in the past has provided a large population with the means of subsistence, and that to-day the country promises to rival its ancient condition in value and prosperity."—*London Mining Journal*, Aug. 21, 1909. (A. R.)

INVESTIGATION OF THE PEAT BOGS AND PEAT INDUSTRY OF CANADA DURING THE SEASON 1908-9. By ERIK NYSTRÖM, M.E., and S. A. ANREP, M.E. (Ottawa, Canada: Department of Mines.)

"A year ago, the Mines Branch of the Department of Mines, Ottawa, issued a report on 'Peat and Lignite, their Manufacture and Uses in Europe,' with the object of giving to Canadians as complete a review as possible of this industry in those countries in which it has been most successfully carried on. This report is now followed by a bulletin, entitled 'The Investigation of the Peat Bogs and Peat Industry of Canada during the Season 1908-9,' by Erik Nyström, M.E., peat expert. This bulletin comprises 25 pages of text, and includes six large scale maps of peat bogs.

The bulletin contains a descriptive report of each bog, showing the location, area and structure, and giving an estimate of the available supply of peat fuel, with records and analyses, calorific values, etc., and should be of particular interest to those engaged in, or connected with, the development of Canadian peat resources.

A fuel-testing plant is now being erected at Ottawa, in which the value of peat for the production of power-gas will be demonstrated, and the Department proposes to carry on a very thorough investigation of this subject." — *London Mining Journal*, Aug. 21, 1909. (A. R.)

METALLURGY OF THE COMMON (NON-FERROUS) METALS. By A. HUMBOLT SEXTON, F.I.C., F.C.S., etc., and JOHN S. G. PRIMROSE, A.G.T.C., etc. (Manchester: The Scientific Publishing Company.)

"The exhaustive treatment which has been accorded to various common non-ferrous metals by different authors rather tends to dwarf the importance of a work which, in one volume of less than 500 pages, deals with the metallurgy of a number of these metals. But it must be explained that this book is only intended as an introduction. The requirements of those engaged in metallurgical work will be better met by the larger works, a list of which is given by the authors, for it is obviously impossible in the space occupied to deal fully with the metallurgy of each metal. For instance, antimony occupies 12 pages only, whereas another author finds sufficient material to fill a complete volume on the subject. Similarly other metals are only dealt with briefly, though copper comes in for much fuller treatment. This is only meet, in view of the extensive and varied uses to which that metal is put and the number of processes employed for extracting and refining it. In fact, about one-third of the book is devoted to the subject of copper, including the mineral, the principles of copper smelting, processes for treating the ore, and refining processes. The subject of tin is summarised in about 40 pages, and then follow zinc, lead, antimony, aluminium, and nickel. It may be suggested that in such a work it would have been well to include some information on other of the metals which, while scarcely classed as common, yet are in more or less extensive commercial use. That, however, is beside the point now, for the authors have set the limitation and have worked within it, producing a book which is certainly an excellent introduction to fuller works. Naturally, there has been made little attempt to offer original matter; rather has existing information been gathered together and condensed into one volume. This, we may say, is one of the leading characteristics of the work, and the most important features of the metallurgical processes dealt with are thus presented in a concise and systematic way."—*Iron and Coal Trades Review*, Sept. 5, 1909. (A. R.)

The following reports have been received from the Mines Departments mentioned, and have been transferred, for the convenience of members and for ready reference, to the Seymour Library, Transvaal University College.

A. From the Department of Mines, Canada.

1. Report on the Chrome Iron Deposits of the Eastern Townships, Province of Quebec. By Fritz Cirkel. pp. I.-IV. and 1-141.

This is the 29th number of the valuable series now being issued by the Canadian Department of Mines under the direction of Dr. Haanel. Its primary object is to meet the demand for information on certain aspects of the iron and steel industry, which in recent years has developed so rapidly in Canada. It deals very fully with the subject of its title, and in addition there are sections on the ores of other countries, including the Transvaal (c.f. Hall and Humphrey, Trans. Geo. Soc. S. A., Vol. XI., May 18, 1908), and Rhodesia, as well as the composition, statistics and chronology of the chrome industry. A chapter on the determination of the value of chromite follows, and then one on its uses. It will be of interest to our members to quote the following from p. 102: "Chrome steel is extremely hard, tough and dense. It is possessed of great tensile strength, and is superior to any metal known for the wearing parts of mills used for crushing and pulverising gold, silver and other ores. This steel is especially adapted to severe service and resistance to abrasion, and for the wearing parts of stamp mills, such as battery shoes and dies, cams, tappets and bosses, chrome steel has proven, by reason of its durability, the most economical and satisfactory material to be obtained." A useful chapter on the technology of chromium and its compounds follows, and two appendices *(a)* a reprint from the Mineral Industry on the Metallurgy of Chromium, *(b)* Experiments with Chromite at McGill University, Montreal. The report is well printed and illustrated with fifteen diagrams and eleven plates in half-tone, and will be valuable to all who are concerned with this industry. (J.A.W.)

2. Report on the Investigation of an Electric Shaft Furnace, Domnarfvet, Sweden, etc. By E. Haanel, Ph. D. pp. 1-38.

This report forms a sequence to the reports published in 1904, 1906 and 1907 by the department on the electric smelting of iron ores, and the investigation was undertaken by Dr. Haanel in December last on the invitation of the inventors. The results given represent the latest developments of the electric smelting of iron ores. The report is divided into four parts, the first of which deals very fully with the trial runs, and it is interesting to note that the experiments undertaken show that a saving could probably be effected under Swedish conditions at Domnarfvet of about $1·55 per ton by the use of this furnace. In addition to the above there are two valuable reports constituting Parts III. and IV., the former dealing very fully with the general arrangements and full working details of the factories for the manufacture of carbon electrodes, and the latter with modern methods of charcoal making. Part II. describes a new electric furnace for the manufacture of steel. (J. A. W.)

B. From the Department of Mines, Western Australia.

1. Report on the Progress of Mining in the districts between Leonora and Wilma. By A. Montgomery, M.A., F.G.S., State Mining Engineer. pp. 1-85.
2. Report on the Waverley or Siberia District. By A. Montgomery, M.A., F.G.S., State Mining Engineer. pp. 1-26.
3. Report of the Department of Mines for the year 1908.
4. Bulletin No. 35. Geological Survey, Western Australia. Geological Report on the Gold and Copper Deposits of the Phillips River Goldfield. By H. P. Woodward, Assistant Government Geologist, with which is incorporated a description of the Crystalline Rocks of the District. By E. S Sampson and L. Glauert (with two maps, eight plates and seven photographs). pp. 1-175.

C. From the Department of Mines, India.

1. Report of the Chief Inspector of Mines in India for the year ending 31st December, 1908. By J. R. R. Wilson, M.I.C.E., F.G.S. pp. 1-140.

D. From the Missouri Bureau of Geology and Mines.

1. Vol. VI., 2nd Series. The Lime and Cement resources of Missouri (U.S.A.) By H. A. Buehler, B.S., Assistant State Geologist. pp. I.-XVI. and 1-254.
2. Vol. V., 2nd Series. Public Roads. Their Improvement and Maintenance. By E. R. Buckley. (J.A.W.)

Adams, Prof. Henry. Cassell's Building Construction. Comprising Notes on Materials, Processes, Principles and Practice, including 2,284 engravings and 12 Plates. Part I. 4to., sd., pp. 24. *Cassell.* Net 3d.

British Standard Specification for Cast Iron Pipes for Hydraulic Power. (Engineering Standard Committee, No. 44. Folio, sd. *Lockwood.* Net 5s.

Butler, Edward. Carburettors, Vaporisers, and Distributing Valves Used in Internal Combustion Engines. 8vo., pp. 188. *C. Griffin.* Net 6s.

Caven, R. M. Systematic Qualitative Analysis. For Students of Inorganic Chemistry. Cr. 8vo., pp. 240. *Blackie.* Net 3s. 6d.

Colby, Albert Ladd. Reinforced Concrete in Europe. Its Applications, Economic and Endurance. The Systems, the Forms of Bars, and the Metal used in England and on the Continent. Royal 8vo. *Williams & Norgate.* Net 14s. 6d.

Crane, W. R. Index of Mining Engineering Literature 8vo. *Chapman & Hall.* Net 17.

Harrison, Newton. Wireless Telephone. Construction. Cr. 8vo., pp. 74. *Spon.* Limp, net 1s. 6d.

Hatch, F. H., and Corstorphine, G. S. The Geology of South Africa. 2nd Edition, 8vo., pp. xvi., 394. *Macmillan.* Net 21s.

Haughton, C. E. The Elements of Mechanics of Materials. A Text Book for Students in Engineering Courses. 8vo., pp. 194. *Constable.* Net 7s. 6d.

Holley, C. D. The Lead and Zinc Pigments. Cr. 8vo *Chapman & Hall.* Net 12s. 6d.

Koester, Frank. Hydroelectric Developments and Engineering. Illustrated. Imp. 8vo., pp. 480. *Constable.* Net 21s.

McDonald, J. G. Hints to South African Farmers. Cr. 8vo., pp. 88. *Argus Printing Co.* Net 2s. 6d.

Moor, C. G. The Recognition of Minerals. Being a Collection of Notes and Simple Tests for the Use of Travellers and Prospectors. With Monographs on Geology. Ore Deposits, etc., By Donald A. MacAlister. Cr. 8vo., pp. vii., 231. *Mining Journal.* Net 7s. 6d.

Morris, W. L. Steam Power Plant Piping Systems. 8vo. *Spon.* Net 21.

Philip, James C. The Romance of Modern Chemistry. A Description in Non-Technical Language of the Diverse and Wonderful Ways in which Chemical Forces are at Work and of their Manifold Application in Modern Life. With 29 Illustrations and 15 Diagrams. Cr. 8vo., pp. 348. *Seeley.* 5s.

Ramsay, John. Engineering Units of Measurements, with Symbols and Abbreviations. British and Metric Equivalents. 8vo., sd. *Simpkin.* Net 1s.

Raynes, F. W. Domestic Sanitary and Engineering and Plumbing. 8vo., pp. 488. *Longmans.* Net 10s. 6d.

Report of the Senior Analyst of the Cape of Good Hope for the Year 1908. Folio, sd., pp. 25. *W. Wesley.* Net 2s.

Scales, F. Shillington. Practical Microscopy. An introduction to the Microscopical Methods. 2nd Ed. Cr. 8vo., pp. 350. *Bailliere.* Net 5s.

Science Chemistry Papers. Being the Questions set at the Intermediate Science Examination of the University of London. From 1892 to 1909. (University Tutoial Series.) Cr. 8vo., pp. 56. *Clive.* 2s. 6d.

Thomas, H. H., and MacAlister, D. A. The Geology of Ore Deposits. Illustrated. Cr. 8vo., pp. 428. *E. Arnold* Net 7s. 6d.

Thomas, H. H., and MacAlister, D. N. The Geology of Ore Deposits (Geological Series). 12mo., pp. 416. *E. Arnold.* Net 7s. 6d.

Tower, Walter Sheldon. The Story of Oil. Illustrated. Cr. 8vo., pp. 284. *Appleton.* Net 4s.

Trade, Board of. Electric Lighting Acts, 1882 and 1888. Regulations: *(a)* for securing the safety of the Public, and *(b)* for ensuring a proper and sufficient Supply of Electrical Energy.

Unwin, W. Cawthorne. The Elements of Machine Design. Part I. New Edition Revised and Enlarged. 8vo., pp. 546. *Longmans.* Net 7s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 416/09. Maurice Leblane. Improvements relating to condensing and refrigerating apparatus. 10.9.09.

(C.) 417/09. The Hon. Charles Algernon Parsons, C.B. Improvements relating to turbines. 10/9/09.

(P.) 418/09. Thomas William Weston. Weston's patent wedge drive. 11.9.09.

(P.) 419/09. Herbert Briggs. Improvements in mortar boxes for stamp mills. 14.9.09.

(C.) 420/09. Edwin Ernest Spargo. A contrivance for holding and operating hand rock drills. 14.9.09.

(P.) 421/09. Henry Cawood Embleton. Improvements in or connected with pneumatic stamps for crushing ores and the like. 14.9.09.

(P.) 422/09. Austin John Roberts Atkin. Improvements in jockeys or rope grips. 15.9 09.

(C.) 424/09. George Lester Whitcombe. Improvements in or connected with concentrators or apparatus for separating or grading metals, ores and other substances in wet recovery processes. 17.9.09.

(C.) 425/09. James Miners Holman (1), John Leonard Holman (2). Improvements in or connected with jigging mechanism for operating conveyors, separators and the like for ores and substances. 17.9.09.

(P.) 427/09. George Allen Fowler. Improvements in tunnelling machines. 18.9.09.

(P.) 428/09. David Meffan Grassie (1), Edward Chard (2). Cover for amalgamating plates, zinc boxes, and the like. 18.9.09.

(P.) 429/09. Charles Reeves. Improved method for the recovery of gold from liquids and the like. 20.9.09.

(C.) 433/09. William Stansfield (1), John Joseph Hatt (2). Improvements in and relating to magnet windings of electro-magnetic apparatus. 24.9.09.

(C.) 434/09. Percy William Cook. Improvements in and relating to armoured concrete constructions. 24.9.09.

(C.) 435/09. Ernst Schmatolla. Improvements in kilns for burning limestone, dolomite and the like. 24.9.09.

(P.) 436/09. Frank William Marsden. Improvements in milling and amalgamating apparatus. 25.9.09.

(C.) 437/09. Samuel Russell. Improvements in the fixing of pulleys and the like upon shafts. 25.9.09.

(P.) 438/09. William Cochran Boyd. Improvements in mortar box. 27.9.09.

(P.) 439/09. John Daniel Robertson. Automatic sanitary tidal tank. 27.9.09.

(P.) 440/09. John Hughes (1), Samuel Bates McKee (2). Improvements in means for transmitting power. 28.9.09.

(C.) 441/09. Edward Brice Killen. Improvements in and connected with rubber tyres and their attachment to wheels. 28.9.09.

(C.) 442/09. Edward Brice Killen. Improvements in or relating to wooden wheels. 28.9.09.

(C.) 445/09. Edward Farrar. Improvements in classifying and concentrating apparatus. 30.9.09.

(C.) 446/09. John William Hall (1), Cyril Baynes (2). Improvements in or relating to detachable rims for wheels. 30.9.09.

(P.) 447/09. Henry Tyzzer. An improved clutch for ropes and relating to mechanical haulage. 1.10.09.

(P.) 448/09. Rollo Bowman Ballantine. An improved process for the manufacture of cyanides. 1.10.09.

(P.) 449/09. Rollo Bowman Ballantine. An improved process for the manufacture of cyanides. 1.10.09.

(P.) 450/09. Rollo Bowman Ballantine. An improved process for the manufacture of cyanides. 1.10.09.

(P.) 451/09. George Taylor Philip (1), Arthur Chichele Plowden (2). Improvements in or relating to vessels for holding oil or other fluids. 2.10.09.

(P.) 452/09. Edmund Arthur Hamblin. An improvement in and relating to the method or manner of coupling railway engines, carriages, trucks or vehicles of any description. 5.10.09.

(P.) 453/09. Harry August Wilson. A new or improved combined automatic depth and weight indicator for use on mines. 6.10.09.

(P.) 454/09. Charles Reeves. Improved method for treatment of slimes and the like. 6.10.09.

(P.) 456/09. John Mitchell Lafayette Henry. Method of treating acid water. 6.10.09.

(C.) 457/09. John Hugh Fortescue. Roller mill concentrate crusher. 6.10.09.

(C.) 458/09. Edward James Horwood. Improvements in and relating to the separation of zinc blende and other metalliferous constituent from ore concentrates and slimes by flotation or granulation. 9.10.09.

(P.) Hans Charles Behr. Improvements in hoist engines. 9.10.09.

(P.) 461/09. Hans Charles Behr. Improvements in ore storage and handling devices. 9.10.09.

(P.) 462/09. Henry John Barclay. An invention for the development of photographic plates without the use of a dark room. 9.10.09.

(P.) 463/09. David Laird. A coating for metallic or earthenware surfaces of carborundum to prevent abrasion. 11.10.09.

(P.) 465/09. Edward Henry Johnson. Improvements in the process of recovering metals from their solutions. 12.10.09.

(P.) 466/09. Hans Charles Behr. Improvements in pneumatic ore stamps, hammers and the like. 12.10.09.

(P.) 467/09. James Philip McKeown. Improvements in mortar boxes for stamp mills. 13.10.09.

(P.) 468/09. Adrien Marie Aloysins Birs. Gas purifier for suction gas plant. 15.10.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

The List of Members issued with this *Journal* is corrected to the date of going to press.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. NOVEMBER, 1909. No. 5.

Proceedings
AT
Ordinary General Meeting, November 20, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, November 20th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present:—

76 Members: Messrs. F. F. Alexander, E. H. Croghan, E. J. Laschinger, G. O. Smart, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, R. G. Bevington, W. A. Caldecott, A. F. Crosse, E. H. Johnson, S. H. Pearce, Prof. J. Yates, H. A. Adams, R. Allen, A. Avent, S. G. Bartlett, S. Beaton, F. W. Bentley, F. L. Bosqui, P. Carter, F. Chapman, W. M. Coulter, J. Daniel, J. F. Ferguson, J. Gibson, J. R. Gibson, G. Goodwin, J. Gray, W. L. Hamilton, J. H. Harris, B. J. Hastings, Dr. J. McC. Henderson, S. A. Herbert, C. B. Hilliard, J. H. Johnson, T. Johnson, J. Kennedy, R. N. Kotzé, G. A. Lawson, H. Lea, J. Lea, R. Lindsay, R. MacGregor, P. T. Morrisby, M. T. Murray, S. Newton, F. B. Ogle, C. F. Parry, D. J. Pepler, J. Pyles, E. T. Rand, W. H. Roe, O D. Ross, A. Salkinson, C. O. Schmitt, S. B. Schwabe, R. D. Scotland, W. Sharp, C. B. Simpson, J. A. Smith, S. H. Steels, R. Stokes, W. H. Stout, J. B. Sutton, W. A. C. Tayler, A. Thomas, J. E. Thomas, J. M. Thorburn, K. Tonnesen, A. D. Viney, J. F. Walker, J. P. Ward, P. H. Whyte and L J. Wilmoth.

26 Associates and Students: Messrs. A. R. Adams, H. H. Chadwick, D. Coll, C. W. Craig, J. Cronin, D. W. Greig, A. R. B. Hains, G. Hawkins, J. W. Hawkins, W. J. R. Hunter, R. L. Impey, A. King, C. J. McCaffrey, R. McMaster, G. Musson, J. H. Neethling, W. S. New, N. Newland, F. J. Pooler, A. Scott, H. Stadler, J. B. Sutton, A. M. Thomas, G. G. Thomas, W. G. Urquhart, and W. Waters.

20 Visitors, including Messrs. F. W. Harbord, A. Cobbe, J. A. Vaughan (Chief Inspector of Machinery), and Fred. Rowland, Secretary.

The Chairman: I regret having to announce that Dr. Moir, who was to have taken the chair to-night, is unable to be present owing to ill health.

The minutes of the previous meeting, as printed in the October *Journal*, were confirmed.

NEW MEMBERS.

Messrs. J. Littlejohn and S. H. Pearce were appointed scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been unanimously elected, as follows:—

Coles, William Alison, Ritanga Syndicate, Dunraven Mine, Selukwe, Southern Rhodesia. Amalgamator.
Mackenzie, Angus Campbell, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Foreman.
Semple, Clarence Carleton, 5, West 65th St., New York City. Mining Engineer.
Johnston, Joseph William, Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mine Captain.
Leslie, Hugh Mackenzie, A.Inst.M.M., M.Am.I.M.E., Kolar Gold Field Central Metallurgical Establishment, Marikuppam, Mysore State, S. India. Consulting Metallurgical Engineer. (*Transfer from Associate Roll.*)
Whaley, William Samuel, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Adendorff, John. 12, Pretoria Street, Troyeville. Mechanical Engineer.
Clevenger, G. Howell, 381, Hawthorne Avenue, Palo Alto, Cal., U.S.A. Metallurgical Engineer.
Hawkins, George, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Tube-miller.
Hawkins, J. W., City & Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.
Innes, John, The Observatory, Johannesburg. Sampler. (*Transfer from Student Roll.*)
McMaster, Richard, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Amalgamator.

Nettle, F. P., Van Ryn G. M. Estates, Ltd., P. O. Box 22, Benoni.
Nuttall, T. H., Van Ryn G. M. Estates, Ltd., P. O. Box 22, Benoni.
Smith, James Houston, Rose Deep, Ltd., P. O. Box 6, Germiston. Mine Overseer.
Wagner, Percy Albert, Transvaal University College, P. O. Box 1176, Johannesburg. *(Transfer from Student Roll.)*

As Students.—

Ashburner, William Coulton, Treasury Gold Mines, Ltd., Cleveland. Cyanide apprentice.
Neethling, John Henoch, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide learner.
Esplin, Douglas Gibson, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill learner.

General Business.

The Chairman: I have much pleasure in welcoming on your behalf several visitors to our meeting this evening. We are fortunate enough to have with us to-night Mr. F. W. Harbord, the well-known authority on steel, and author of "The Metallurgy of Steel." Mr. Harbord is also a member of the council of our sister society, the Institution of Mining and Metallurgy, and with him is his colleague, Mr. H. N. G. Cobbe, a mining and metallurgical engineer of extensive experience in Australia and elsewhere. We all trust that the valuable advice which these gentlemen can give the Government and people of this country upon the utilisation of its iron resources will be taken full advantage of. Our friend, Mr. F. L. Bosqui, from the United States, is also with us to-night, as a member of our Society. Mr. Bosqui has just completed the erection of a large reduction plant for the Goldfield Consolidated in Southern Nevada, which is working with excellent results. We are also pleased to welcome Mr. Robert Allen, the well known author of Westralian metallurgical practice, who, after many years metallurgical experience in Kalgoorli and Mexico, has come to the Consolidated Gold Fields of S.A., to join us in the solution of our local problems, and who, I am sure, will be a valuable acquisition to our Society as a resident member.

It is my pleasure, on your behalf, to offer congratulations to Mr. W. A. Caldecott, who has been awarded "The Consolidated Gold Fields of South Africa, Ltd.," Gold Medal by the Institution of Mining and Metallurgy. When awarding this medal the Council of The Institution of Mining and Metallurgy passed the following resolution:—

"That the 'Consolidated Gold Fields of South Africa, Ltd.,' Gold Medal be and is hereby awarded to William Arthur Caldecott, in recognition of his work in the investigation of methods of reduction and treatment of gold ores; and of his contributions to the literature on the subject."

Mr. Caldecott is one of the original foundation members of our Society, and as an ordinary member, member of council, president, and past-president, has always done more than his share of the work of the Society, and he continues to do so, not only by his valuable contributions but also by his work on the council.

Mr. Caldecott has done a great deal of original work and investigation in the recovery of gold and silver from their ores, and generally gives the mining and metallurgical public the results of his knowledge, as the pages of our *Journal*, those of other technical societies, and the mining press of the world, show.

I can safely say that we think the award is well earned and a recognition of knowledge and ability, which we who know him appreciate.

Mr. R. G. Bevington (*Past-President*): I think, Mr. Vice-President, your words are very few and that they might be supplemented by a few more from one who has known Mr. Caldecott from his earliest days on these fields, and who holds him in the greatest respect and esteem. Mr. Caldecott's work has always been most thorough, and I think the honour which has been shown him by the Institute of Mining and Metallurgy has been most worthily earned. I think also that considering Mr. Caldecott is one of our members—one of our oldest members and one of our Past-Presidents, our Society must also feel honoured in one of its members having received that distinguished honour.

Mr. W. A. Caldecott (*Past-President*): Mr. Chairman, I greatly appreciate the remarks which yourself and Mr. Bevington have been good enough to make, and the kind way in which they have been received by members present this evening. As a South African and a member of this Society, the award which the Institution of Mining and Metallurgy has seen fit to make is a source of great gratification to me, and a fresh evidence of the liberal spirit and world-wide scope of our sister institution.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

Part I.—Crucible Experiments.

By Prof. G. H. Stanley (Member of Council).

As is well known, there exist in many parts of the world immense deposits of titaniferous iron ore, many of them well situated for smelting purposes, but of which, as a source of iron, no use is made at present. It is, however, a fact, that we are within measurable distance of the exhaustion of other ores, recent calculations having placed the available amount from known deposits at only sufficient to meet the demand for the next sixty or seventy years, and scarcity will undoubtedly be felt long before then. In the Transvaal, notably, deposits of such ore occur in quantity, but in common with other ores of like character are generally looked upon as valueless on account of the high percentage of titanium they contain.

We have other ores, of course, but the majority as far as my experience goes, are very silicious and not to be considered as of good quality, though sometimes good enough to smelt under oversea conditions; I am informed, however, that excellent ores occur in certain places, often unfortunately far from a railway. Still, it seemed worth while making some investigations with the object of determining, if possible, a valid reason for the extreme disfavour with which titaniferous ores are looked upon by iron masters, for although laboratory experiments cannot approximate very closely to actual practice, yet valuable information can be gained with regard to many points likely to give trouble, and curiously enough, it is difficult to find in technical literature very definite causes for the non-use of this class of ore, and there is marked difference of opinion apparent among experts.

Beyond statements, unsupported by my personal experience, that the ore is very difficult to reduce on account of the formation of pasty and infusible slags, or of bears in the hearth, more or less due to the formation of cyanonitride of titanium or of other infusible titanium compounds, no reasons are given, though nearly all text books state in one way or another that titaniferous ores are refractory and incapable of being commercially smelted in the blast furnace. It has been demonstrated that these ores can be smelted electrically to produce good pig iron, but the electrical smelting even of first class ores can scarcely be considered as beyond the experimental stage yet, and in any case, cannot under usual conditions, compete with ordinary blast furnace smelting. It is only where water power is available and suitable fuel dear that it appears to have a field for this purpose.

It is stated that for each ton of iron smelted electrically something like ·3 E.H.P. year is required, and about one-third of a ton of carbonaceous material, such as coke or charcoal, which need not be of first quality. In the blast furnace from 1 ton to $1\frac{1}{2}$ tons of coke—the usual fuel—is required per ton of iron produced. This relates only to the cost of heat and reducing material in the two cases, and assuming other charges to be the same) which is by no means the case, the advantage for large outputs lying with the blast furnace); comparisons in this respect can be readily made, but in most cases where it is proposed to smelt iron ores, it is very necessary to demonstrate the impossibility of doing so in blast furnaces before resorting to electrical methods. It must be remembered too that contracts for electrical power for smelting purposes have been made in Europe at a figure approximating to £1 10s. per E.H.P. year, and in at least one case the cost is stated to be even less than £1.

Estimates based on such figures show less cost as compared with blast furnaces using coke at ordinary prices, but leaving coke out of the question for the moment, the question before the advocates of electric smelting in any instance would appear to be, "can electrical power be generated here and supplied to us at a sufficiently low price to enable us to compete with other producers obtaining their power at the rates mentioned?"

With regard to coke smelting in South Africa, in Natal there is coal, which from the result of my own examinations I may say is capable of producing coke, which while not perhaps of first class quality, is quite equal to that used in certain iron smelting districts: and from a Transvaal colliery I have received a sample, said to represent a 9-ft. seam, which also made quite a good coke, containing only 9·7% ash. I therefore think that if coke smelting were demonstrated to be feasible, a supply of coke could be relied upon. This, however, is only of local bearing, and in many other places the coke supply is assured, while cheap electrical power, *i.e.*, at anything like the figures previously quoted, is out of the question; and assuming that the fluxing of the titanium is the main difficulty, this would arise almost as much with electrical as with blast furnace smelting, though trouble might conceivably be minimised in the former case by the higher temperatures attainable. Consequently, any results obtained seemed likely to be of more than local interest.

Although the feeling of dislike of these ores is so widely held, it has been pointed out on more than one occasion that this is scarcely justified, except in cases where the proportion of titanium is so high, that, were it any other waste matter, it would be equally objectionable from the point of view of the expense of the extra fluxes and fuel required.

For instance, at Norton-on-Tyne, England, Norwegian titaniferous ores containing approximately 40% TiO_2 were regularly smelted for several years between 1869-1876, though this percentage of titanium was so high as to render them comparatively poor in iron and hence, presumably, barely profitable. In Sweden also similar results were obtained. Again, in the Adirondacks, in the U.S.A., similar ores, though not so high in titanium, were smelted in small furnaces for eight years (1848-1856), commercial reasons again causing the cessation of operations. Of late years Mr. A. J. Rossi, in the U.S.A., has devoted considerable time and energy to the problem of smelting these ores, and although he satisfactorily demonstrates the technical feasibility of such smelting he appears to be almost alone; and I am not aware that any extensive utilisation is made of them, at least as a source of pig iron.

It is noteworthy, however, that in the discussion on his papers, general objections were urged to the employment of such ores, and it was stated although without giving very definite reasons or specific cases, that quite small quantities of TiO_2 were almost as injurious as large and as certain to produce trouble sooner or later. On the other hand, it was stated that many works were smelting ores containing titanium, in which it was never determined but obtained with silica in the analysis, and in at least one case a certain works had been smelting an ore containing 5% TiO_2 for some years without knowing it. It appears obvious therefore that there are two sides to the question, and more light on the problem would be useful.

Briefly, Rossi's results showed that slags containing heavy percentages of titanium could be made easily and perfectly fusible, by working on the supposed analogy between silicates and titanates as regards the more easy fusibility of mixed than single silicates.

He figured the titanium present as TiO_2 and considered it as an acid like silica, so that the slags which he demonstrated to be fusible appeared to be of very acid character, mixed silico-titanates of lime, aluminium, and magnesium in fact.

For my experiments, the ore used was kindly obtained by Mr. Kotzé, Government Mining Engineer, from Onderstepoort, and the analysis was as follows:—

	Per cent.
Silica, SiO_2	0·80
Titanium dioxide, TiO_2 ...	15·20
Aluminium and chromium oxides, Al_2O_3 and Cr_2O_3	3·51
Lime, CaO	0·32
Iron oxide, Fe_2O_3 ...	81·43

Equivalent to metallic iron 57·00%.

This totals up to 101·26 but some of the iron is present as magnetic oxide, Fe_3O_4, and as is noted later, the state of oxidation of the titanium is open to question. It contained merely a trace of phosphorus and only ·11% sulphur.

The first experiment (No. 0) with the ore itself, was made in the hope of producing a slag of roughly the average composition of those instanced by Rossi in his paper of 1902* (which at that time was the only one I had read) as having been produced in practice. Clean silver sand was used to supply the silica, and marble the lime. The furnace employed was a Deville, intended for retort carbon and worked by forced draught. The charge was heated in a clay pot to between 1,400 and 1,500° Centigrade, Seger cones being used to indicate the temperature. On subsequent examination it was found that the slag had been too pasty to allow of the collection of the iron, which a polished section, examined under the microscope, showed to be pure and in the form of grains scattered through the slag, as shown in Fig. I. The dark looking grains are the iron, and the slag shows a very interesting structure, quite large crystals of a light coloured constituent being apparent. In the next experiment (1) the charge was modified to produce a slag of the CaO, SiO_2, TiO_2, type (sphene). This was heated to 1,530° C., and fused well, yielding a black slag and a button of metal which micrographically proved to be medium-high carbon steel, Fig. II., the yield being 39·5%.

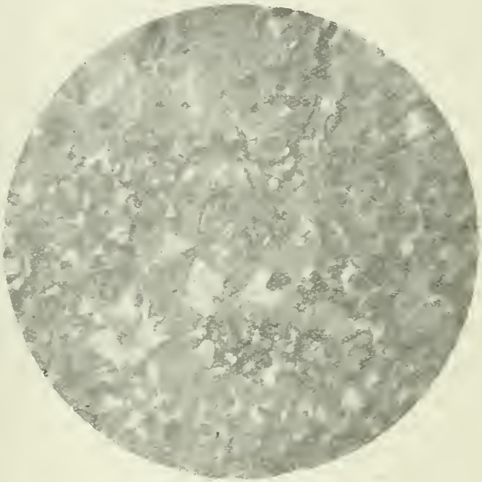


Fig. I.—×400 V.

In experiment (2) the proportion of carbon was increased in the hope of reducing more iron and increasing its carbon content; graphite crucibles were also employed in this and subsequent experiments, except where otherwise stated. The proportions of fluxes were also adjusted to give equal proportions of CaO, TiO_2, and SiO_2 in the slag. The result was poor, the iron, though in globules, being largely held up in the slag, though heated to the same temperature as before.

---

* "The Metallurgy of Titanium," read before the American Institute of Mining Engineers, May 15, 1902.

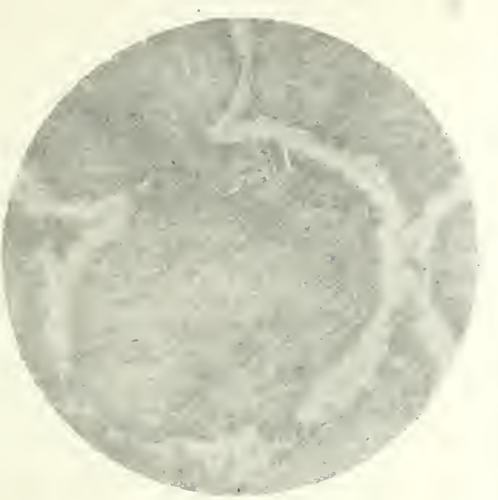


FIG. II.—×1,000 V.

Experiment (3) was then made with the proportions the same as in (1), except that carbon was increased; the result, however, was much the same, some excess carbon remaining with the slag, which was pasty, and the iron in consequence not so well collected. Up to this point coke had been used in the furnace; retort carbon was substituted for this in the next experiments in order both to obtain a higher temperature and to allow of the temperature being maintained for a longer time, and except with (5), which was a repetition of (2), the mixtures were made more basic. Experiments (4), (5), (6) and (7) were all heated to above 1,600° in the furnace at the same time, the Seger cone and a clay crucible which contained it being melted to a shapeless lump. In each case the slag escaped through the crucible, leaving buttons of grey iron varying from 12·5% to 52·5% of the ore taken. The proportions of SiO_2, TiO_2 and CaO are shown in the appended table, but experiment (7) was a repetition of (6), except that lime and magnesia was substituted for lime alone by using local dolomite as a flux. This was the mixture which yielded the highest recovery of iron. This seemed to indicate that the more basic mixtures made the best slags so far as iron yield was concerned, but of such a nature that the crucible was readily attacked. The higher temperature employed was doubtless an important factor also. These results therefore could not be considered conclusive, except that they showed that slags could be quite fusible, although containing high proportions of titanium, and consequently it was decided to examine next the fusibility or otherwise of various ideal mixtures containing titanium oxide, and for this purpose some TiO_2 was prepared from the ore itself. The various mixtures examined were prepared from this by mixing with pure precipitated silica and lime, and heated in platinum crucibles, embedded in magnesia inside clay crucibles in the Deville furnace, temperatures being now determined by a thermo-couple pyrometer. The first proportion tried, experiment (*a*), was that representing the composition of the mineral sphene, CaO, TiO_2. SiO_2, known to be fusible at furnace temperatures. This was heated to 1,400° C., and after removal found to have been perfectly fused, resulting in a medium brown sub-translucent glass. Experiment (*b*) was on a mixture containing equal parts of CaO, SiO_2 and TiO_2, and also fused well at 1,400°, but on cooling exhibited light spots indicating segregation of some constituent. In experiment (*c*) the total proportion of SiO_2 (taking into account that in the dolomite), was reduced to ·9 of the TiO_2, and instead of lime, dolomite was employed for the base, sufficient being taken to yield CaO and MgO equivalent to 1·65 parts CaO. This was evidently less fusible, and the segregation much more pronounced. In experiment (*d*) silica was increased again to equal the TiO_2, and mixed with 1·6 lime. This fused imperfectly at 1,400°. In experiment (*e*) the mixture was made up in the same proportions as in (*a*), but using dolomite to replace part of the CaO by MgO. This fused well, and appeared to be similar to (*a*). Experiment (*f*).—In this lime was omitted altogether, and the TiO_2 and SiO_2 mixed according to the formula TiO_2. SiO_2. This mixture, instead of being heated in the Deville furnace at first, was heated to 1,350°, the highest attainable temperature in a gas fired blast furnace. It did not fuse, but was strongly fritted. On subsequent heating to 1,400° in the Deville furnace it fused to a dark mass. Experiment (*g*): this time silica was omitted, and CaO and TiO_2 mixed according to the formula CaO. TiO_2. This fritted slightly at the same temperatures. Experiment (*h*) consisted in heating TiO_2 alone in the Deville furnace to 1,400°, when it melted imperfectly to a black mass. These results seemed to indicate that the titanium oxide behaved as a base quite as probably as that it acted as an acid, but at any rate they proved that the compound CaO. TiO_2. SiO_2 was the most satisfactory of those examined. The next experiment (No. I.) was therefore made with larger quantities of materials, using dolomite for base, proportioned to yield the slag CaO. TiO_2. SiO_2, and heated in a clay crucible to 1,600°. This resulted in the production of a button of steel weighing only 30% of the ore, the slag having run through the crucible. Experiment (No. I. *a*) was a repetition of this, except that a graphite crucible was used instead. The button of iron obtained was gray, and weighed 41·2% of the ore, but the slag had again escaped. This seemed to indicate that the slag was very basic in its behaviour, although from its composition, reckoning TiO_2 as an acid, it should have been acid. Therefore in Experiment (II.), to prevent so vigorous an attack on the crucible, the proportion of SiO_2 was increased, the proportion $TiO_2 : SiO_2 : CaO\ (MgO) :: 34 : 41 : 25$, and the crucible, of graphite, was only heated to 1,400° The result was a button of mottled iron weighing 52·4% and a glassy black slag which was poured off. This slag was analysed, and found to contain TiO_2 19·5%, SiO_2 37·5%, CaO 11·5%, MgO 6·3%, Al_2O_3 12·5%, and FeO 11·5%. Neglecting the oxides of iron and aluminium, the proportion of $TiO_2 : SiO_2 : CaO\ (MgO)$ were as 26 : 49 : 23, indicating that again the crucible had been attacked. Therefore, in the next charge, Experiment (III.), the proportion of silica was still further increased, to preserving the same order as before, to 31 : 46 : 23, and more carbon introduced with the object of trying to reduce more silicon and so render the iron gray. The temperature employed was 1,400°, as before. This was not quite so successful, the button of iron being white and weighing 50%, but the slag was still well fused, though not quite so glassy as in (II.) Its analysis was: TiO_2 23·0%, SiO_2 39·0%, CaO 10·0%, MgO 4·5%, Al_2O_3 8·5%, FeO 10·9%, showing that the attack on the crucible was practically stopped. Experiment (IV.) was another trial of a more basic slag at this temperature, to see whether, although attacking the crucible, the yield of iron might not be improved, since it was thought that the high silica content might be keeping iron in the slag. The proportions used were 37 : 37 : 26; the iron obtained was of the same kind and weight as in (III.), but the slag was not very satisfactory, being rather pasty. It was not analysed. Experiment (V.) was similar, except that the proportion of silica was slightly increased, the amounts being 36 : 38 : 26. The temperature was probably rather higher than with (IV.), but was not determined on account of the pyrometer protecting tube having been accidentally broken. The button of iron obtained, which was white, weighed 46·8%, the slag being fairly well fused, but holding beads of iron. Experiment (VI.) was an attempt to make an acid slag and gray iron, by slightly increasing silica and cutting down lime considerably. Part of the slag, however, escaped through the pot, leaving a pasty mass behind, also a button of iron weighing 44%. Experiment (VI.*a*) was a repetition which behaved exactly in the same manner, though leaving 44·8% iron. The pasty mass was analysed, and found to consist of 32·8% TiO_2, 34·2% SiO_2 5·6% lime, and by difference, 23% of iron oxide, magnesia and alumina. In Experiment (VII.), while the lime was kept as in (VI.), the silica was reduced to less than the TiO_2. This, using coke as fuel, was only pasty at 1,300°, but on reheating to 1,400°, using retort carbon again, it melted. In (VII.*a*), therefore, this charge was repeated, and yielded 46% of iron and a well melted slag.

By this time it appeared certain that titanium oxide always acted as a base, and assuming this to be so, the most satisfactory slags were not far removed from being monosilicates of titanium and the other bases, excess silica in the charge having been brought down to this level by keeping up some iron oxide, etc. So far, although the charges had in most cases been heated to fairly definite temperatures, it had only been possible to observe whether the charge had melted or not at those temperatures, and the next experiments were directed towards determining the melting points of the compounds formed. The method employed, which I believe is novel, was developed ultimately on account of the difficulty of obtaining arrest points in cooling with such small quantities, and consisted simply in fusing the mass in a platinum crucible in a gas blast furnace, inserting the bare couple wires and allowing to solidify. Matters were then so arranged that the crucible and contents were suspended from the couple wires inside the furnace in such a position that they could be seen. The temperature was then raised again, observing the galvanometer deflections continuously, till the crucible, owing to the slag melting, suddenly dropped off, and this point was taken as the melting point. Whether this point would agree with that obtained by a cooling curve method I have had as yet, no opportunity of testing, but it certainly appears to give very nearly the temperature at which the slag liquifies, and that, after all, is the important point in the present connection, and further, the determination is quite simple, and when the apparatus is set up can be made in a few minutes.

The artificial slag (*a*) was first examined and melted off at about 1,150° C.; on repeating this determination, however, it dropped off at a considerably lower temperature, and two or three repetitions led to similar discordant results. In seeking for an explanation, it was noticed that the colour of the slag, when cold, varied considerably between almost white and a deep brown, accompanied by a stony fracture when white and a glassy fracture when brown, in different experiments. A little investigation soon showed that the dark colour was the result of heating in a reducing atmosphere, and that this was accompanied by a lowering of the melting point. After this it was easy, by prolonged heating in a reducing atmosphere, to lower the melting point to 1,000°, while reoxidation raised it to 1,250°. and this was confirmed several times. Now it seemed that the only reducible body present was TiO_2, unless that used and supposed to be pure was considerably contaminated with iron oxide. The iron in this slag was therefore determined, and found to be present to the extent of 1·25% only, which I do not think is nearly sufficient to account for the results obtained. It seems certain then that titanium may exist in slags in different states of oxidation, and this is apparently the explanation of a good many peculiar results previously obtained. In a blast furnace slag, naturally formed under strong reducing conditions—it most probably does not exist as TiO_2, but more probably, or to a large extent, as a lower oxidised compound which acts as a base, and is, besides, more fusible itself than TiO_2, since TiO_2 fused in a former experiment on heating in the reducing atmosphere of the Deville furnace. Slag (*b*), though less fusible, behaved quite similarly; thus, at the first attempt it fused at 1,350°, while after oxidation it would not fuse at 1,400°, at which temperature it had been prepared; strong reduction lowered it to 1,140°; (*c*) examined similarly melted under ordinary conditions at 1,396°, and after reduction (probably partial, as it was done hurriedly) at 1,360°, at which it was allowed to rest and not examined further. Reference to the diagram shows that this contained very high lime, *i.e.*, too basic in character, but if TiO_2 were present and acting as an acid it might have been expected to be more easily fusible.

The next point for consideration was the composition of this lower oxide; does it exist as TiO or Ti_2O_3? Here, I confess, I have reached no conclusion, but think it is most probably Ti_2O_3. Rose, according to Roscoe and Schorlemmer, states that in ilmenite the iron is present as Fe_2O_3 and the titanium as Ti_2O_3, not TiO_2. The isomorphism of ilmenite and hæmatite and the fact that when dissolved in acid a violet colour may be obtained, lending support to this view, which in fact is widely accepted. It is further stated to be very difficult to reduce as far as TiO, though Ti_2O_3 may be produced by heating TiO_2 in hydrogen, and on dissolving in acid gives this characteristic coloured solution. Now if a lower oxide is formed, a loss of weight should occur on reduction and a gain on oxidation; this I have proved experimentally, though so far it has not been possible to do so very exactly, and the gains and losses observed have certainly been less than should occur on complete conversion from TiO_2 to Ti_2O_3, and *vice versa*, though much more than would be required by the small amount of iron found to be present. Further, on treating samples of these reduced ideal slags with strong sulphuric acid and allowing to stand, a violet colouration was distinctly noted in the solution, which persisted on dilution, and was destroyed by addition of oxidising agents such as nitric acid. Some of the slags from the iron ore itself also afforded evidence of the same effect, though considerably masked by the iron present.

A reference to the analyses of earlier succesful slags (II.) and (III.) shows that reckoning the titanium as Ti_2O_3 and a base, the oxygen ratios in base and acid are almost unity, being practically 1 : 1 in (II.) and 1 : 1·1 in (III.), *i.e.*, they are monosilicates (or, according to Rossi's classification, bi-basic) with a slight excess of silica. To test this point further a small charge was made up (*i*) to produce $2TiO, SiO_2$ and heated in a platinum crucible as before; this was imperfectly fused, and it seems very unlikely that TiO is produced. On the other hand, the (*f*) experiment instead of giving TiO_2, SiO_2 was probably, at least in part, Ti_2O_3, SiO_2 and had fused. An attempt was next made to form in experiment (*j*) a bisilicate $Ti_2O_3, 3SiO_2$, this, however, destroyed the platinum crucible, the bottom having been mostly melted out, and part of the slag run into the magnesia leaving a porous mass behind. The cause of this was most probably that the conditions had been so reducing that some silicon was reduced and combined with the platinum, but for want of time this has not been proved. Not caring to risk another crucible of platinum just then, I reverted to the ore charges in graphite pots.

In Experiment (VIII.) the charge was proportioned to yield a slag $3CaO, Ti_2O_3, 3SiO_2$. This fused well, giving a good glassy slag, rather vesicular at the top and yielded 51·5% iron. The fuel was coke, and as the temperature obtained was not considered satisfactory (the pyrometer of this furnace being still out of order), Experiment (VIII*a*.) was made with the same proportions, but using retort carbon as fuel. This charge was accidentally spilt while pouring, but nevertheless 50·4% of iron was obtained, accompanied by a good slag. This was felt to be sufficiently encouraging to justify a repetition, and in Experiment (VIII*b*.) 52·4% of iron was recovered, but part of the slag escaped by boiling over. To test whether this might not have been due to reaction occurring with the graphite of the pot, two attempts were made in clay pots which unfortunately resulted in melting everything, pots and lids included. Smaller charges being taken. A third trial was then made, (VIII*c*.) controlled with a pyrometer again, and the crucible kept at

1,200° only ; a well fused slag was obtained but at the expense of a low yield of iron. The dolomite was by this time exhausted so that equivalent lime was intended to have been substituted in this charge, but inadvertently limestone was weighed out instead, and consequently the lime proportion was lower than was intended, thus rendering the slag acid. Still, as this charge had fused, a small charge of the same kind, (VIII*d*), was tried again in the same sized graphite pot as before, hoping to thus prevent any loss by boiling over and allowing a higher temperature than with the clay pot. This was heated to 1,500°, and the slag was more vesicular but the iron yield increased to 48·5%. (VIII*e*.) was another trial with the same proportions as in (VIII.), but using smaller quantities, and lime again instead of dolomite, the slag was well fused and liquid, and although some iron was lost during pouring the remainder weighed 42%. The iron from all these latter trials had been white, so in the last Experiment, (IX.), a more acid mixture was tried again, strongly heated and the melt allowed to cool in the crucible. The slag appeared to have been well melted, and the iron, which was grey, weighed 48%. Both were easily detached from the interior of the crucible and left it ready for use again. Both these last slags were magnetted after crushing and found to contain a considerable amount of finely divided iron, (VIII.) more so than (IX.), the fusions having been made rather hurriedly and possibly insufficient time allowed for it to settle out of the slag ; the analyses also show high proportions of ferrous oxide. Last of all, in Experiment (*k*) a mixture of the pure materials in the proportion $3CaO$, Ti_2O_3, $3SiO_2$, was made up and heated in a platinum crucible under reducing conditions in a gas blast furnace to 1,380°. It fused perfectly, and its melting point was then determined by the method described, to be 1,230° ; on attempting to repeat the determination, however, the crucible was destroyed exactly as in Experiment (*j*).

The calculated proportions of CaO (including equivalents of MgO and Al_2O_3 present) silica and titanium oxide (expressed both as TiO_2 and Ti_2O_3 have been plotted graphically in the diagram, keeping the titanium constant. The two inclined lines *ab* and *cd* are so arranged that the lengths or ordinates intercepted between them and the datum represents proportions of CaO and SiO_2 required to form a monosilicate in conjunction with the constant percentage of titanium. These were obtained by calculating the proportions present in $2CaO$, $2Ti_2O_3$, $4SiO_2$ and $6CaO$, $2Ti_2O_3$, $6Si_2O$, and setting them off to scale from the convenient points *x*, *y*, on the datum. When considering any particular case in the experiments plotted, therefore, all that is necessary to ascertain the degree of closeness with which the theoretical slag approaches a monosilicate, is to select either the CaO or SiO_2 point, observe where the line through it parallel to the datum cuts the particular CaO or SiO_2 line, follow the ordinate through that point to the SiO_2 or CaO line and read off the proportion of SiO_2 or CaO required, which may then be compared with the amount theoretically present in the slag.

Thus, slags (4), I., IV., V., VII., VIII., VIII*b*. and VIII*e*., are not very different from monosilicate and are all fusible ; and while, at the same time, it is clear that considerable variations in composition may still result in fluid slags, within the limits dealt with it is safe to say that a monosilicate slag will be fusible. Slag VII., for instance, is rather more acid than a monosilicate, and to produce it, for every unit of TiO_2 in the ore about ·35 unit of lime or say ·6 or ·7 dolomite, and ·9 unit silica must be used as flux, or a total of 1·5 units. Therefore an ore containing 15% TiO_2 will only require roughly 23% of its weight of flux and will produce nearly 40% of its weight of slag, and each ton of iron from a 57% ore will be accompanied by about ·7 ton of slag—quite a moderate quantity when it is remembered that twice this quantity is frequently formed. The slag would doubtless be perfectly fluid at blast furnace temperatures and is high enough in SiO_2 to produce a grey iron, but if this were not needed, some silica could be cut down, or lime brought up, without much risk of cutting the hearth walls.

In Rossi's blast furnace experiments, in which he made highly titaniferous slags, the corrosion of the furnace lining, even after a short run, was considerable, pointing again to the basic character of the high titanium slag and furnishing a strong reason for intentionally using silica as a flux, to produce a silicate and not relying on titanium to form titanates. An interesting point may be referred to here also ; if a blast furnace manager in the old days found his slags becoming sticky it is more than likely that he would use more lime, on general principles, and certainly if he knew his ores contained titanium and thought it acted as an acid. Now if the titanium acted as base, this would merely accentuate the evil by making the slag too basic, and the resulting troubles would have been due to lime rather than to titanium : it seems very possible that titanium may have earned some, at least, of its bad reputation in this manner.

To resume, under conditions such as exist in the Transvaal, with silicious iron ores abundant, these would naturally furnish the bulk of the silica required, though the dolomite, which is always more or less silicious would also furnish some. Still it is perfectly obvious that by mixing the two classes of iron ore, a mixed ore could be produced, which is not far from being self-fluxing. Approximately half a ton of a 30% silica, iron ore would furnish all the silica required by a ton of the 15% titaniferous ore and this 1·5 ton of mixed ore would only require about 2 or 3 cwt. of non-silicious dolomite. The net result would be that having so little slag to make and melt, the cost of flux and fuel would be likely to be very low, and the cost of handling and disposal of slag also low for the same reason.

Further, with regard to the coke, if its mechanical properties are satisfactory, a comparatively high percentage of ash would not be a very serious drawback as it would, very possibly, be of such a nature to help with the fluxing: but with a good quality the amount used would doubtless be nearer the ton than the one and-a-half tons previously mentioned as being possibly required, even allowing for the reduction of TiO_2.

But even in places where no suitable coking coal exists, it is still open to experiment to determine whether or no non-coking coals (resembling the Scotch splint coals, or else anthracitic in character), which exist in many places, could be satisfactorily used in place of coke, as is frequently the case in smelting other classes of iron ores. Such coals occur in South Africa, and if so used, the bye-products obtainable in the shape of tar, oils, ammonia compounds, and so on, would doubtless meet with a ready sale.

With regard to electric smelting: mixtures made up to produce monosilicates have been experimented upon and found to smelt perfectly, as far as could be judged in the small 50 amp. furnace employed, but, as I pointed out previously, slags which are very refractory in ordinary furnaces can be readily melted at the higher temperatures which can be generated electrically.

In conclusion, I may summarise as follows:—1st, with regard to the slags. The titanium, in all probability, exists in the slags wholly or partly as Ti_2O_3, which acts as a base, at any rate when sufficient silica is present also. I think it is quite conceivable, however, that, like alumina, it may act as an acid in very basic slags, but naturally renders them scouring and liable to attack brickwork, with which they come in contact. If the slags are calculated to form monosilicate with the proportion of silica about the same as Ti_2O_3 (assuming all titanium to be present in this form) they will be fusible and fluid. Second, with regard to the iron. This, in most cases, was white, and when grey was obtained it seemed to be as often a consequence of extra high temperature as of extra silica in the slag, though it was demonstrated that by increasing this a grey iron could be produced.

White iron is, of course, well adapted for forge purposes (making puddled iron) and for open hearth steel furnaces, but as such is not usable in any quantity in foundries, and it was in order to see whether a grey iron could be made that some of the variations in charges were made. Several analyses of the specimens of iron were made, the white showing from traces to ·104 silicon and little if any titanium. The greys contained from 1·8 to 3·2 silicon, and though in general there was scarcely any determinable titanium, in those (Nos. 4, 5, 6 and 7) which had been heated to very high temperatures 2·8% to 3·48% was found. Some of the white iron was melted and "physicked" by adding a little ferro-silicon and thereby converted to grey iron, and in all probability the only cause of the formation of white iron was the non-reduction of silicon owing to the comparitively low temperature employed. All the buttons were extremely strong under the hammer, and in fact it is universally admitted that titaniferous iron is of excellent quality. Samples showing the different grades are exhibited on the table; also a small hammer cast in the laboratory from the various buttons obtained. Figs. III. and IV. show the microstructure of the grey and white iron respectively.

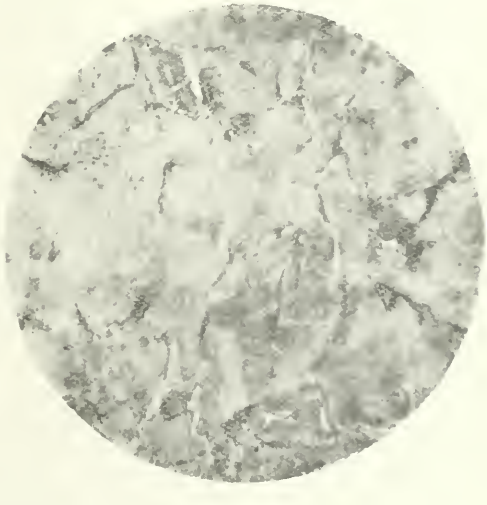


Fig. III. — × 400 V.

I trust I have been able to demonstrate that very useful results have been obtained from these small tests but, of course, it does not follow that they would be duplicated in the blast furnace, one very important difference in the conditions being that in the latter nitrogen and other gases are passing through the materials all the time. For that reason I have had a small blast furnace erected, and propose to make further tests, the results of which will be communicated in due course

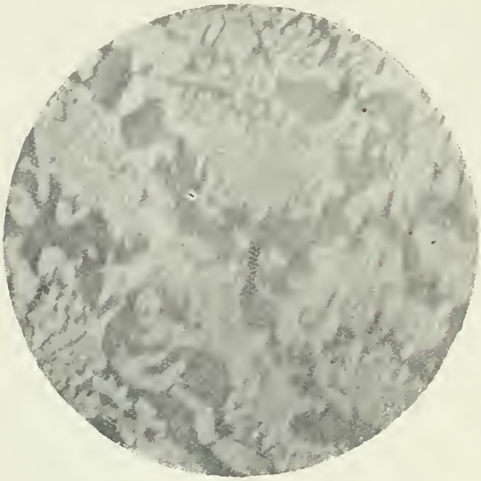


Fig. IV.—×400 V.

I cannot conclude without making reference to the valuable assistance rendered by Mr. Murray during the progress of these experiments, the greater part of the actual manipulations having been carried out by him, Mr. Moon has also assisted, and to both of them I have to express my thanks.

APPENDICES.

(*a*) Slag diagram and table (see p. 171).

(*b*) Analysis of dolomite used.

	Per cent.
CaO	36·12
MgO	15·05
Al_2O_3 and Fe_2O_3 ...	1·05
SiO_2	4·18

(*c*) Method of analysis of slags.

The method adopted, after trying many others, was as follows:—·5 gm. of the slag crushed to – 80 was weighed out and placed in a nickel basin containing about 2 gm., say, $\frac{3}{4}$ in. stick of caustic soda previously fused and allowed to solidify in order to minimise risk of loss by spitting. This was then held over a Bunsen flame by means of crucible tongs and melted, and when melted kept in gentle agitation for about two minutes, then allowed to cool, and so manipulated while cooling, as to spread the melt up the sides of the basin in as uniformly thin a layer as possible, in order to facilitate subsequent dissolution.

When solidified it was cooled in water and then nearly filled with water. After a few minutes the sides were scraped down with a nickel spatula, and the cloudy liquid transferred to a beaker (Jena 400 c.c. affords a convenient bottom area) and the dish cleaned out in the usual manner, then acidulated with strong hydrochloric acid and placed on the hot plate.

The bulk should not exceed 50 c c. (to save time in evaporation), and as soon as the solution becomes hot it clears, so that a perfect solution of the slag is obtained in less than 20 minutes from weighing out. After going to dryness and baking it was taken up as usual with hydrochloric acid, filtered and washed. The precipitate contained the silica, together with a variable amount of titanic oxide. After weighing, therefore, it was evaporated with hydrofluoric and sulphuric acids, ignited, and the residue of TiO_2 weighed, and SiO_2 determined by loss.

The filtrate was neutralised with ammonia and reacidified very slightly with hydrochloric acid, diluted to 300 c.c., some crystals of sodium sulphite added (not enough however to destroy free acid), and boiled for two hours, adding a little more sulphite occasionally, and also taking care to keep acid and so prevent precipitation of alumina, using a few drops of methyl orange as indicator.

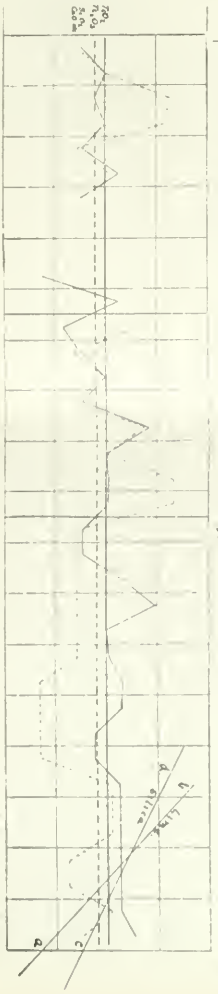
The hydrated TiO_2 was then settled, filtered, and finally weighed as TiO_2, and added to that from the SiO_2 evaporation. In the filtrate after oxidation by HNO_3, iron and aluminium hydrates were precipitated together and weighed together as Fe_2O_3 and Al_2O_3. Lime and magnesia were determined in the filtrate as usual. Iron was determined in a fresh portion: being obtained in solution in the same manner, except that sulphuric acid was used throughout instead of hydrochloric; diluted, reduced by nearly neutralising and boiling with sodium sulphite, then titrating with permanganate in the usual manner. Alumina was then determined by difference, but it is advisable to examine the FeAl precipitrate for titanium.

The Chairman: I beg to propose a very hearty vote of thanks to Prof. Stanley for his most valuable paper. If these large bodies of iron ore in the Transvaal can be successfully treated, it will be of the greatest importance to the Transvaal and South Africa. The time has come when an industry to meet some of the large iron requirements of the country, particularly of the Witwatersrand, should be started. I hope the efforts of the Government to promote this iron industry will meet with success. I should like to ask our visitors, Messrs. Harbord and Cobbe, whether they would not like to contribute to the discussion of this paper this evening.

Mr. F. W. Harbord (*Visitor*): I have listened with the greatest pleasure to this paper by Prof. Stanley, and it was of particular interest to me, coming out here as I have done to look into the question generally of your iron ore resources.

The smelting of titaniferous ores has never, in my opinion, been systematically thrashed out,

APPENDIX A.



	Weights Taken.					Theoretical Composition of Slag.			Analysis of Slag.						Iron.		Slag.	Temperature.	Remarks.
	Ore	Carbon.	TiO_2.	Total Silica.	Lime and Lime equiv. of other Bases.	Ti as TiO_2.	SiO_2.	CaO, etc.	Ti as TiO_2.	SiO_2.	CaO.	MgO.	FeO.	Al_2O_3.	Per cent. Yield	Nature.			
(a)	—	—	·8	·6	·56	41	30·5	28·5	—	—	—	—	1·55	—	—	—	Perfectly fused	1,400	Sphene
(b)	—	—	1·0	1·0	1·0	33	33	33	—	—	—	—	—	—	—	—	,, ,,	,,	Shows some segregation
(c)	—	—	1·0	·9	1·65	28	25	47	—	—	—	—	—	—	—	—	Well fused	,,	Shows segregation, dolomite used
(d)	—	—	1·0	1·0	1·6	28	28	44	—	—	—	—	—	—	—	—	Imperfectly fused	,,	
(e)	—	—	1·0	·75	·7	41	31	28	—	—	—	—	—	—	—	—	Well fused	,,	Similar to (*a*), but dolomite used
(k)	—	—	·32	·36	·336	29·3 (Ti_2O_3)	36·5	34·2	—	—	—	—	—	—	—	—	,,	1,380	Gas furnace used—crucible subsequently destroyed on reduction
(f)	—	—	1·0	·75	—	57	43	—	—	—	—	—	—	—	—	—	Fritted, Imperfectly fused	1,350 1,400	Gas furnace used
(g)	—	—	·8	—	·56	59	—	41	—	—	—	—	—	—	—	—	Fritted	1,400	Dark mass—Deville furnace
(h)	—	—	1·0	—	—	100	—	—	—	—	—	—	—	—	—	—	Melted imperfectly	1,400	Black mass
(i)	—	—	·8	·3	—	73	27	—	—	—	—	—	—	—	—	—	,,	1,400	
(j)	—	—	·8	·9	—	47	53	—	—	—	—	—	—	—	—	—	Portion melted, lost	1,400	Fritted residue, crucible destroyed
0	20 gr.	3·4	3	1·75	4	34	20	46	—	—	—	—	—	—	—	Not collected	Pasty	Between 1400 1500	Clay crucible used
1	,,	5	,,	2·28	2·1	41	31	28	—	—	—	—	—	—	39·5	Steel	Well fused	1,530	,, ,,
2	,,	6·5	,,	3·05	3·0	33	34	33	—	—	—	—	—	—	—	Not collected	Poor	,,	Graphite crucible, excess carbon
3	,,	6·5	,,	2·28	2·1	41	31	28	—	—	—	—	—	—	—	,,	Rather pasty	,,	,, ,, ,, ,,
4	,,	5	,,	4·05	4·0	27	37	36	—	—	—	—	—	—	47·5	Very grey	Well fused	1,600	,, ,, slag escaped
5	,,	5	,,	3·05	3·0	33	34	33	—	—	—	—	—	—	42·5	,,	,,	,,	,, ,, ,, ,,
6	,,	5	,,	3·15	5·0	27	28	45	—	—	—	—	—	—	48·5	,,	,,	,,	,, ,, ,, ,,
7	,,	5	,,	3·15	5·0	27	28	45	—	—	—	—	—	—	52·5	,,	,,	,,	,, ,, dolomite as flux
i	250 gr.	40	37·5	28·5	27·5	40	31	29	—	—	—	—	—	—	30	Steel	,,	1,600	Clay crucible, slag escaped, dolomite
ia	,,	40	37·5	28·5	,,	40	31	29	—	—	—	—	—	—	41·2	Grey	,,	1,600	Graphite ,, ,, ,, ,,
ii	,,	40	,,	44·4	,,	34·4	40·5	25·1	19·5	37·5	11·5	6·3	11·5	12·5	52·4	Mottled	Well fused, glassy	1,400	,, ,, ,, ,, ,,
iii	,,	40	,,	54·4	,,	31	46	23	23·0	39·0	10·0	4·5	10·9	8·5	50·0	White	,,	,,	,, ,, ,,
iv	,,	40	,,	37·4	,,	37	37	26	—	—	—	—	—	—	50·0	,,	Rather pasty	,,	,, ,, ,,
v	,,	,,	,,	39·4	,,	36	38	26	—	—	—	—	—	—	46·8	,,	Fused	1,450?	,, ,, ,,
vi	,,	,,	,,	43·15	13·2	40	46	14	—	—	—	—	—	—	44·0	Mottled	Part fused, pt. pasty	,,	,, ,, slag partly escaped ,,
via	,,	,,	,,	43·15	,,	40	46	14	32·8	34·2	5·6	?	?	?	44·8	Mottled	,,	,,	,, ,, ,, ,, ,, ,,
vii	,,	,,	,,	33·15	,,	45	10	16	—	—	—	—	—	—	?	White	Pasty Melted well	1,300 1,400	,, ,, ,,
viia	,,	,,	,,	33·15	,,	45	40	16	—	—	—	—	—	—	46	Mottled	Melted well	1,400—1,500ca.	,, ,, ,,
viii	,,	,,	,,	42·2	39·3	29·3 (Ti_2O_3)	36·5	34·2	—	—	—	—	—	—	51·5	White	,,	1,400—1,500ca.	,, ,, ,,
viii*a*	,,	,,	,,	,,	,,	,,	,,	,,	—	—	—	—	—	—	50·4	,,	,,	1,500	,, ,, ,,
viii*b*	,,	,,	,,	,,	,,	,,	,,	,,	—	—	—	—	—	—	52·4	,,	Fused well	1,500	,, ,, boiled over ,,
viii*c*	100 gr.	16	15	17·0	9·4	36·3	41·0	22·7	—	—	—	—	—	—	low?	,,	Fused	1,200	Clay crucible Limestone
viii*d*	,,	,,	,,	17·0	9·4	36·3	41·0	22·7	—	—	—	—	—	—	48·5	,,	Well fused	1,500	Graphite ,, ,,
viii*e*	,,	,,	,,	17·0	15·8	31·4	35·5	33·1	22·0	27·4	18·7	—	24·1	5·9	42	,,	,,	1,500	,, ,, spilt some iron ,,
ix	,,	,,	,,	20·0	11·2	32·5	43·3	24·2	24·0	33·7	16·0	—	16·5	6·9	48	Grey	,,	1,600	,, ,, ,,

and I have always thought that the prejudice against using these ores has not been fully justified. At home I have repeatedly tried to persuade iron masters to use small quantities of such ores in their blast furnaces, gradually increasing the quantity should the result prove satisfactory. The large scale experiments in America conducted by Mr. Rossi, referred to in the paper, were, so far as the results published show, successful from a technical point of view. But whether due to exceptional wear and tear in the lining or to other causes, these experiments were not commercially successful, the result being that no one in America has followed on similar lines.

I have discussed the matter at different times with people in America, and have always been met by the statement that Rossi failed commercially. The ordinary blast furnace manager always insists that a fluid slag cannot be maintained, and the furnace kept open, if titaniferous ores are used, but I have never met an English manager who was able to substantiate this statement as the result of his personal experience extending over any reasonable period. I know one large plant which has been using ores containing small quantities of titanium (about 1%) for years without any trouble, and although it would not be safe to rely solely upon ores containing considerable quantities of titanium when starting a new industry, yet when once pig iron was being made satisfactorily from other ores, small quantities of titaniferous ores could, I think, be used with perfect safety.

As Prof. Stanley has pointed out, you cannot duplicate blast furnace conditions in a crucible, for in a blast furnace the conditions are constantly varying according to the way the furnace is working, but you can, as he has done, demonstrate the effect of varying quantities of titanium on the fusibility of slags of different composition. The most important point raised in the paper is whether oxide of titanium acts as a base or as an acid, and I am inclined to think the real trouble has been that under different conditions it some times acts as an acid and at other times as a base, and the exact conditions which determine how it will act in relation to silica in the blast furnace have never been accurately determined. The usual assumption has been that it acts as an acid in the same way as silica, and that it is therefore necessary to add lime to flux it, and this may have been the cause of a great deal of the trouble experienced with these ores. The paper is one of greatest interest, and I shall follow most carefully the results Prof. Stanley obtains from the larger experiments he has in view, and these I am sure will not only be of interest here but to all iron-producing countries. I hope he may be able to conduct these experiments on a sufficiently large scale to arrive at some definite practical conclusion as to the behaviour of these ores in the blast furnace, for it is a most important matter to the Colony, and I therefore hope that Prof. Stanley will have the support of every one in the country when carrying out this work.

Mr. A. F. Crosse (*Past-President*): I should like to remark that in my young days, when chemist to Messrs. Johnson, Matthey & Co., we frequently sampled and analysed whole cargoes of titaniferous iron ores from Norway, which were sold in England.

TREATMENT OF ORE SLIME.

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

The chief difficulty in treating slimy material with a cyanide solution is the difficulty of removing and obtaining the dissolved gold, or silver, from the slime. Many methods have been tried, and the methods chiefly in use to-day are the well-known decantation process and various filter-press processes. The decantation is very slow, and therefore requires an enormous plant, and a very large quantity of solution has to be handled in proportion to the weight of slime treated.

The filter-press method has many advantages, but it requires a costly plant, requiring much handling and constant renewals. In my process the slime, having been collected in a suitable manner, is washed with a weak cyanide solution into a conical treatment vat. This vat is provided with an interior conical baffle, open at the bottom, and reaching about half way down the treatment vat. Inside the vat is a pipe extending, from nearly the bottom of the cone, to a foot or two above the surface of the pulp. This pipe has a small pipe introduced at the lower end, through which air is forced; this causes a current of concentrated or thickened pulp to be forced up the tube. At the same time the clear liquid, in the outer portion of the conical vat, is overflowing into the zinc box where the gold and silver are deposited, and then this clear solution, deprived of its gold and silver, is pumped by a small pump, which can be regulated as to speed, back into the top of the inside cone, where it mixes with the underflow brought up by the air lift pump. By this means the whole mass is kept in continuous circulation under the most advantageous condition for the solution of the gold or silver; as the air circulates and ærates the pulp, the gold and silver are being continually extracted from the solution, the same solution acts as a wash, practically all the gold and silver having been extracted and deposited in the zinc box. During the working of the process it is very easy to control the strength of the cyanide solution or the alkalinity, and should either be found too low, more cyanide or alkali can be added.

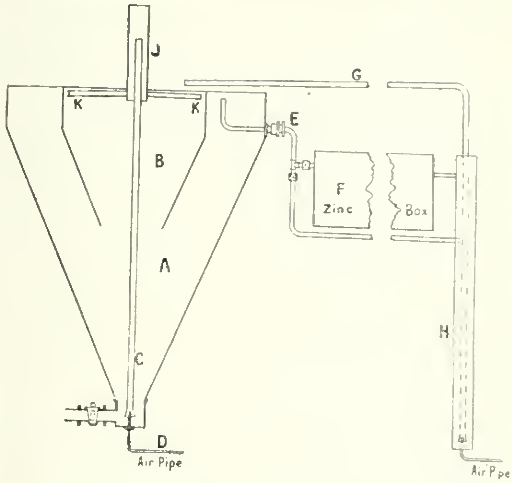
Clear solution containing gold is passing through zinc boxes and depositing its gold in every cyanide plant. The circulation of cyanide solution and slime by means of an air lift pump is quite the ideal method of solution, as I have proved in hundreds of trials. Most gold ores produce on crushing a certain proportion of real slime, which is usually a hydrated silicate of alumina with an excess of silica, or a very finely divided hydrate of ferric oxide, and finely ground pyrites.

We metallurgists all recognise to-day that no hydro-metallurgical gold treatment can be successful unless the ore has been sufficiently finely crushed, in order to render every particle of gold or silver accessible to the solvent action of the cyanide solution. In order to carry out this, tube mills, pans and other devices have been introduced, so that a large proportion of the crushed ore *cannot* be treated by percolation. In my scheme I can handle any fine material, and I can treat successfully either ordinary argillaceous slime, fine sand, or finely ground pyrites. I have treated a highly ferruginous slime and obtained an extraction of 93·2% of the gold contents from slime containing 7 dwt. fine gold per short ton. This means that the gold so obtained was precipitated on the zinc. This same slime was treated by a filter press plant, and only gave extraction of 61% of the gold; this was owing to the nature of the slime, as it contained so much ferric hydrate, and it was impossible to wash out the dissolved gold from the pressed slime.

The cost of erecting a plant according to my plan, would be very much less than any other that I know of, as it merely requires the ordinary conical vats, pipes for air lift pumps, zinc boxes, and an air compressor. The mechanical efficiency of an air lift pump is quite equal to other pumps for slime pulp, as the proportion submerged is very large in proportion to the actual lift required. Very little labour is required to run the plant, and one attendant could look after several vats.

As regards the strength of the cyanide solution and consumption of chemicals, that would depend on the nature of the ore, and would be neither more or less than that required by other methods.

My plant (see diagram) consists essentially of a conical vat, A, having within it a cone-shaped baffle, B, and a pipe, C, up which a jet of air is directed from the small pipe, D; a further pipe, E, leads from near the top of the vat A into the zinc precipitating box, F, which is provided with a pipe, G, through which a small air-lift pump, H, returns the liquor from the zinc box to the inside of the baffle B.



Section Showing Essential Parts.

The method of operating the plant is as follows:—The slime, having been collected by any suitable method and the excess of water decanted off, is washed by means of a weak cyanide solution into the treatment vat A through the baffle B. Further weak cyanide solution is then added and air turned on through the pipe D. This forces up through the pipe C into the receiver J a mixture of slime, dilute cyanide, and air, to a point a foot or two above the level of the liquor in the vat A to which it returns through the pipes K, which are so arranged as to give their discharge a circular motion within the baffle cone B. The slime then settles down continuously, reaching the bottom of the cone in the form of a thickened pulp, and is again lifted through the pipe C. The pulp occupying the space between the baffle cone and the wall of the cone A is in a state of comparative quiescence, the only movement being a gentle upward flow of the enriched solution to the pipe E, on its way to the zinc box. The conical shape of the vat A reduces the rate of upward flow and insures that the top few inches of liquor are clear and free from slime, and thus are in a fit state for precipitation. The solution deposits its gold content in the zinc box and is then returned by the pump H to the inside of the baffle where it continues its work, and the sequence above described is allowed to proceed until a large proportion of the gold has been precipitated.

Having described my slime process, I would now like to go into further details.

I have made a considerable number of experiments as to replacement of the solution when the pulp is being agitated in the conical vat, as described. The method I used was to have a weak chloride of sodium solution and no cyanide.

I found that with the vat constructed, as shown in the diagram, and having a pulp containing 20% of dry slime, or 1 – 5, I could obtain a clear overflow of 18% of the contents of the conical vat per hour, and that when a volume of solution equal to the solution in the vat had been replaced by water running into the inner cone B, that 75% of the chloride of sodium had been removed. I obtained this result several times with different slimes, both in London and in the Transvaal. The same thing happens with gold or silver in solution, and as the solution, when it is returning from the zinc box, pours into the inner cone, the gold-bearing solution is taken off by the decanting arm, as described.

I want to draw your attention to one point. In beginning the operation with a solution not containing gold, the upper portion of the solution which begins to overflow would not contain any gold for some time, and it would be useless to pass it through the zinc box, therefore, in working this process I would recommend circulating for about two hours before allowing any overflow to take place. This is very easily arranged, as the pulp in the inner cone B is concentrated, and has a much higher specific gravity than in the outer portion; the surface of the liquid is several inches below the level of the outer portion ; by means of four or five 2 in. holes just below the surface of the outer portion the clear solution flows into B. When this circulation has proceeded long enough the holes are plugged with wooden plugs, and the clear overflow is allowed to pass through the zinc box and return to the inner cone free, or nearly free, of gold and silver. After this circulation has been started, it will be apparent that at any given time the upper portion of the solution in the outer vat, between the inner and outer cone, is the richer in gold. Therefore, if the pulp be allowed to settle in the vat, the super-natant liquid is much richer in gold than that portion in the settled pulp. One can, if one likes, replace a strong cyanide solution by a very weak one. I treated some slime in London (from Nundydroog Mine, Mysore, India) by my process, and after seven hours' treatment the unwashed residue assayed ·07 dwt. fine gold per ton. The original slime assayed 2·5 dwt. fine gold. This slime was slightly pyritic. There is, as you all know, even in slime, an insoluble minimum of gold, and it would be an interesting problem for some of our younger members to find out in what condition this very finely divided gold is, and to explain why it is not soluble in cyanide solution.

It is quite evident that the capital outlay required for a plant which requires so short a time for treatment (the time for settlement must or course be added) is very much less than the present plants in use on these fields. This plant is both easy and inexpensive to work. If the overflow is not absolutely clear, it will have to be passed through a clarifier of some kind ; this method is usual to-day on most slime plants.

A plant has been erected on the above principle and has been working for fourteen months on the Crown Mines, giving excellent results, which will be described in the paper which follows, so that it will be evident that my process has now passed the experimental stage, as a proof of which I append some notes by Mr. Fraser Alexander on the Treatment of Black Sands.

ADDENDUM ON THE TREATMENT OF BLACK SANDS,

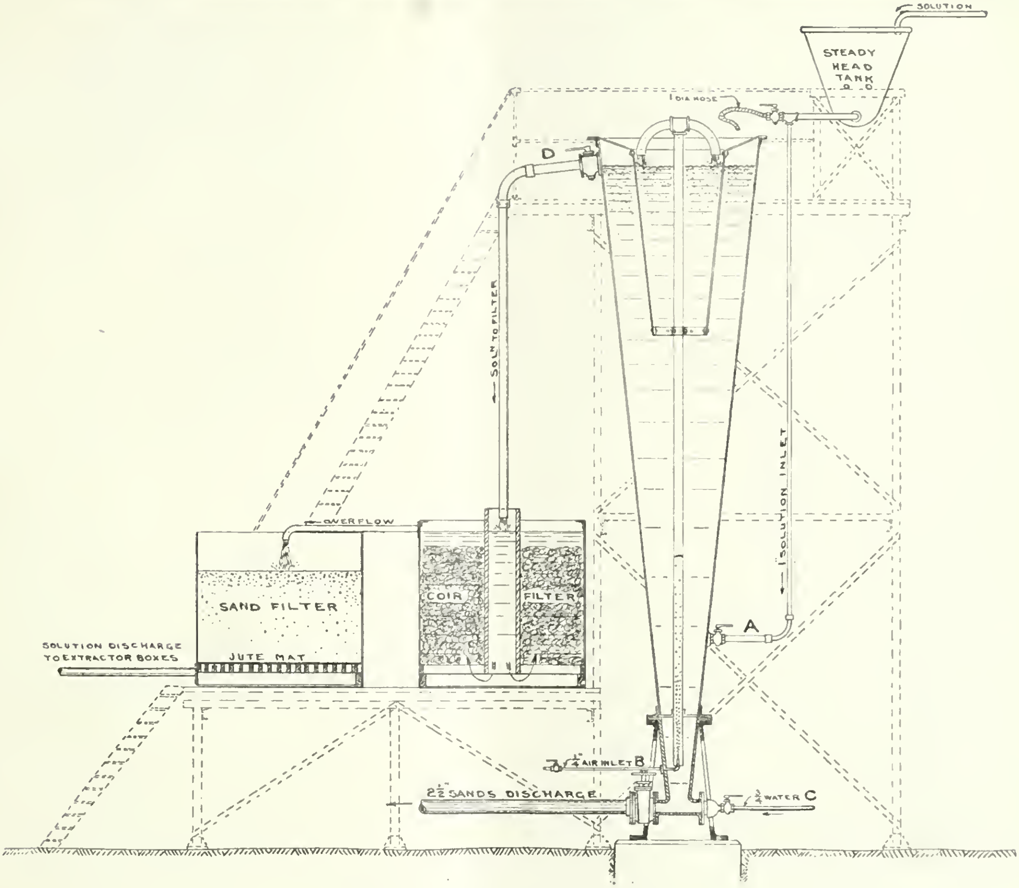
by Fraser Alexander (Member of Council).

I have been asked by several members of this Society to explain the method of treatment and plant adopted on the Crown Mines and worked by me with considerable success since October 1908, for the quick treatment with cyanide of our battery by-product commonly called "Black Sand." This subject has already occupied the attention of members of this Society, see February *Journal* of 1906, "A Quick Treatment by Cyanide of Black Sands," by Mr. D. V. Burnett.

After working the plant described by Mr. Burnett for a considerable period, I decided that the sampling of residues could not be relied on, and I often found amalgam accumulated immediately below the stirring gear in sufficiently large quantities to upset all calculations of actual extraction, and this was my chief objection to the paddle stirring process. Seeing a plan of Mr. Andrew F. Crosse's design for the treatment of concentrates, I decided to try the experiment on a working scale, and used for my purpose an old conical shaped water purifier 18 ft. deep by 4 ft. 3 in. at the top, tapering to 2½ in., to which I fitted air, cyanide solution, and water connections, as shown in the accompanying sectional diagram. This apparatus, which is simple in construction and costs practically nothing for maintenance, is extremely reliable and efficient for the treatment of refractory ores with cyanide.

Since installing this plant I have treated 61,340 lb. of black sands, having an average value of 23 0 oz. per ton, from which 683 oz. fine gold = 96·88 and about 150 oz. of mercury were recovered. In many instances I checked the assay values of charges by sampling the solutions, from which I obtained most satisfactory results in measured quantities.

Appended is a fair specimen treatment sheet : Dry weight of charge = 500 lb. charged into the cone with ordinary working cyanide solution to



Black Sands Treatment Plant.

which is added 15 lb. of cyanide and 10 lb. lime. Charged 2 p.m., circulated to 8 p.m., started washing 9 p.m., rate of wash = 75 tons per hour, 3 a.m., KCy = ·13, CaO % = ·015. Average value solution = 23·35 dwt. After further washing with weak solution (= ·05 KCy) for six hours, the average value was reduced to 4·3 dwt. 11 a.m., solution KCy = ·01, CaO = ·C08, Au = ·30 dwt. 12 noon to 1 p.m., water wash, completing the charge within 24 hours.

Original value of charge = 25·0 oz. per ton.
Value of residue = ·51 oz. per ton.
Theoretical extraction = 97·96 %

The process is practically continuous and requires very little attention.

I have no doubt this method of treating refractory ores with cyanide is capable of considerable extension, as a quick method of extracting gold and silver from their ores.

The partial analysis of the sample of treated black sands from the Crown Mines is as follows:—

	Per cent.
Silica (SiO_2) … …	51·94
Iron, calculated as Fe_2O_3	32·62
Sulphur (S) … …	16·98

Assuming the sulphur to be contained as iron pyrites having a formula of FeS_2, calculation gives the following data:—

	Per cent.
Silica (SiO_2) … …	51·94
Iron disulphide (FeS_2) …	31·84
Ferric oxide (Fe_2O_3) …	11·39

A qualitative investigation showed the presence of a very small quantity of arsenic, and traces of copper, titanium, cobalt and nickel. A small amount of lime was also noted, this being a subsequent addition for treatment purposes. A negligible quantity of magnetic iron is present.

I have to thank Mr. Croghan for the foregoing analysis, and Mr. F. W. Leeds for his courtesy in drawing the illustrative diagram.

Mr. E. H. Johnson (*Past-President*): I should like to move a hearty vote of thanks to our old confrere, Mr. Crosse, and also to Mr. Fraser

Alexander, for their interesting account of Mr. Crosse's new process. I have seen the plant in operation at the Robinson mine. The system of agitation is perfect in the central cone and gives a very clean overflow from the outer cone, but I think the point in question is the amount of solution which will be required to complete the washing. In Mr. Alexander's case I do not think it is a matter of importance owing to the high value of the original slimes.

Mr. W. A. Caldecott (*Past President*): In seconding the vote of thanks to the author, I may say that the use of the air lift for dissolving gold in slime pulp was not always as well-known a device as is now the case. Some years ago, in consequence of the wide claims of a patent application from abroad, Mr. Crosse's earlier experimental use of the air lift for slime treatment became more generally known.

The author has referred us to the filter press as being the main competitor of the decantation process of slime treatment. I think that we are generally agreed that ore slime filter pressing now belongs to the "has been" class of processes, but since we are fortunate enough to have Mr. Bosqui and Mr. Patterson from the United States with us, I trust that these gentlemen will before long give us their views upon the application of vacuum filtration to current low-grade banket or slime.

Mr. Andrew F. Crosse (*Past-President*): I should like to thank Mr. Johnson and Mr. Caldecott for the kind remarks they have made, and at the same time to mention that if any of the members of this Society would like to see a model plant at work, in which I can treat 80 lb. of slime, I shall be glad to show it them at Mr. Croghan's laboratory, opposite the Corner House, in Simmonds Street.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(*Read at May Meeting, 1909.*)

By Jas. Gray, F.I.C. (Member).

REPLY TO DISCUSSION.

Mr. Jas. Gray (*Member*): Although the discussion on my paper has been very full, I regret that the lime sellers have not come forward to give us the benefit of their views on the matter.

There is not much to reply to in regard to Dr. Moir's contribution, and I have only to supply the further figures he desires. I find at this season of the year that the changed atmospheric conditions tend to give different results to those obtained at the beginning of the year when the previous experiments were carried out, and in order to obtain corroborative results the time of exposure had to be reduced.

The details of the experiments are as follows:

30 Hours' Exposure —

Original weight of lime ...			11·7664 gm.
Increase in weight			12·94%
Available lime in sample before			100·0%
Available lime in sample after ...			86·52%
Total CaO present, after ...			88·4%
Loss on ignition			11·84%
CaO as $CaCO_3$,	0·268 gm.	0·4786 gm.,	3·60%
CO_2	0·2106 gm.		
CaO as $Ca(OH)_2$	4·0842 gm.	5·397 gm.	40·61%
H_2O	1·3128 gm.		
Unchanged CaO		7·4141 gm.	55·78%

Ratio CO_2 to H_2O 1 to 6·25.

17 Days' Exposure—

Original weight of lime ...			12·762 gm.
Increase in weight			40·34%
Available lime in sample, before...			100·0%
Available lime in sample, after ...			51·52%
Loss on ignition			28·83%
Total CaO present, after ...			71·56%
CaO as $CaCO_3$,	3·535 gm.	6·312 gm.	35·24%
CO_2	2·777 gm.		
CaO as $Ca(OH)_2$	7·379 gm.	9·751 gm.	51·44%
H_2O	2·372 gm.		
Unchanged CaO	1·848 gm.		10·31%

Ratio CO_2 to H_2O 1 to ·85.

Mr. Johnston is entirely in the wrong regarding his failure to come forward with his contribution sooner owing to want of opportunity, as the paper was before the June meeting for discussion at an early hour without result.

To turn now to Mr. Johnston's contribution. His experiments and mine show that when lime is exposed in an open vessel to moist atmospheric influences, the absorption of carbon dioxide is, after 3—4 days, greater than the absorption of water, although I now consider that this depends to a certain extent on the season of the year, even under the artificial conditions already described. It must, however, be admitted that an exposure such as described is much more severe than that to which the lime is subjected in transit from the works to the mine when the lime is to a certain extent protected by the bags. Mr. Johnston supplies details of two experiments which to him tend to disprove my conclusion that the absorption of carbon dioxide is slow compared with the absorption of water. The first experiment, owing to lack of sufficient data, cannot be examined, and I would only remark that something would appear to be wrong in the figures for the first day, which give a ratio of carbon dioxide to water of 1 to 3, but the second experiment carried out in a sack has all the necessary data, and from it I conclude that an exposure of 2–3 days in an open vessel is comparable with an exposure of about 14 days in a sack.

Mr. Johnston placed 100 oz. of freshly burnt lime containing 67·58% available lime in a sack and exposed it to moist atmospheric influences for 14 days, and at the end of that period found that it had increased in weight by 15 oz., and the available lime had decreased in value to 57·21%, *i.e.*,

Weight of lime originally present ... 67·58 oz.
Weight of lime after $\frac{57{\cdot}21 \times 115}{100}$ ... 65·80 oz.
therefore lime converted into $CaCO_3$ 1·78 oz.
therefore CO_2 absorbed by lime, 1·40 oz. 1·21%
and H_2O absorbed by lime, 13·60 oz.

a ratio of CO_2 to H_2O of 1 to 9·7, or approximately 10 to 1 against Mr. Johnston.

An inconsistency requiring elucidation is the quantity of carbon dioxide found present after the experiment was completed. As will be seen, 1·21% of CO_2 was absorbed, and Mr. Johnston found the outside of the sack to contain 14·2 $CO_2 = 33{\cdot}27\%$ $CaCO_3$ and the inside 3·76% $CO_2 = 8{\cdot}54\%$ $CaCO_3$, but the lime originally contained 32·42% impurities which must have been essentially $CaCO_3$, the lime having been freshly burnt, thus expelling the combined water which may have been originally present. It would be interesting and, no doubt, instructive to learn how Mr. Johnston reconciles these figures.

As this paper originated from observations of the fluctuations in value of the available lime contents in lime supplies during the different seasons of the year, I cannot take Mr. White or his friend in the lime trade seriously. Might I suggest that Mr. White should endeavour to make some real advance in our knowledge instead of worrying about "effete superstitions"?

Mr. Dowling's contribution, if correct, throws an interesting sidelight on the commercial instinct of the lime seller.

Messrs. Graham and Lindsay's remarks have very little bearing on my paper, as they deal with methods of testing the relative values of lime supplies. I would suggest that their ideas be carried to a logical conclusion by arranging that the mines pay for dynamite on the rock broken per month, or the cyanider on the tons of solution pumped, together with the number of glands packed

It is now generally admitted that the system of paying for lime on a guaranteed value of 75% available lime, in the supplies as received at the mine, is unjust to the seller, and it is satisfactory to know that arrangements have been or are being made to remedy this injustice.

In conclusion, I must express thanks to Mr. Alex. Heymann for facilities afforded to carry out the necessary experimental work in his laboratories.

THE EFFECT ON AMALGAMATION, OF DIFFERENT INTERVALS OF TIME BETWEEN THE DRESSINGS OF PLATES.

(*Read at June Meeting, 1909.*)

By G. O. SMART, A.Inst.M.M. (Member of Council).

REPLY TO DISCUSSION.

Mr. G. O. Smart (*Member of Council*): In accordance with the promise given at the reading of this paper, I now submit the results of a similar experiment carried out on tube mill amalgamation, with the view of determining the most suitable times for dressing those plates consistent with economy and efficiency.

A series of tests was undertaken, extending over a period of 12 days, during the latter part of June and the beginning of July, and the figures given below are the averages of the extractions obtained.

The individual results were very irregular, and in order to obtain concordant data on which to base a conclusion, some of the more abnormal figures were discarded before the averages were taken out. In explanation of this irregularity, it may be mentioned that amalgam is invariably present in the pulp discharged from tube mills, due to the mercury escaping from the mill plates forming amalgam with free gold released from the sand by regrinding in the tubes. When the effect on an assay of the presence of amalgam in the sample is considered, the difficulty in obtaining uniformly reliable assay heads on this material will be apparent. However, the average of the tests may be taken as sufficiently accurate to give a fair indication of the extraction at the times stated.

Hours after Dressing.	Per cent. Extraction.	Mesh.	Grading Entering Tubes.	Leaving Tubes.
15 mins	26·05			
2 hours	29·27			
4 ,,	27·04	+ ·01	59 97	16·80
6 ,,	25·01	− ·01 + ·006	23·25	24·88
8 ,,	23·87	− ·006	16 78	58·32
10 ,,	18·44			
12 ,,	19·17			

According to this table 8 hours appears to be the longest time that can safely be allowed between dressings, and in consequence of this test I adopted 8 hour intervals on the tube mill plates for some time, without any apparent falling off in the extraction.

Six hour intervals between dressings have again been adopted, as one-third of the mill is now running without amalgamation except on the tube mill plates, and it seemed inadvisable to apply the additional mercury required (to amalgamate the gold that formerly was caught in the mill) to the shaking table plates while a thick pulp was passing over them, about 85 tons of solids per table per day and more especially when a large portion of the surface was covered with black sand.

With regard to the reasons given by Mr. Bevington in explanation of the highest extraction being obtained some hours after dressing I quite agree, and although from the table given here this improvement in extraction is not so apparent on the tube mill plates; the thicker pulp and side-shake may have some bearing on the question, and in some measure account for the seeming disagreement.

In conclusion, I wish to thank Mr. Dowling for giving the results of a similar experiment at the Knights Deep, which serves to bear out those obtained by me, and also Messrs. Bevington, Alexander, White, Laschinger and Johnston for their contributions which, however, do not call for any further comment.

I may add that previous to adopting 12 hour intervals I had dressed mill plates at 8 hours for over ten years, and always obtained satisfactory amalgamation.

Mr. J. E. Thomas (*Member of Council*) read the following contribution :

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

(*Reprinted in this Journal, September, 1909, from the Bulletin of the Institution of Mining and Metallurgy, August 12, 1909.*)

By W. A. Caldecott (Member).

DISCUSSION.

Mr. W. G. Urquhart (*Associate*): The prevention of the irregular wear of dies is as yet an unsolved problem. I consider it is possible to prevent the uneven or irregular wear of dies by the proper application of a "Diagonal Die Water Service." Let us go back and consider the results obtained in most mills which are not equipped with a diagonal service, and where the water is fed from the top of the boxes, back or front. The wear of the dies is in accordance with the order of drop in use, providing, of course, that the weight and height of drop of the stamps are the same.

Example I.—With order of drop 1 3 5 2 4 Nos. 1 and 2 dies wear least and hollow most, Nos. 4 and 5 wear most and hollow least, because Nos. 1 and 2 have more ore always on them than 4 and 5.

Example II.—With order of drop 1 5 2 4 3 Nos. 4 and 3 wear least and hollow most, Nos. 1 and 5 wear most and hollow least for the same reasons as in Example I.

Supposing that with the same order of drop we use the diagonal service, all jets being the same size, the results as far as wear goes are practically the same. The difference being that the dies are less inclined to hollow, because the water playing on the dies washes off the crushed particles and leaves a clean amount of ore on clean dies ready for the next blow of the stamp. In my opinion the cause of "caving" is the fact that after the stamp has struck its blow, the crushed particles remaining on the dies are gradually ground in, as they are being added to with each drop of the stamp and is not due to steel, softness of the die, or swirl of the pulp. If swirl took place the contents of the cavity of a die would not remain an almost solid mass.

Grading.

+ 36	...	19·5	per cent.
+ 100	...	6·2	,,
+ 400	...	12·8	,,
+ 1,600	...	13·3	,,
+ 60	...	5·0	,,
+ 90	...	9·7	,,
− 90	...	33·5	,,

The grading of the contents of the cavities of a die will help to support my opinion, and the inspection of this piece will show the absence of steel.

To remedy this banking and hollowing of dies it is advisable to use a sufficient quantity of water distributed on each die, in proportion to the way in which the individual dies are cleared by their respective stamps.

For instance, with 1 3 5 2 4 the jets regulated thus will give best results, Nos. 1, 2 and 3 full open, No. 4 half open, and 5 three-quarters open. It will be observed that where the ore banks most, the larger quantity of water is used and *vice versa*. By using too great a quantity of water we get dies slightly rounded on the surface—exactly the reverse of the case where water has been used to sparingly. If it is desired to take the "cave" out of a die it is advisable to feed fine and use plenty of water.

I would like to make one suggestion in reference to the diagonal service in use at present. Instead of using only 5 jets, 7 I think would give better results, distributed as follows: one on each die as at present, the two additional jets to be put in, one at each corner of the box in order to wash round the sides, where most of the banking takes place, especially at the 1 2 end when the order of drop 1 3 5 2 4 is used.

I would like to make it understood in conclusion that the foregoing remarks have been made without any consideration for amalgamation.

Since writing the above I have read the President's remarks at last meeting, urging mill men with heavier stamp experience to bring forward their views. I hope I shall not be the only amalgamator to assist in solving one of the many problems we are faced with daily.

The Chairman: Mr. Urquhart is one of our Associates, and I should like to say that in bringing forward this contribution to the discussion he has set a very good example to the other Associate members of the Society.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution:

THE BARBERTON GOLDFIELD.

(Read at October Meeting, 1909.)

By A. Richardson, M.I.M.M. (Member of Council).

DISCUSSION.

Mr. Tudor G. Trevor (*Member*): With regard to Mr. Richardson's paper on the Barberton Goldfield, published in the October *Journal*, he states under the heading "History":—

"Although the history of Barberton district may be said to have begun with the diggers of 1882, there were many evidences discovered by them, such as the remains of properly graded roads, mining works, and old smelting pits containing small quantities of smelted gold, suggesting an Arabic civilization of pre-Mahommedan date, and also old workings probably of Kafir origin of a few generations back. It is highly probable that the operations of some earlier race, of which we have only the faintest records to-day, may in some measure account for the comparative poverty of modern South Africa in alluvial auriferous deposits."

I have for many years taken a great interest in the question of old workings in this country and Rhodesia, and have made it my business to visit all that I could hear of. Since the war my official work has given me a very intimate knowledge of the De Kaap Fields, and though I have made constant enquiries as to ancient workings, I have, with two exceptions, been unable to find any in this district. The exceptions are the very large native iron workings near Malelane, and on the Swazieland border at Indumene near the Havelock Concession.

In the Lydenburg, North Middelburg, Zoutpansberg and Waterberg districts old workings that can undoubtedly be described as *ancient*, are so numerous that their absence in the De Kaap has always struck me as peculiar, and I should be extremely glad to hear of any places in that district where they can be found.

I have frequently heard people attribute the natural caves in the Black Reef sandstones at the Kantoor to "Phœnician Workings," but these are obviously natural, and similar caves occur all along the edge of the berg from Swazieland to Zoutpansberg. The heaps of stones which the former Basuto occupants of the De Kaap and Steynsdorp valleys collected in their fields, and the rough terraces which they similarly made on their hillside agricultural lands, and which remain to this day, I have likewise heard attributed to ancient alluvial work and roads; but as these occur without any regard to whether the ground is mineralised or not, and as they can be seen in the course of construction to-day near every native Basuto kraal, such a theory for their origin is obviously absurd.

This question of the ancient occupation of Eastern South Africa by a mining race is so interesting, that it is extremely important that the real evidence of its existence should not be smothered, just as attention is being drawn to it, in a mass of false evidence and wild supposition.

In Rhodesia, this habit of attributing everything to a mysterious and wonderful ancient race, has led to the undoubted evidence of that race being so obscured, that first class investigators, despairing of ever being able to sort out the true evidence from the false, have denied, *in toto*, the existence of the race, and I therefore think that before the statement I refer to receives the *imprimatur* of the Society, the evidence for it should be put on record.

THE CONTINUOUS COLLECTION OF SAND FOR CYANIDING.

(Read at October Meeting, 1909.)

By W. A. Caldecott, M.Inst.M.M. (Past-President).

DISCUSSION.

The Chairman: In comparing the experimental plant originally erected on the Knights Deep with the one now in regular use on the Simmer Deep, two important improvements have been introduced. The first is washing the underflow from the primary cones before it enters the table cones. This enables the table to do much more work, as in the early experiments we found that the presence of slime hindered the filtration and reduced the capacity of the table.

The other important improvement is the use of a diaphragm in these large cones.* I consider that this device is of the greatest possible value in enabling the sand filter tables to run smoothly and avoid sudden rushes of thin fluid underflow. Diaphragms have been introduced in the classifying and dewatering cones of the Simmer East tube mills. Break-aways are of very rare occurrence and the cones run for days with an underflow moisture of 30%, with little attention other than distributing the supply of pulp. It is possible to have so large an aperture (from 2 in. to 2¼ in. on Simmer East) for the underflow that chokes rarely occur.

Mr. C. O. Schmitt (*Member*) : The Society is to be congratulated upon having had this very excellent paper by Mr. Caldecott, who has given us a very clear statement of what is undoubtedly a great step forward in the metallurgy of gold, and although we do not notice any extravagant claims, as pointed out by Mr. White, there is no doubt that those capable of judging for themselves will agree that a real advance has at last been made in the treatment of sand.

The satisfactory collecting of the sand contained in the final pulp leaving the battery, preparatory to treatment is, as all cyanide men know, by no means a simple proposition. It is obviously necessary, in order to facilitate treatment, to collect the sand free from all slime, and, as we know, this is not by any means always done, the slime contents of collected sand being some times as high as 10%.

Slime pulp should be treated in the plant provided for the purpose, and since the treatment of slime is, as a rule, less expensive than the treatment of sand, any slime retained in the collected sand, not only costs more to treat than it otherwise would, but it also displaces sand which could be handled in its place, quite apart from the fact that the presence of slime in the sand collecting vats interferes with the treatment. This is obvious to all, but the matter is so important that it will bear quite a lot of repetition.

Bearing in mind the remarks just made, and seeing the result obtained by means of the author's continuous sand collecting plant, one can only admire the author's modesty in not laying more stress upon the excellent performance of the plant. It is only right to say that the plant at the Simmer and Jack, using a 15 ft. table, and of which the result of six months' work is given, is only of an experimental nature and subject to the usual infant ailments, and the writer, for one, has no doubt that the result from running on a large scale at the Simmer Deep will show improvements, when the author furnishes additional information, as it is to be hoped, he will do in his reply.

Many will no doubt object to the introduction of further machinery into the already complicated reduction plant, and the writer had the same feeling when the introduction of the new method was under discussion. Fortunately, the author was able to dispose of any such objections by pointing out in his paper that the continuous running of the filter table presented no difficulties; so much so that the arrangements made for temporarily reverting to the old style of working, in case of temporary failure, have not so far been needed.

The author claims that the new plant does the following :—

(1) The sand collected is cleaner and contains less slime than is the case in the ordinary plant.

(2) The capital cost of the sand plant has been reduced by nearly 40%.

(3) The space occupied by a sand plant has been reduced.

(4) The time of treatment is reduced to about two-thirds of that now occupied.

With regard to item 3, while its importance may not be obvious to everyone, all those whose business it is to lay out plants will no doubt appreciate it.

The question of capital expenditure will appeal to most, and since the saving of 40% on the sand plant amounts to about 10% on the whole reduction plant the result accomplished is worthy of considerable praise. The saving on capital account may be still further increased by providing only one set of treatment vats, and while this may not be acceptable for very high grade propositions there will be many instances where it could be justified.

Considering the question of cleaner sand, the importance of which has already been referred to, the author stated that the percentage of slime was raised from 32% to 40%, a difference of 8% which Mr. White chose to call fine sand sent to the slimes plant. The author deliberately stated that less than 1% of slime was present in the collected sand and since there is up to 10% of slime in the product usually termed sand there is apparently no need for Mr. White's remark. In order to clear up this point the writer trusts that the author will furnish a complete grading analysis of the average product at the Simmer Deep.

The tendency all over the world is towards all sliming, and in view of the ease with which the average slime is treated, this is, in many cases, quite justified. In view of the result obtained it is obvious that the author has done quite a good deal towards the desired end, and (in the writer's opinion) more than those who propose to spend a shilling in crushing and grinding, to save sixpence in the cost of the succeeding treatment. We are not by any means at a stage where the sand plant can be discarded, and in this connection the writer remembers a conversation he had, a few months ago, in London with one of our Past-Presidents, Mr. Charles Butters, who, in explaining plans and photos of a plant designed by him and erected in Mexico, proudly pointed out the success of all-sliming. Upon closer investigation, however, it was found that the particular plant did contain a sand plant, and the explanation offered was that this was added when, upon work having started, it was found that the product from the crushing and grinding plant did contain what is usually termed sand and which did not yield sufficient of its precious metal contents to the persuasion of the slime treatment alone.*

---

* See this *Journal*, March, 1909, p. 312.

The paper is so well written, and all statements were apparently so carefully weighed before being offered to the Society that it is hard to find fault, but in the writer's opinion the author did not lay sufficient stress upon that portion of the work which is done by the cone classifiers, supplemented by the ingenious device he calls diaphragm. It appears to the writer that the insignificant looking diaphragm has as much to do with the success of the continuous sand collecting plant as the filter table, although there is a considerable difference in the cost of the two items. A description of the diaphragm as applied to the cone classifier used in tube milling appeared some time ago,† but a little more information regarding its application in the continuous sand collecting plant would, no doubt, be welcome.

The writer wishes to apologise for these somewhat disjointed remarks, and trusts that others more fit will come forward to discuss Mr. Caldecott's very interesting paper in the manner it deserves.

The meeting then closed.

Contributions and Correspondence.

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

In the July issue of this *Journal*, p. 19, Dr. Moir criticises some of the chemical theories advanced by the authors of this paper as explanations of the effects upon sulpho-telluride gold ores of solutions containing cyanides, persulphates, and iodine. While some of these explanations certainly require further proof, at least one of the criticisms appears to be unfounded.

The authors (May *Journal*, p. 400) remark: "sodium thiosulphate, $Na_2S_2O_3$, being formed by the action of the persulphate upon the pyrite, is probably further oxidised to $Na_2S_4O_6$ (tetrathionate)." This appears to be the statement to which Dr. Moir specifically objects, as not being borne out by the facts, when he writes that "the only conceivable substances producible from pyrites are SO_2 and H_2SO_4." As to the precise effect of oxidation of pyrite by means of persulphate I have no experimental data, but as to the oxidation of pyrite in general Dr. Moir is evidently in error. In the slow oxidation of pyrite in natural deposits by atmospheric agencies we occasionally find native sulphur as one of the products, in addition to the various sulphates of iron enumerated by writers on economic geology, and when moist pyritic concentrates are exposed to air *en masse* hydrogen sulphide is often evident.* Also when pyritic ore is exposed to water containing lime (and presumably any other alkaline hydroxide would act similarly) thiosulphate may be detected in the resulting solution. I am not prepared to say that an absolutely pure cubic pyrite would yield thiosulphate when exposed to lime water, but this result is certainly met when even moderately pyritic ores from some of the mines in the western United States are mixed with lime and water with partial access of air. This effect is particularly notable with certain ores containing both pyrite and pyrrhotite.

Several years ago† I included thiosulphate among the compounds that might be found in working cyanide solutions, having detected it in 1896. Since that time I have found it in solutions from several mines. While a thiosulphate has not been actually isolated from such solutions, and occurs only in small proportions, the evidence of its existence seems unmistakeable, and its source is undoubtedly either pyrite or pyrrhotite.

Tests show: (1) the presence of a compound of sulphur which is neither a sulphide, sulphate, or thiocyanate. After long boiling with addition of bromine the S finally found by precipitation as $BaSO_4$ is slightly greater than the sum of the quantities in the sulphate and thiocyanate found in the original solution, no sulphide being present.

(2) On acidification with HCl or H_2SO_4 there is a liberation of SO_2 and a slow precipitation of sulphur.

(3) After neutralisation or acidification it absorbs iodine.

(4) After acidification with H_2SO_4 it reduces permanganate—sometimes rather slowly.

It seems very probable that this thiosulphate is formed by the oxidation of alkaline sulphides resulting from the action of solutions containing free alkali on iron sulphide, for in rare cases I have detected sulphide in such solutions immediately after flowing from sulphuretted charges, but failed to find it in the same solutions a short time subsequently. In absence of cyanide the thiosulphate is probably oxidised later to tetrathionate and sulphate, but in presence of alkaline cyanide, by far the larger proportion of the sulphur taken up, finally forms thiocyanate, which may accumulate in very considerable quantity when pyrite-pyrrhotite ores are treated. Thiocyanate is the principal cause of the reducing effect observed when cyanide solutions are titrated with permanganate after acidification. This test is part of the daily routine at some works. At the Homestake plants, South Dakota, the "reducing power" is daily determined for each class of solution, and is reported numerically as "c.cs. of $N/10$ permanganate required to oxidise 100 c.cs. of the solution tested. In practice 5 to 50 c.cs. are taken for the test, and the reducing figure found may vary from 5 to 500 or even higher—that is to say, 100 c.cs. of solution, after acidifying with sulphuric acid, will require 5 to 500 c.cs. of $N/10$ permanganate to produce a permanent pink tinge. The value 500, for instance, corresponds to about 0·81% of KCNS in solution, or 0·267% of S.*

---

* See also "All-Sliming," by E. M. Hamilton, *Mining and Scientific Press*, August 21, 1909.

† *Journal of S.A.A.Eng.*, vol. xiv., No. 5.

* See this *Journal*, vol. ii., 1897, pp. 98 and 122.

† *Journal American Chemical Society*, May, 1897, p. 401.

The following may be cited to show the extent to which sulphur passes into solution during contact of ore with lime water. A charge of about 600 tons of mill tailings free from slime with about 800 tons of water was run by a revolving distributor into a vat previously almost full of clear water, 2,400 lb. of lime being uniformly mixed with the pulp in the launder leading to the distributor. Charging occupied 8 hours. The tailings contained slightly over 4% of sulphur as pyrite and pyrrhotite with a small proportion of arsenical pyrites, no other sulphide being present; the lime carried at least 80% of available CaO, the water contained lime and magnesia, but only traces of sulphate. As soon as charging was complete the surface water was siphoned off the sand, and a bottom valve was opened to allow the mass to drain for 16 hours, during which time some 50 to 60 tons flowed out, samples being taken hourly. The first two samples were disregarded as they contained traces of KCN and KCNS derived from a little solution remaining under the filter bottom; later samples showed no trace of either. The average effluent contained ·05 to ·06% CaO by precipitation as oxalate; tests of the alkalinity by standard acid showed uniformly a very slightly higher result, owing to the presence of traces of soda in lime and water. The average absorption of iodine (using starch indicator) was 2·8 c.cs. of $N/10$ I per 100 c.cs. On acidification a distinct odour of SO_2 was noted. No indication of sulphide was found. Supposing that all the iodine was consumed by CaS_2O_3 (1 at. I = 2 at. S) 100 c.cs. of this solution contain $2{\cdot}8 \times {\cdot}0064 = 0{\cdot}018$ gm. S, or 0·018% S. Then the entire 55 tons or thereabout of effluent contain approximately ·55 × ·018 or ·01 ton of sulphur as thiosulphate. But the 600 tons of tailings from which it was derived contained over 4% S or, say, 25 tons. Hence only about 0·04% or one part in 2,500 of the total sulphur in the charge is thus removed in the preliminary draining, the time of contact having averaged less than 12 hours. Some remained dissolved in the moisture adhering to the drained tailings; this moisture carried a slightly higher percentage of sulphur and CaO; this was partly drained off later by displacement, and partly mixed with the first ongoing cyanide solution; this mixture was removed as "low solution," treated with zinc, and then passed out of the system. Such sulphur as dissolves later in the treatment remains in large part in the stock solution, where it gradually accumulates as thiocyanate and sulphate.

The extent of this accumulation of sulphur in the circulating system is shown by a calculation made on the stock solution at a time when the "reduction test" indicated 500 c.cs. of $N/10$ permanganate consumed per 100 c.cs. of solution. As before noted, this corresponds to 0·267% of S as thiocyanate, while the S in the form of sulphate was found to be about 0·015%, or about one-eighteenth of the former. At these percentages the entire stock of solution (nearly 4,000 tons at this plant) contained over 10 tons of sulphur as thiocyanate and half a ton as sulphate.

The number of tons of tailings required to yield these ten or more tons is not directly determinable, as the exact proportion of the stock solution daily discarded is not easily ascertained. Some idea of the rate of solution may be got by going back to the previous instance. From 600 tons of tailings, after about 12 hours' contact, some 55 tons of water removed about 0·01 ton of sulphur. The total solution, etc., percolating during the treatment of this charge was over ten times this volume, at the same rate the entire charge would lose 0·1 ton of S. Approximately 1,000 charges are treated in one year at this plant, or about 100 tons of sulphur are dissolved in that period if this class of ore is maintained.

The figures given are obviously round numbers, but approximate fairly to the actual values, and at any rate serve to show that the products other than sulphates, obtained in leaching pyritic materials, while small in relation to the total sulphur of the ore, are by no means negligible in quantity. While the exact mechanism of the reactions is not definitely known, it is plain that even the thiocyanate is an oxidation product.*

---

* The permanganate consumed by cuprous compounds is very small here, and the ferrocyanide exists only in infinitesimal quantities.

W. J. SHARWOOD.

Lead, S. Dak., U.S.A., Sept., 1909.

Mr. Sharwood is quite correct about the separation of sulphur and hydrogen sulphide from pyrrhotite under certain conditions: but as the original authors did not mention the existence of pyrrhotite, my remarks were intended for true pyrite. However, I entirely disagree with Mr. Sharwood as to the evidence for the presence of thiosulphate in these solutions, and in any case NH_4SO_4 is so energetic that it would destroy both thiosulphate and tetrathionate. I cannot understand in the first place why ferrocyanide should be infinitesimal whilst sulphocyanide is high; for experiment shows that the first action of cyanide on FeS or pyrrhotite is to form ferrocyanide and monosulphide. (1) I am not satisfied that a determination of total S as $BaSO_4$ is accurate enough in this case to prove the existence of anything beyond sulphide, sulphate, and sulphocyanide. Mr. Sharwood should try to destroy zinc ferricyanide: he will find that it is almost impossible to remove it (when present) from the $BaSO_4$ precipitate. (2) I have never detected SO_2 from such a solution: I have seen a turbidity on acidifying, which is probably due to decomposition of a disulphide. (3) Absorption of iodine proves nothing: it might even be due to formic acid. (4) *Slow* reduction of permanganate is certainly not due to thiosulphate or SO_2—it is more like the behaviour of finely divided sulphur. (5) I do not believe that tetrathionate, once formed, could get oxidised to sulphate by the air. There are plenty of specific tests for the thiosulphate radicle, and I hope Mr. Sharwood will try them with his solutions, because here I can only work with artificial mixtures, as our working solutions do not contain anything but ferrocyanide and sulphocyanide.

JAMES MOIR.

Johannesburg, Nov., 1909.

STOPE MEASUREMENTS.

With regard to Mr. K. Tonnesen's remarks I have to say, (1) that now that I have seen his diagram, I agree that Mr. O. Tonnesen's calculation of area is substantially correct, but think I was justified in charging him with obscurity, seeing that he led me to suppose that his "circumscribed parallelogram" corresponded to the circumscribed triangle —which I now find it does not; (2) as for Mr. Coaton's calculation, I said expressly that the parabola is the same as the catenary near the bottom (*i.e.*, tape horizontal); hence it is no wonder that the error is less than 1 in 1,000 in the case considered. Mr. Coaton will find that even if he assumes the curve to be a circle, the result will be almost as close, and I daresay any curve he likes will give $A = \frac{2}{3}Sl$ like the parabola *provided the tape is horizontal.* Curiously enough, though Mr. Coaton's result is correct his formula for the area is wrong: it should be $2\left(xy - a^2 \sinh \frac{x}{a}\right)$ or alternatively $(ax - a^2)e^{x/a} + (ax + a^2)e^{-x/a}$ On the other hand, my own "higher mathematics" of July are correct, but my arithmetic is wrong. The sag (measured at right angles to tape) should have been $\frac{l^2 \cos a}{8a}$ (*i.e.*, half of what I said in July), and consequently the area should be $\frac{2}{3}Sl\left(1 + \frac{l \sin a}{a}\right)$ Mr O. Tonnesen's formula only holds when the tape is fairly level; the above holds up to 50° or 60°: when the tape is nearly vertical the area cannot be expressed in terms of the sag, as the curve is then the exponential one $y = \frac{c}{2}e^{x/c}$

JAMES MOIR.

Johannesburg,
November 19, 1909.

MINE VALUATION.

Under General Business last month, Mr. Wade suggested the calculation of ore developed in a block by using the horizontal area and the "vertical width" of the reef, in order, by some unexplained process, to take into account the influence of dip and width on working costs. This is a fantastic suggestion, which is most unlikely to find favour on any class of mining field. Even if Mr. Wade's proposal, which is virtually an attack on accepted methods, were followed, we should require accurate records of dip for other purposes and also, to get the best results, sampling would still be done at right angles to dip when possible. True stope widths would also continue to be an essential item. In any case, therefore, we get all the data required for calculation by the ordinary methods. Why, then, work on a practically useless figure such as vertical width! To take an example typical of the Rand;—a 5 ft. stope would, at 40° dip, have a 78 in. vertical width and at 60°, a 120 in. width. These figures, 78 in. and 120 in., have no value as factors in an ore reserve estimation. The ratio is quite useless for indicating relative costs. Under any circumstances, the final valuation must be based on profits, and it is in the estimation of working costs that allowance for the influence of dip and the host of other factors should logically be made. Mr. Wade himself admits that the dip only affects the position in some cases. The scheme's only apparent advantage is that it may save errors of measurement due to misjudgment of true perpendicular. This benefit is of small importance. Mr. Wade's scheme has certainly the merit of originality, though possibly the novelty of the idea is in part explained by its lack of practical utility.

---

* W. A. Caldecott, this *Journal*, March, 1908, p. 267.

RALPH STOKES.

Johannesburg, Nov., 1909.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE CHEMISTRY OF ZINC SMELTING.—"As the result of experimental investigations, Mr. V. Lepiarczyk (*Metallurgie*, July 8) finds that pure zinc sulphide can be roasted completely to zinc oxide, provided the conditions are such as enable the resulting zinc sulphate to undergo decomposition and allow free escape to the sulphur dioxide or trioxide formed. The imperfect elimination of sulphur in roasting blende is due—apart from incomplete decomposition of the zinc sulphate—to the presence of iron, either as oxide (which forms ferrous sulphide, and also prevents the roasting of a portion of the zinc sulphide, causing a loss of 6% of sulphur), or as pyrites, which, in spite of the presence of a large proportion of calcium carbonate, checks the formation of sulphate and gives a loss of 9½% of sulphur. A still more injurious effect is due to calcium carbonate, which is transformed into sulphate, and causes a loss of 12% of sulphur. In reduction distillation, a portion of the calcium sulphide formed by the reduction of calcium sulphate, reacts on the latter to form lime. The metallic sulphides are attacked by the carbon, a portion of the zinc sulphide being volatilised, and the ferrous sulphide still more strongly affected. In the muffle, with exclusion of air, two molecules of ferric oxide react on four molecules of zinc sulphide, the simultaneous presence of carbon leading to the formation of metallic iron and zinc, together with carbon monoxide and carbon disulphide, this substance being also formed during the reaction of carbon on zinc and iron sulphides. In the presence of zinc oxide, ferrous oxide and manganous oxide, no calcium sulphide results from the reaction of carbon on calcium sulphide, lime alone being produced. The practical conclusions drawn are to the effect that, whilst the iron almost invariably present in zinc blende has a slightly injurious effect by preventing the roasting of a small portion of the zinc sulphide, it is very useful in zinc smelting when present in the form of ferric oxide. The reasons for this are, firstly, that it reduces to the condition of oxide any of the zinc sulphide that has escaped roasting, and thus renders the same accessible to the processes of reduction and distillation; and, secondly, because it fixes the sulphur resulting from the reduction of calcium sulphate, and prevents the same from combining with zinc. Consequently, in order to increase the yield of zinc, it is advisable to add ferric oxide when charging the muffle in the case of roasted calcareous blende. On the other hand, the necessity for such addition could be obviated by eliminating the greater portion of the lime from the ore previous to roasting—if possible, concurrently with the washing of the ore. In such case the roasted blende would be as simple to treat as roasted calamine, the amount of ferric oxide in the ore being sufficient, in the absence of calcium sulphate, to convert any unroasted zinc sulphide into zinc oxide, and render it available for reduction to metal. No appreciable increase in the yield of zinc can be expected to result from the diminution of the percentage of sulphide sulphur, so long as no measures have been taken beforehand to prevent the formation of sulphate sulphur, the latter being transformed into the former in large quantities as soon as the reduction process begins, and combining with the metal and not with the calcium. Hence, more extensive roasting would merely increase the cost and lengthen the process, without any appreciable increase in the production of zinc."—*London Mining Journal*, Sept. 4, 1909, p. 326. (A. R.)

WHITE PRECIPITATE FOUND IN ZINC BOXES.—The white precipitates treated in this paper were received from the Kendall Gold Mining Co., of Kendall, Montana. The analysis of the ore is as follows:—

Loss on ignition	3·46%
SiO_2	75·94
Al_2O_3	10·98
Fe_2O_3	1·89
CaO	3·15
MgO	1·94
SO_3	·46
	98·33 (*sic*)
CO_2	1·85%
Na, Ag	traces
Au—	4·80 dwt. per ton

An outline of the treatment is as follows:—

1. Ore is crushed dry to ⅜ in. mesh and is placed in the tanks (dry) with 1 lb. of lime per ton of ore.
2. Tank is charged with strong solution (3 lb. of KCy per ton of solution) introduced under the charge. (Time 8 hours.)
3. Tank is leached with strong solution 144 hours.
4. First wash run on tank, strength 1·5 lb. KCy to ton of solution. (Time 8 hours.)
5. Second wash run on tank, strength ·5 lb. KCy to ton of solution. (Time 48 hours.)
6. Fresh water run on tank. (Time 20 hours.)
7. Tank drained 8 hours.
8. Tank sluiced 4 hours.

Total time of treatment, 10 days. The ore value is from 3 to 8 dwt.

The method of precipitation of the gold is by means of zinc shavings. The zinc boxes used for the strong solution are above the storage tanks and consist of eight double boxes. Each double box (4 ft. × 14 ft.) contains two rows of eight compartments each; seven of these are packed with zinc and the last used for settling. The size of compartments is 1½ × 1½ × 2 ft., and the bottom slope 1½ in. to the foot. Each compartment has capacity of 4½ cub. ft., making an aggregate zinc holding capacity of 520 cub. ft. The total weight of zinc in the above described boxes is 4,420 lb., or 8½ lb. per cub. ft. The amount of solution passing the strong boxes is from 800 tons to 1,000 tons per 24 hours, or from 1·5 tons to 1·9 tons per cub. ft. of zinc content.

The zinc consumption is about ·5 lb. per ton of ore treated, or ·09 lb. per ton of solution in circulation. The zinc shavings are cut from No. 9 sheet zinc, and are 1-1,400th of an inch in thickness. The strength of the strong solution is 3 lb. of cyanide per ton and protective alkalinity from 0 to ·2 lb. Very little lime is used owing to the oxidised condition of the ore and the absence of cyanicides. The boxes used for the precipitation of the weak solutions and the wash water are four in number. The amount of solution and wash passing the above boxes is from 500 tons to 600 tons per 24 hours. Precipitation in both strong and weak boxes can be called good; as the head samples to the strong boxes run from ·60 dwt. to 1·50 dwt. per ton, and to the weak boxes from ·10 dwt. to ·50 dwt. per ton, while the tail samples from both are less than ·04 dwt.; this result is considered high. The zinc shavings in the weak boxes become somewhat coated with white precipitates, the analysis of which will follow. The total consumption of cyanide is from ·5 lb. to ·6 lb. per ton of ore treated. Both strong and weak boxes are cleaned up twice a month. One feature of the acid treatment of the gold slime that has proved exceptionally successful is the use of steam to heat up the solution, near the end of the treatment, as the acid at this point becomes inactive, and the heating of the solution brings about a violent action. Hot washes are found much more effective than cold washes in dissolving out the soluble sulphates.

There are two kinds of white precipitates formed at the Kendall Mill. White precipitate No. 1 is always found in the zinc boxes to a greater or less extent, and is abundant when mill solutions become foul with zinc and other salts. It is found most in wash water boxes where strength of free cyanide is low and when gold value of head samples to boxes is low. It is light and flocculent and floats around in the boxes when the zinc is moved slightly. There is little or no protective alkalinity in the solution when it enters the zinc boxes.

Analysis of White Precipitate No. 1.

Water at 100 C. ...	5·06%
Loss on ignition.. ...	23·30
ZnO	37·46
Zn = 30·09	$Zn(OH)_2$ = 45·74
SiO_2	13·12
Al_2O_3	5·84
Fe_2O_3	1·55
CaO	3·62
MgO	5·12
SO_3	4·42
Au	·48
Ag	trace

White precipitate No. 2 was taken from bottom of acid treatment tank after the gold slime had been treated with H_2SO_4 and washed out. It is only slightly soluble in H_2SO_4 and undoubtedly comes from the white precipitate (No. 1) formed in the zinc boxes. It is light coloured and heavier than precipitate No. 1. It is not hard to wash from the gold slime, although some of it is washed out with the gold slime and goes to the melting furnace.

Analysis of White Precipitate No. 2.

Water at 100 C. ...	2·57%
Loss on ignition.. ...	14·10
ZnO	2·07
SiO_2	1·69
Al_2O_3	2·32
Fe_2O_3	·33
CaO	35·19
MgO	·17
SO_3	39·65
Au	2·39
Ag	trace

The writer made several tests on the solubility of this white precipitate in KCy solution and found that the precipitate was quite soluble. This was reported to the cyanide man at the Kendall mill, who increased the mill solution to 3 lb. KCy per ton of solution with 1 lb. protective alkalinity, and has just reported that he is now troubled with very little white precipitate.—R. F. COOLIDGE.—Thesis submitted to South Dakota State School of Mines. (H. A. W.)

THE FIXATION OF ATMOSPHERIC NITROGEN IN CYANIDE MANUFACTURE.—"Spasmodic attempts at the manufacture of cyanide, based upon reactions discovered over 60 years ago, for the fixation of atmospheric nitrogen as an alkaline—or alkaline earthy—cyanide have from time to time been made with more or less success, but I do not think that a description of a plant which was in use in the early nineties, but subsequently closed down owing to its unpayable nature, has hitherto appeared in print.

The works I refer to were in Scotland. A coal-gas works was obtained, suitable additions were made, the process being carried out as follows:—Charcoal, selected for its low ash-contents and even porosity, was soaked in a saturated solution of potassium carbonate. The charcoal was drained and dried, charged into gas retorts and dry nitrogen (from the air), deprived of oxygen as completely as possible, was passed over the heated charcoal (+ K_2CO_3) until the absorption of nitrogen slackened. The charges were then "drawn," out of contact with air, being cooled by the incoming dry nitrogen, and fresh charges introduced. On the average, about 25 per cent. of the potassium salt was converted into cyanide by the following reaction: $K_2CO_3 + 2C_2 + N_2 = 2KCN + 3CO$. At the same time a certain amount of metallic potassium was formed by the reduction of the carbonate with charcoal: $K_2CO_3 + C_2 = K_2 + 3CO$. The carbon monoxide under certain undefined conditions formed a carbonyl (potassium carbonyl) $K_6(CO)_6$, which has explosive properties.

The crude cyanide so produced was subjected to systematic lixiviation and the saturated solution of $K_2CO_3 + KCN$ decomposed by the CO_2 present in gases from a slow combustion furnace fed with gas-coke. The operation consisted in passing the gases containing nitrogen and carbon dioxide (with some impurities), through a tower containing coke over which the stream of cyanide solution trickled. The decomposition was as follows: $2KCN + 2H_2O + 2CO_2 = 2HCN + 2KHCO_3$.

The exhausted liquor, after suitable concentration, was re-used for saturating the leached and dried charcoal, with suitable additions of fresh material. The saturated charcoal was then dried as before and went through the above-mentioned cycle of operations.

The hydrocyanic acid gas swept out of the "scrubber" by the nitrogen was absorbed by a cold-water spray in another tower, also filled with coke, the gas being dissolved in the water as hydrocyanic acid. This liquid was collected in stills (actually large steam boilers) and heated with steam coils. The HCN, with some water vapour, passed through a coil cooled to 25 to 28 C., which allowed the water to condense and return to the still, while the HCN passed forward. This gas was then condensed in another coil, kept as cold as possible, the liquefied HCN dropped into caustic soda, the compound $NaCN,H_2O$ being formed. The crystals were "fished" out and "centrified." The trace of caustic soda adhering to the cyanide became converted into sodium carbonate, and the crystals were sold as 80 (equivalent of KCy) strength. By long-continued drying the combined water could be removed to a considerable extent, the "cyanide" content being similarly increased.

When the works were started, it was expected that the total cost of manufacturing cyanide (based on 98/100 KCy) would be about 3d. per lb., but the actual cost was about 1s. per lb. Among the causes which contributed to this unexpected increased cost were the following:—

(1) Loss of carbonate of potassium by formation of silicate of potassium from ashes in wood and impurities in coke.

(2) Loss of carbonate of potassium by formation of sulphide, sulphate and sulphite of potassium from sulphur compounds in furnace gases and from coke.

(3) Waste of charcoal from oxidation due to porosity of retorts.

(4) Excessive fuel consumption (over that calculated as necessary) for power and heating.

(5) Excessive wear and tear through damages to plant from explosions, etc.

I have in my possession some of the sodium cyanide made by this method, but it has lost most of its water of crystallisation. Modern methods in vogue in Europe and America for the manufacture of cyanide of sodium and the mixed (98/100%) cyanides of potassium and sodium are now brought to such a state of perfection that it seems useless to try and make cyanide here by the fixation of atmospheric nitrogen. Recent experiments have convinced me that—in this country at least—such processes are foredoomed to failure from a financial standpoint.—William Bettel.—*South African Mining Journal*, Aug. 7th, 1909, p. 640. (A. R.)

METALLURGY.

The Position of Amalgamating Plates in the Stamp-Mill.—"The ordinary position of the amalgamated copper plates in a stamp mill directly in front of the stamps is one which seems natural enough, but it is accompanied by a number of disadvantages. Among the advantages connected with a separate plate room are the following:—

1. The absence of noise and vibration caused by the stamps.

2 Much better lighting during daylight hours can be provided.

3. Increased space for working about the stamps, making repairs, etc.

4. Increased space also about the tables, which can be placed as far apart as desired, and built at a convenient height above the floor of the plate room.

5. Better arrangements can be made for altering the grade of the tables, etc.

6. It is easy to arrange a system of pulp distribution, so that the supply of pulp can be shut off any one table, and be distributed equally between the others, or as many of them as may be wished. This obviates the necessity of having to hang up stamps when dressing the plates or collecting amalgam. The ordinary plan of avoiding this by diverting all the pulp on one plate to a single adjoining plate is also obviously bad practice.

7. In consequence, there is no need for the amalgamator to hurry over his work. He can take his time, and do his work thoroughly.

8. By having the doors and windows of the plate room properly secured, casual pilfering by dishonest employees can be prevented, and the risk of robbery by outsiders reduced.

9. It becomes a simple matter to classify the pulp if desired, before passing it over the plates, and to vary the mode of treatment for each class.

I do not think that there is anything to be said against the separate plate room, except the slight initial cost, the need in some cases of an elevator to give the fall required for the launder conveying the pulp from the stamp shed to the plate room, and the fact that in small mills where only one attendant is employed to look after the stamps and plates, the latter would be less constantly under the eye of the attendant. Even then the advantages in most cases would outweigh the drawbracks.

The benefits to be derived from the separate plate room are such that it should be generally adopted in all new mills. In the majority of mills of old design the change to separate plate room might be advantageously made. I may add that the system here advocated was adopted by the Waihi Company some years ago, and, it is almost needless to say, with entire success."—*Mexican Mining Journal*, Sept. 15, 1908. (C. B. S.)

Tube Milling at El Oro.—"The first mills supplied the El Oro Company were from the Krupp foundry, having 1 in. steel liners. Silex linings were tried and discarded. We found we could cast a satisfactory chilled white-iron lining. The next move was to increase the thickness, and ribs were cast on the plate to resist the abrasive action of the pebbles. It was then noted the pebbles had a tendency to adhere between the ribs. To further this action the ribs were tapered, being wider outside than within. The pebbles wedged themselves, thus forming a flint lining automatically laid. Hence was evolved the 'El Oro' lining. These plates lasted some eight or nine months. In the course of time flint was substituted by hard mine quartz. This rock was not nearly as hard as flint, in consequence the permanent wedging of the pebbles lost much of its liner value. Still the ribbed plates last usually six months.

From a recent inspection of the plates it is apparent that, apart from the lodging of the pebbles, the chilled-iron ridges are in themselves no mean factor. In the first place they effectually break up any sliding action of the body of pebbles; in consequence, flat pebbles are now unknown. Then the ridges cause a tumbling over in wave motion of the pebbles, and finally assist in giving them a higher lift. These are evidently important features. There are two distinct principles involved in pebble linings. The one method employs a fixed bed, and the other a changing bed. The El Oro liner employs the former. This was advisable when using flint pebbles. With the softer rock it is possible the latter might prove more efficient. Plates are therefore designed in which the bed of pebbles is retained by suitable bars, but at each revolution the bed falls out and is replaced by other pebbles, thus forming successive beds. The life of the plate would therefore be independent of the hardness of the rock, and governed by the life of the bars, which are easily renewable. Theoretically, therefore, the plate would last indefinitely.

The economy of the mine rock over the imported pebble is well illustrated by figures.

Using flint pebbles, the consumption per month was about 8 tons for Krupp No. 3, 14 tons for No. 4, and 20 tons for No. 5 mill. With the nine mills in operation this is equivalent to 90 tons per month. At the present, with mine rock, 40 tons per day is consumed, or 1,200 tons per month (100 tons are maintained in the mills, varying from 8 to 16 tons in quartz), in other words, thirteen times by weight of mine rock to imported flints. This is, roughly, 5% of the monthly mill run. The pebbles cost $60 (Mex.) per ton, or 5,400 pesos per month. The mine rock averages $6 (gold) per ton. The increase to the bullion return through this crushing rock is around $6,000 (gold) per month. Allowing the cost of mining and handling to equal the value of the rock, it is apparent the cost of the pebbles, or 5,400 pesos per month, is saved.

It is true the consumption of the liners is greater and that the efficiency of the mill is reduced, as a lesser quantity of finally crushed product is liberated per cycle. In other words, the tube mills need augmenting. Taking everything into consideration, the economy lies with the mine rock.

Another interesting feature consists in the method of feeding the crushing rock into the mills. Owing to the wedging of the pebbles in the liners, the manholes were often opened with difficulty. Again, the quantity of crushing rock was largely increased, so that some method of continuous feed with constant operation of the mill was desirable. This was effected by means of a coarse screw-feeder attached to the discharge end, feeding through the trunnion. Rock by this means can be added at pleasure.

Although fed into the discharge end, it was found the coarse rock worked its way to the upper end, where the maximum breaking down of the ore particles takes place.

By such means were the tube mills converted from a frequent-change-of-liner and stoppage-for-pebble-feed machine to a pebble-bedded protected liner, with a continuous, non-stoppage feed of crushing rock—both items of economy.

The following is a brief statement of the costs averaged over eleven months:—

With liners costing 15 cents (Mex.) per kilo, and pebbles $70 per 1,000 kilos, in 11,000 tons crushed monthly, results were—

Pebbles used	45,000 kilos.
Liners	6,800 ,,
Pebbles per ton ground ...	9·1 pesos.
Liners	1·5 ,,
Cost per ton ground—	
Pebbles	30 c. (Mex.)
Liners	8·6 ,,

H. E. West.—*London Mining Journal*, July 31, 1909, p. 149. (A. R.)

Metal Ribbon.—This is now manufactured by an exceedingly simple, rapid and inexpensive process. A water cooled brass cylinder about 5 in. in diameter is rotated at a speed of about 1,600 revolutions per minute. While it is rotating at this velocity a thin stream of molten metal is allowed to fall on to the surface of the drum. Immediately it comes into contact with the latter it solidifies, and by centrifugal force is thrown off in the form of a continuous band of ribbon about ¼ in. in width by about $\frac{1}{500}$th part of an inch in thickness. By means of this process some 28 lb. of thin shavings can be produced per hour. The process is available for use with a variety of metals, such as lead, zinc, tin, etc.—*Chamber's Journal*, Aug., 1909. (A. L. E.)

Dressing of Battery Plates.—"A new system of plate dressing, invented by Mr. E. H. Martin, the battery manager of the Langlaagte Estate, Rand (S. Africa), allows the amalgamation plates to be dressed without hanging up the stamps. 'This is accomplished by placing a board across the head of the amalgamating table, and thus running the pulp over the back of a plate into a launder, which runs beneath the plate. The pulp is diverted, so that, instead of running into the main tube mill launder, it runs over a separate amalgamation table, which is kept specially for this purpose. Ordinarily a strip of wood ½ in. to 1 in. thick prevents the pulp running over the back end of the plate. When the board is put in place the pulp is dammed up, and runs over this strip of wood. The necessary alteration in the tables costs a total of from £3 to £3 10s. per table, including the necessary labour and materials. The change is accomplished by taking up the plate and relaying it on 3 in. × 9 in. timbers, leaving a 4½ in. × 3 in. launder under the plate.'"—*The Australian Mining Standard*, July 21, 1909, p. 57. (W. A. C.)

Calculations in Ore Concentration.—"These calculations are made with reference to the concentration of copper ores, the size of American plants for the purpose having increased very considerably in recent years, and hence slight errors in mill operation lead to heavy financial loss. The argument is as follows:—(*a*) *Assays.*

Let y = weight of concentrates from 1 ton ore milled.
z = weight of tailings from 1 ton ore milled.
a = assay of ore.
b = assay of concentrates.
c = assay of tailings.

Then

$$y+z=1 \tag{1}$$

$$by+cz=a \tag{2}$$

whence eliminating z, $y=\frac{a-c}{b-c}$ (3)

$$\text{and saving}=\frac{\text{amount of copper in concentrates}}{\text{amount of copper in 1 ton of ore}}$$

or

$$S=\frac{by}{a}=\frac{b(a-c)}{a(b-c)} \tag{4}$$

a formula giving the saving from assays alone. To check this there are three more possibilities, namely,

$$S=\frac{by}{ax}=\frac{\text{Cu in concentrates}}{\text{Cu in ore}} \tag{5}$$

$$S=\frac{by}{by+cz}=\frac{\text{Cu in concentrates}}{\text{Cu in concentrates and tailings}} \tag{6}$$

$$S=\frac{ax-cz}{ax}=\frac{\text{Cu in ore}-\text{Cu in tailings}}{\text{Cu in ore}} \tag{7}$$

If weights and assays are correct, these four methods should check, but in practice, of course, they do not check correctly. Equation (3) is also interesting because it represents the "rate of concentration" expressed by assays alone, and can be used for any class of ore and any other constituents in the ore, *e.g.*, S, Fe, etc.

(*b*) *Screen Tests.*—The author then extends this method of analysis to problems, such as the following. The tailings from a concentrator are classified hydraulically, the spigot discharge being used to build a dam across the mouth of a gulch adjoining the tailings end of a plant. The overflow is conducted into the basin thus formed, where the suspended solid matter settles, liberating clear water which is used again in concentrating. What percentages go (*a*) to the dam (*b*) to the pond? He shows how such a problem can be solved from accurate screen results and also from assays alone, the latter being used to check the former, errors being great only when the two products into which the one is split are quite different and not nearly alike. Hence where screen tests show particles of practically the same size assays should be used, and *vice versa* if assays are nearly the same, screen tests should be used, and will give better results. Finally, the question is asked whether it is of so great importance to know how the pulp is being distributed in a mill, and whether the millman cannot tell from the appearance of his machines whether the distribution is correct. On the basis of a comparison between actual determinations and estimates from appearance of the machines the author concludes that such estimates are in no way trustworthy."—Dr. RUDOLF GAHL, *Western Chemist and Metallurgist*, April.—*Electro-chemical and Metallurgical Industry*, vol. vii., 7, July, 1909, p. 306. (J. A. W.)

ELECTRIC FURNACES IN GERMANY.—"Hitherto electric furnaces have been used for small quantities of material, but now the leading works are beginning to employ them on an extensive scale for handling large quantities of metal. At the end of 1907 one of the most important steel works in Germany decided to replace its battery of crucible furnaces by a large electric fusion furnace, the main cause for the change being to lessen materially the cost of fusion without altering the quality of steel produced. The old equipment of 32 furnaces, each containing 6 crucibles with 77 lb. capacity each, was replaced by a single electric furnace of 250 h.p. holding 1 metric ton, a duplicate of this being installed as a reserve for emergencies. Electricity is supplied from Cologne in the form of a three-phase current of 5,200 volts, this being transformed to 110 volts. The production of cast steel has been carried on in this furnace since 1908 without interruption with success, the main product containing ·08 – ·18% C., ·03% S, ·06% P. Occasionally charges of tool steel have been cast containing ·7 to 1·3% C. with and without the addition of Ni, Cr. or W. The material employed is ordinarily slugs, cuttings, turnings, and other waste of wrought iron, costing from \$14 to \$15 per metric ton or \$13 to \$14 per short ton.

A fusion, including the refining, lasts about five hours and involves a consumption of about 900 kw. hours for 1 metric ton of cast steel. The entire charge of 1 ton is poured at once. When the furnace is kept in continual use the magnesite lining lasts for about 70 operations, *i.e.*, 20 to 25 days. A stoppage of five days is required for each renewal, but during this period the reserve furnace is employed.

The cost of the electric current is exceedingly low, on account of the large amounts consumed. It amounts to 4½ pfennigs (1·071 cents) per kilowatt hour, so that the expense for a single operation is about 40 marks or \$8·64 per short ton.

The total cost of cast steel, thus produced, per metric ton, is as follows:—

For—	Cost.
Amortisation of cost of plant	\$1·19
Raw materials, including additions, chromium, etc.	16·66
Electric current, including heating during intervals	10·71
Renewal of magnesite lining	2·85
Labour	2·38
Electrodes	·60
Water for cooling purposes	·12
Total	34·51

This amounts to \$31·32 per short ton, and is materially less than the cost of crucible steel.

Advantages of the Electric Process.—The manufacturing firm notes in addition these advantages in the electric process, as compared with the old method: (1) Cheaper raw material can be employed in the place of the more expensive Swedish iron formerly required; (2) in fusing quantities of a ton at a time it is possible to produce homogeneous masses of a distinct, well-defined grade; (3) the molten steel leaves the furnace at a much higher temperature than when prepared in crucibles, is therefore in a more liquid condition, and is easily transferred to ordinary ladles, preparatory to pouring into the moulds of small castings.

These electric furnaces are arranged in the Stassano system, and are supplied in all sizes, for charges varying from 400 lb. to 10 tons, and are adapted for the fusion of metals of every kind.

The furnace for cast steel is constructed of iron plates, heavily lined with magnesite bricks. Carbon electrodes enter by three openings on the sides, and are so adjusted as to terminate slightly above the surface of the molten metal, whereby the heat of the arc is absorbed almost entirely by the contents of the furnace, and but a small portion is radiated upward and is reflected from the concave dome.

Mixing and Fusion—Other Uses.—A thorough mixing of the component parts of a charge is effected by mounting the furnace on an axis, slightly inclined from the vertical. A steady revolution about this axis is maintained through a cogged connection, with a dynamo of 5 h.p., the edge of the furnace resting on ball bearings. This rotation causes the lower layers to gradually rise to the top of the molten mass, and brings about a thoroughly uniform mixing. The electrode supports are inclosed in cylinders and surrounded by a current of cold water. These supports are in rigid connection with hydraulic cylinders, by means of which the movements of the electrodes are under perfect control.

A fusion requires 3½ hours to be complete: a further period of 1½ hours is requisite for the removal in succession, of phosphorus and sulphur, and the final deoxidation of the contents of the furnace. No interruption occurs in either the current of electricity, or in the supply of water for cooling the electrode supports, while the furnace is in rotation. Special devices have been provided for both purposes.

The services of three workmen are required to conduct an electrical furnace, change electrodes, transport the raw materials, etc. Furnaces of the type in question are now being introduced in Germany for the treatment of other metals and alloys, and it is apparently only a question of time before the crucible processes hitherto in such general use will be completely superseded."—*Consular Report—Journal of the Franklin Institute*, Aug., 1909, p. 104. (J.A.W.)

THE OXY-HYDRO SMELTING PROCESS.—The Mornya correspondent of the *Sydney Morning Herald*, telegraphing on the 9th July, gives the following interesting description of a new process for treating refractory ores;—

A new smelting process for the treatment of refractory ores has been started on the lease of the Moruya Gold and Silver Syndicate, with the object of extracting the gold and silver from an ore body, embracing, in addition to gold and silver, also arsenic, zinc, galena, antimony, and traces of copper, which render it impossible to treat effectively and economically by ordinary mining methods. If the inventor of the new smelting process—Mr. Frank Cotton, of Newtown—succeeds in achieving all he claims for it, and it proves a commercial success, it will mean a revolution in the treatment of refractory ores in Australia.

About four months ago Mr. G. G. Johnston acquired this property, which, in the records of the Department of Mines, is the most refractory proposition in the State, and set to work to prove what had already been proved in the laboratory—that the problem of 50 years' standing had been solved, and that the ores of this mine, and other refractory ores, could be profitably treated. This result was not brought about without nearly two years of research work, which brought to light the process known as the oxy-hydro process.

The furnace has a floor-space of 98 square ft. The ore is passed into it, and is treated for from half an hour to an hour and a half, according to the density of the sulphide and the percentage of arsenic to be eliminated. It is then drawn from another door into the cement-floored ore bins, behind the two five-head stamper batteries. From the boiler an insulated steam pipe runs to the furnace and passes along the top of it. Immediately over the arch of the firebox there is a muffle furnace, and in this the steam pipe is carried backwards and forwards in the shape of a grill. From the muffle the pipe is split into two sections, which are carried down to the foot of each end of the firebox. One end of each section is used for blowing off any condensed steam at starting, and the other end is carried into the firebox through a wrought iron tuyere of 2 in., in which the steam pipe is centred. At the end of the pipe is a cone with an aperture of one-eighth of an inch. The action of the dry steam on the charcoal accumulated at the bottom of the surface is the dissociation of the steam combining with the carbon of the charcoal, forming carbon monoxide and free hydrogen. The heavier hydro-carbons of the wood rise to the hollow arch on the top of the firebox, are there volatilised, and drawn down again, having the effect of a carburetter, and making a fuel gas of high value. Sufficient air is drawn in by the tuyeres automatically at the base of firebox to burn the gases over the hearth of the furnace. Beyond the ore chamber is a dust chamber, with an intermediate stack, and beyond that are three condensing chambers, each 7 ft. by 7 ft. by 7 ft., through which the fumes pass alternately top and bottom of the walls. In these chambers the arsenic eliminated from the ore is thrown down, the operation being assisted by sprinklers across the top of each chamber. From the furnace the treated ore is passed through the battery, the pulp passes over a Card concentrating table, and the residues go on to the cyanide plant. The result is a free, high-grade concentrate, which is treated on the ground by cyanidation, and the recovery of a white oxide of arsenic, for which latter there is an unlimited market at from £35 to £40 per ton.

The preliminary runs with this plant have been made on all classes of the ore in the mine—from the partly oxidised ores to the dense sulphides. The results in each instance have been the same, and the plant is now running full time. Ore containing from 20 to 30 per cent. of arsenic, as well as other refractory elements, has been successfully dealt with. Residues previously requiring 12 days in the cyanide vats with special treatment have been operated on in four days by reason of the elimination of the refractory elements, and the extraction from the treated material has been 90 per cent., as against 75 per cent. from the raw material.

In the presence of Mr. Warden Shepherd, Mr. G. G. Johnston, who is now the general manager of the Moruya Gold and Silver Syndicate, and other representative people, sulphide ore was treated, and from the parcel put through 6 cwt. of white oxide of arsenic was recovered in the chambers. The Warden selected a piece of raw ore, half of which he had treated in the furnace, and he has now forwarded the raw and treated parcels to the Department of Mines for analysis.

In addition to arsenical gold ores, the oxy-hydro process has been proved on the following:—Arsenical copper ores, carbonate copper ores, zinc blende, copper ores, lead-silver, zinc ores, antimony ores, and lode tin ores. By the elimination of the refractory elements it is claimed that the ores of mines now lying idle can be successfully handled. It is also claimed that not only are the ores rendered possible of treatment, where they have previously defied the metallurgist, but the refractory elements are turned into marketable products of high value. It is also said that the cost of treatment is reduced to a minimum. On the furnace now running the cost of firewood is said to be not more than 1s. per ton of treated ore, and the steam consumption does not exceed 1 h.-p. per hour. The capacity of the furnace on dense sulphide is 50 tons per week and up to 75 tons on the higher sulphides.

The main reef of the mine selected for the proving of the process on arsenical gold ores has a north and south bearing in slate country, west of the granite belt about 200 yards, and is an average width of 3 ft. 6 in., with 18 in. of mineralised formation. The ore is firstly of a mispickel composition; but, in addition to the arsenic and iron, it contains zinc blende, galena, antimony in sulphide form, and traces of copper. Such a conglomeration of refractory elements is seldom met with, and yet it obtains more or less in all the ores raised on the Bergalia and Donkey Hill fields. At periods the reef is divided by horses of slate and granity sandstone, and they are interlaced with heavy mineralised lode matter; but the hanging and footwalls are true slate, and have always remained well defined.

In its issue of the 11th July the *Herald* says:—'Particulars of the oxy-hydro process of dealing with refractory ores published in the *Herald* yesterday, have been the subject of comment in mining circles. Upon enquiry at the Department of Mines yesterday it was ascertained that what our Moruya correspondent says about the refractory nature of the ores dealth with there is quite correct. They are classed as almost impossible, and if complete success is achieved in the treatment of them big results are looked for. Mr. Cotton is not unknown in official circles. He has been working at it persistently for some years, and has got nearer to dealing with refractory ores in commercial quantities than ever before. For a long time he has been able to show good results with small quantities, and the true test lies in the operations he has now undertaken. The metal is dealt with in solid chunks, without crushing, and afterwards treated by the ordinary cyaniding process. The refractory ores, after undergoing the process, are spongy and workable.'"—*Queensland Government Mining Journal*, Aug. 15, 1908, p. 409. (C. B. S.)

THE METALLURGY OF BISMUTH.—"Generally (says the *Mexican Mining Journal*) very little is known of the methods employed to win bismuth from its ores. Schnabel says that ores containing bismuthine (sulphide) alone can either be treated by calcination and fusion with carbon, or in the raw state by fusion with iron. In the one case, the chemical processes are the same as for the extraction of lead from galena by calcination and reduction; in the other case the process is the same as the extraction of lead by precipitation. When ores contain considerable quantities of gangue, the bismuthine may be separated from it by a simple fusion without the addition of iron. Oxidised ores of bismuth, as a rule, are mixed with native bismuth and are treated together. They can be treated, mixed with charcoal, by being fused in crucibles or reverberatory furnaces. When bismuth is present as oxide it is generally dissolved by hydrochloric acid, more rarely by nitric acid which is far dearer. Bismuth is precipitated from solutions either directly or as basic salt. It is genarally thrown down as metal by means of iron. As a basic salt it can be thrown down by diluting its chloride with water, when an oxychloride is precipitated. From the oxychloride bismuth can be reduced either by fusion with lime and charcoal, or by treating the moist salt with iron or zinc. The addition of lime is necessary when the oxychloride is treated in the dry way, in order to combine with the chlorine. The lime used may be either slaked or caustic. Metallic bismuth thrown down in the wet way is melted in graphite or iron crucibles, with the addition in case of need, of alkaline flux. The wet way has been employed for oxidised ores at Meymac, France, and for calcined tin ores rich in bismuth at Altenberg, in Saxony. The ore treated at Meymac is a hydrated carbonate of bismuth in a quartzose gangue, containing small quantities of arsenic, antimony, lead, iron, and lime. The crushed ore is treated three times over, in earthenware vessels, with hydrochloric acid, at a gentle heat and by constantly stirring. The nearly exhausted ore is treated with fresh hydrochloric acid, while the almost saturated solution is allowed to act upon fresh ore, saturated solutions being thus obtained, while the bulk of the bismuth is removed from the ore. The bismuth is thrown down from the solution in the form of a black powder by means of bars of iron. Immediately after its precipitation the bismuth is separated from the solution, washed with water, pressed in canvas bags, and is finally melted in graphite crucibles, under a layer of charcoal, and is then cast in iron moulds."—*Queensland Government Mining Journal*, Aug. 14, 1909, p. 395. (C. B. S.)

MINING.

ELECTRICITY IN COAL MINING.—"Some interesting figures have been obtained regarding various percentages of dust from hand, puncher-machine, and chain-machine-mined coal. The results found in each case have been reduced to the number of pounds of dust per square foot undercut, and the dust gathered was screened through a 40-mesh screen. At the mines of the Washington Coal and Coke Company, on the lower edge of the Connellsville region, the results obtained were as follows:—

By hand-mining, 1·80 lb. per square foot undercut.

By puncher machine, 4·58 lb. per square foot undercut.

By chain machine, with pick-pointed bits, 1·48 lb. per square foot undercut.

At the mines of the Westmoreland Coal Company the puncher machine gave 6·18 lbs., and the chain machine, with the standard chisel bits, gave 2·43 lbs. of dust. At the mine of the Pittsburg Coal Company, Midland No. 1, the chain machine gave 2·43 lb. and 2·51 lb. at two different trials, and the puncher machine gave 6·18 lb. There is no disputing, of course, that the puncher machine will cut more large coal, or in other words that the average coal cut by the chain machine will be smaller than that cut by the puncher or by hand, but the amount of coal cut out in undercutting by the puncher is so much greater than the amount taken out by the chain machine, that the actual amount of dust will be greater, as given above.

It has been repeatedly attempted to ignite very fine coal dust by an electric arc. I have no doubt that this can be done, but up to the present time it has not been done, so far as I know. I am inclined to think that if this ignition occurs, it will be with the distillation of gas from the dust by the heat of an electric arc. There is, of course, not the slightest doubt that an electric spark of almost any potential will ignite gas in proper percentages, since we have an illustration of this in practically all automobiles. It is probable, to my mind, that an explosive mixture could be ignited from the minute spark from the ordinary signal wires where ten or more cells of battery are used.

Turning to the question of safety of the men, it has been considered useless for some years past to insulate electric wires underground, since there seemed to be no commercial insulation which was of value after a few years' use, or in some cases, a few months' use. It has, therefore, been preferred to use bare wires, even for machine lines in room heading, and depend on the miners' intelligence to recognise the danger of a bare wire. In some mines these wires have been protected by means of boards nailed to wooden plugs along the rib, and the wires suspended behind these boards, also protected by wooden troughs where they cross over the heading. This protection is effective but very expensive, and must be renewed at intervals of every two or three years in most mines. At the testing station of the Geological Survey in Pittsburg, the electrical department is making some experiments on various insulations subjected to extreme mining conditions. Cables with rubber and other insulations are submerged in acid mine water, readings taken of their initial insulation resistance, and these readings repeated at frequent intervals. The weakness of the insulation will be shown by the gradual or sometimes sudden drop in these values. This can be plotted on a curve, and the permanence of the insulation commonly called varnished cloth or varnished cambric will be effective under mine conditions, and I have used it in severe service for over two years without any sign of trouble. It does not seem to be affected by acid and is perfectly moisture proof. The cost of this insulation will, I believe, be considerably less than the best rubber, and if the results obtained at the testing laboratory measure up to those I have found in practice, I believe it will be only a short time until we can confidently recommend that all machine wires carrying 500 volts or over be insulated, where carried in room headings.

The use of electric lights in mines is strongly recommended, particularly at partings, turnouts, and all points where men must travel along or across the rails. In many cases these lights can be connected directly from the trolley and in connection with the use of automatic trolley switches; these lights will show when the current is on. It is advisable to have more than one circuit where lights are used as danger signals, to provide for burn-outs of the lamps. In some of the mines, in the Connellsville region particularly, one or more red lights are installed at each danger point, in addition to the white lights, which are connected at back of the automatic switches. The white lights burn at all times, but the red lights show only when the current is on the wires at the parting.

In all cases where large tracts of coal are to be developed I would recommend the use of 275 volts potential, using high-tension alternating-current transmission from the central plant, and locating sub-stations as close to the work as possible. Where the cover is excessive, these sub-stations can be located underground at the centre of distribution.

Many mining men are possibly familiar with plants on the three-wire system, of which a number have been installed. In these plants the current is generated by standard railway machines at 550 to 600 volts. The neutral tap is taken off these generators by means of proper connections to the windings from collector rings, and these taps by means of balancing coils furnish a neutral lead, between which and either main lead the potential is one-half that of the generator. Apparatus in the mine is operated between either main lead and the rail, which is the neutral, and the maximum potential which can be received will be 300 volts. Stationary apparatus, such as fan motors, car hauls, large pumps, etc., can if desired be operated on full potential, or if a wide speed range is desired, motors may be operated first on 300 and later on 600 volts. The wiring in the mine is so sectionalised that an approximate balance of load is maintained between the two sides of the system, and in the case of a perfect balance, the system is equivalent to a metallic circuit 600-volt transmission using twice as much copper as a grounded return system, or one-half the copper as compared with a 300-volt grounded return system. Where there are a large number of small units operated, such as locomotives, machines and pumps, a close balance may be maintained, and the efficiency of the transmission depends on the balancing, since with total unbalance it will be a straight 300-volt transmission."—GEORGE R. WOOD, *Coal Mining Institute of America — Iron and Coal Trades Review*, July 30, 1909, p. 163. (A. R.)

DUST COLLECTOR FOR ROCK DRILLS.—"At a recent session of the Société de l'Industrié Minérale, M. Remaux gave a number of suggestions as to the use of rock-drills, and, among other novelties, described an easily-made apparatus for catching the dust as it comes from the drill-hole, thus avoiding the use of a hollow drill with spray of water. He objects to the latter method of preventing contamination of the air, because the mud and the dampness are almost as disagreeable as the dust would be. His apparatus consists of a conical wire frame, which can be made of 3-mm. steel wire, by coiling it on a wooden mandel; the resultant coil is covered with flexible leather. The small end of the cone is fastened to the forward part of the drill cylinder. The large end has a heavy leather washer, and a tight contact between the washer and the rock wall is made with clay. The air escaping around the drill piston is sufficient to carry the dust off through the discharge at the back end of the cone, leading into a pail of water. The dust collector has a length of 75 cm., and a diameter of 5·5 cm. at the small end and 10 cm. at the large end. Only this one size is required, for a bit will usually get dull, and will be replaced by a longer bit after every advance of 50 cm., which is within the range of elasticity of the dust-collector."—*Queensland Govt. Mining Journal*, July 15, 1908. (C. B. S.)

MOISTURE IN MINE AIR.—"Attention has already been called to the circumstance that the proportion of oxygen and nitrogen in the free atmosphere remains uniformly constant. Moisture, however, which is always present, varies in amount very considerably.

A singular circumstance, with regard to the varying amount of moisture in the atmosphere, is the fact that under certain conditions the atmosphere may contain a large proportion of moisture and still not be as damp as at other times when there is actually less moisture present. This paradox becomes intelligible when it is explained that the capacity of the atmosphere for moisture is a question of temperature. Perhaps this explanation may be still further elucidated by the following figures, which set forth the amount of moisture, in grains per cubic foot, which the air is capable of holding at various temperatures.

Temperature in deg. F.	Grains per cub. ft.
30	2·0
35	2·4
40	2·9
45	3·4
50	4·1
55	4·9
60	5·8
65	6·8
70	8·0
75	9·4
80	11·0
85	12·8
90	14·8
95	17·2
100	19·8

These figures show the amount of moisture in the atmosphere at various temperatures when the air is 'saturated,' that is to say when the air has taken up all the moisture it can carry *invisibly*. If the temperature of saturated air be reduced, some of the moisture is condensed and becomes visible, either as rain or mist. For example, suppose a body of saturated air, at a temperature of 70°, is cooled to 49° or 50°, practically one-half of the moisture will be deposited—about 4 gr. per cubic foot.

The above figures not only show that the capacity of the atmosphere for moisture increases with the temperature, but also that the capacity for moisture increases more rapidly than the temperature. The increase of moisture, for example, from 30° to 40° is less than one grain per cubic foot, but the increase in the capacity for moisture at the other end of the scale, for a corresponding increase of temperature, from 90° to 100° is 5 grains per cubic foot.

Of course it must be understood that the air is not always saturated, or even nearly saturated. On the contrary it is often very dry, that is to say that for a given temperature the amount of moisture present may be much less than the amount which the air could contain.

In fairly hot summer weather, with a temperature of say 80°, the air would be dry although it carried as much as 5 or 6 grains of moisture per cubic foot. On the other hand, in winter, with a temperature of say 40°, the air would be very damp if it only carried $2\frac{1}{2}$ grains per cubic foot.

The question of the amount of moisture present in the atmosphere of mines is of great importance, especially in deep mines.—*Mining Engineering*, July, 1909, p. 156. (A. R.)

THE HEAT OF DEEP MINES.—"It is somewhat difficult to account for the extremely even rise in temperature obtaining in the Rand. The increase in temperature proportionate to depth attained is due in the main to the secular heat of the earth, but is also dependent on a number of contributory conditions, such as the presence of highly pyritic or carbonaceous shales, or the percolation of warm springs. The Rand, it seems, is little affected by such causes, and the fact that at the bottom of the Brakpan and Jupiter shafts the rock temperature is only 82° F. is a most happy augury for the future expansion of deeper level mining on the Main Reef beds. It should be remembered that hitherto the question of ventilation has not been given any very great amount of consideration on these fields. We scarcely think it has required it, except in a few exceptional cases, and we write with some experience of the conditions obtaining in other parts of the world at depth. With extensive artificial ventilators, working in conjunction with the higher air pressures which it is proposed to use, there is no reason why work should not be carried out at an even greater depth than that spoken of by Mr. Phillips; no reason affecting the miners' health, at any rate. Let it also be remembered that co-operative workings and the connecting up of all shafts and drives will further improve matters underground.—*South African Mining Journal.—Science and Art of Mining*, July 24, 1909, p. 577. (A. R.)

Temperature Gradient. — "The temperature results obtained in a recent bore hole, put down to a depth of nearly 5,000 ft., are interesting. The records show that the average gradient from the surface down to the lowest point in the bore hole was 1° F. in 87½ ft. This result compares with 1° F. in 64 ft., as noted in the report of the British Association for the Advancement of Science."—*Engineering and Mining Journal*, July 31, 1909, p. 226. (A. R.)

A Solution of a Diamond Drilling Problem. —" Recently in putting in a diamond drill hole on the property of the Michigan Copper Mining Co., Rockland, Mich., a breakage occurred, that was dealt with in a peculiar manner, and, in view of the importance of diamond drilling as a means of prospecting, the following note may be of interest. The mode of procedure was original, in so far that no one connected with the drill work at the mine had ever seen or heard of a similar solution of the problem.

A nearly horizontal prospecting hole was being drilled from a point in one of the underground workings. The machine was taking out a 1⅛ in. core. The core-shell had become worn thin by use, so that, at a depth of 460 ft., it buckled under the working pressure of the rods and jammed in the hole.

The first step was to get the drill rods out of the hole. Jack screws were applied at the mouth of the hole to a gland bolted on the rods, sufficient strain being applied to break off the core-shell close to the core-lifter. It was found that the force required for this stretched the 460 ft. of rods 6 in. Next, the stump of core left in the hole had to be removed. As a chopping bit could not be used for this purpose, without great risk of injuring the diamonds, a special tapering bit was made. The larger end of this bit fitted closely enough in the hole to centre and guide the boring end in grinding out the core. This done, the bit was replaced by a taper tap, with which the core-lifter was recovered.

Finally, a pointed drill was made of tool steel, properly tempered, and put on the rods. With this the remains of the broken core-shell, together with a portion of the original diamond bit, were cut away until the bit itself was loosened in the hole. What was left of the bit, though only 1 in. long, was then easily recovered by a tap, and the diamonds were found to be uninjured. In using the pointed drill, great care had to be taken to avoid running into the diamonds. To this end, the feed of the drill was accurately measured while the cutting was in progress. On recovering the bit it was found that there was still a margin of ⅜ in. of steel above the first diamond."—Clarence M. Haight.—*The Mining World*, July 10, 1909. (C. B. S.)

Mine Sampling Device.—" In mine sampling a simple and efficient device for catching the rock as it is chipped or broken is made as follows:—Take a piece of stiff wire about 5 ft. long and bend it into the form of a square, with sides 10 in. or 11 in. in length; the surplus wire remaining on each end of the piece is bent outward from the middle of the open side to form the handle of the device. Now take a piece of canvas or heavy duck 4 ft. in length and about 20 in. deep and shear off the ends diagonally so that the length of the strip at the top shall be 4 ft., and that at the bottom about 2 ft. Lap the upper end over about ½ in. and stitch it, providing ample room for the wire to pass through. Sew the sheared ends together, thus forming a funnel-shaped bag. Now pass the wire through the lap or hem, and the bag is complete. While catching the sample, a stout piece of twine is tied in a slip knot around the bottom of the bag. When it is desired to transfer the sample to the sample sack, the twine is removed and the contents of the bag poured into the sack. By the use of this bag, samples may be taken from points which seem inaccessible when using a candle box or some other method of catching the rock."—*Engineering and Mining Journal*, July 31, 1909, p. 226. (A. R.)

A Tunnelling Record in California.—" A progress of 579 ft in the north heading of the Fernando tunnel, during the month of May, establishes a new world's record for hand work tunnel driving.

This is one of a series of tunnels being driven in conjunction with the construction of the Los Angeles Aqueduct, perhaps better known as the Owens river project. The Fernando tunnel is located 25 miles north of Los Angeles, and pierces the Sierra Madre range of mountains, at an altitude of 1,450 ft., and when completed will be 9,112 ft. in length, being the second longest tunnel on the aqueduct. The formation is Cenozoic Tertiary, in which are many fossils in a high state of preservation. The ground is a soft sandstone, containing many nodulous boulders and frequent strata of hard altered sandstone.

Immediately surrounding the line of this tunnel is located, what is known as the Elsemere oil field, many of its derricks being 500 ft. above the level of the tunnel. The pumping level of the field being about 350 ft. below the tunnel level. No gas of any consequence has been encountered in this field; due largely to the fact that the oil is of an asphaltum base. Some oil was encountered in the first 500 ft. of the tunnel driven, but to date no gas, and very little water, has been found in the north heading. A total progress of 1,932 ft. has been made in the north heading since December 20, 1908, the date excavation was begun.

The volume of excavation is 5 cub. yd. per lineal foot in the timbered section, upon a slope of 0·0005 with a hydraulic capacity of 430 second-feet.

The form of excavation section is full arch-back, vertical sides, flat bottom; timbered with 8 × 8 Oregon pine timbers, a 4-piece segmental arch cap being used. The dimensions inside timbers being 10 ft. 8 in. wide, by 11 ft. 2 in. high.

In excavating, the full section was carried forward at one operation, no bench work nor trimming being resorted to. The mechanical equipment is as follows: One 10 h.p. electrically driven blower for ventilation; one electric locomotive of 1,200 lb. D.B.P.; 19 steel muck cars, of rocker dump design, of 32 cub. ft capacity; a motor generator set to furnish D.C. current for the locomotive.

In the machine shop is a 15-in. engine lathe, drill press, emery grinder and hack saw, all power driven. The timber framing sheds contains one swinging cut-off, one table and one band saw, driven by an 18 h.p. A.C. motor. The blacksmith shop has all of the necessary tools and equipments for the work. A change room is provided, equipped with three showers, hot and cold water for the convenience of the employees.

The excavation and timbering was carried forward by three shifts, of 16 men each, classified as follows: One shift boss, six miners, six muckers, one car shifter with mule, one motorman, one dumpman per shift. In addition, one boss and four men to lay track and maintain same. The outside force engaged was, two blacksmiths, one helper, one machinist, one electrician, timber foreman, three framers, one labourer, one janitor, one construction clerk and foreman. The superintendent's time being divided between the four camps on section No. 3. The timbers are placed by the regular heading crew on 6 ft. centres, and are kept close up to the face on account of the ground being unsafe to open, unless supported, which is done in the usual manner, by fore-poling over the two top segments of the arch. The sides seldom need support, due to the fact that the ground is large and blocky.

Each round requires from 14 to 17 holes to break the ground. The side holes being usually 6 ft. in depth, and the cut holes 8 ft., the down V cut system is used.

The miners' wages are $3 per day and muckers $2.50. In addition a bonus of 25 cents per foot per man for all progress above 10·5 ft. per day was paid; only the miners and muckers working in the heading participating in the bonus earned. The nearest approach to this progress was made in the Ricker tunnel, Switzerland, in 1903, where a progress of 531 ft. was made in a 31-day month. The details in unit cost and excavation area have been unobtainable.

In compiling the following data I have assumed the unit cost base, in labour, material and supplies, instead of the usual dollar and cent method. In doing this I believe it to be more intelligible for comparison.

The unit base I have adopted is per cubic yard of excavation, and the cost is as follows: Powder 2·12 lb., candles 0·2 lb., fuse 6·2 ft., timbers (B.M.) 26·2 ft., power 9·8 kw. hours' labour 0·6535 men days. It is my intention to give such detail of operative equipment and unit cost, and will enable engineers and tunnel superintendents to gain an understanding of the work as performed. In the majority of such articles, more especially those pertaining to tunnel construction in foreign countries, only the salient features are touched upon, leaving the reader in the dark so far as the value of the work is concerned, from the standpoint of unit cost.

Leaving out the local cost in dollars and cents of the work on the Fernando tunnel (which is not permissible in this article), I believe the work done in the month of May is unequalled in the history of hand work tunnel driving as regards progress and unit cost.

In a further consideration of the work I wish to call attention to the fact that to team training is largely due to the credit for the high rate of progress, for even should we have a high wage without intelligent work the same progress could not have been accomplished.

For 60 days previous to the record month the crews on the south, as well as the north heading, were being instructed in the methods evolved as being the most productive of speed and economy, for it was hoped, the ground being good (which I had reason to believe would prove true) to complete the driving in the coming December. In April the crews on the south heading made 510 ft. and the north heading 487 ft. During May some very hard strata, with a heavy flow of water cut the progress in the south heading to 446 ft. It has been my experience from the standpoint of speed and economy that team work is desirable and should be adopted more extensively by superintendents and contractors. It means a better and more reliable class of work done, because it is done by the more intelligent class of workmen, for it is this kind of men who seek this class of employment, as a result better accommodations and sanitary conveniences are required, all of which tends toward a higher development in the industrial world.

The progress for the present and near future will be materially reduced, due to an unexpected reduction of 66⅔% of the bonus and 12½ reduction in the number of men in the heading crews."—J. W. Henderson.—*The Mining World*, July 31, 1909, p. 270. (C. B. S.)

MISCELLANEOUS.

Some Notes on Testing Wire Ropes.—"The writer annually tests from 160 to 200 ropes, and it has occurred to him that others may be interested in the results obtained when testing ropes in use.

Most of the ropes used in Queensland mines are of simple construction—that is to say, of six strands, each containing seven wires, of Lang's lay, having an ultimate stress of from 17 to 35 tons. Frequently one wire in each strand is merely a core wire, being made of low-grade steel or of iron. Unless the core wire is equal to the other wires in the strand it is not considered in calculating the ultimate stress of the rope. A few compound ropes are in use having an ultimate stress of from 45 to 60 tons.

Manufacturers have adopted a classification of ropes presumably based on the composition of the steel, but this classification, judging by the results of tests, does not always present the uniformity that is desirable. In its report the Transvaal commission states:—'That the deterioration of a winding rope should be capable of being assessed by a competent person, while making the customary examination, is a most essential point.' In Queensland sets of rope-testing machines have been placed in centres most convenient to the Inspectors of Mines. With these machines wires are subjected to bending, torsion and tension tests. The two first-named tests, of course, have reference to the temper of the steel.

It may not perhaps be out of place to here describe the system of rope testing adopted by the Department of Mines, in this State. When a new rope is purchased a piece is cut off by the purchaser, and sent by him to the Inspector of Mines, together with a copy of the manufacturer's certificate. That officer then tests it and enters the results in a register kept for the purpose, and also furnishes the purchaser with a copy of the entry in the register. Afterwards, whenever the rope is re-shod a similar test is made, the wires at the same time being carefully examined for signs of deterioration. The wires are tested singly, and in calculating the ultimate strength of the rope a deduction of 10% is generally made from the aggregate of the wires, experience having shown that when the result obtained by this method of testing is compared with the result obtained by testing a whole piece of rope this is a fair average deduction. In comparing tests of new ropes with the manufacturers' certificates of the ultimate stress, it is found that a variation of 8% below the guarantee is permissible, as it is impossible to make steel wire perfectly uniform in temper, strength, and composition.

Exception has sometimes been taken to laboratory tests of wire ropes, on the ground that there is no guarantee that the piece so dealt with will reveal the defects in the whole rope. This at the first glance may appear to be a very strong objection, and there are those who argue that the only way to test a wire rope is to attach to it a weight equal to about one and a half times its usual working load. Then if it does not break it is said to be safe. To this argument it may be replied that a dead weight test is, if unintelligently applied, a source of considerable danger. The writer has known a new piece of rope 500 ft. long, when subjected to this test, to stretch 11 ft, and take three weeks to shrink to its former length. During this time it will be readily understood that the engine-driver was subjected to considerable annoyance. When applied to a rope that has seen service, the dead-weight test may be a death trap, since having stretched very much while working it will require very close observation to determine the amount of elongation, if any, under test. The rope's limit of elasticity may be exceeded, and what may be called 'a permanent set' put in it which may cause it to break in the near future.

The customary examination of winding ropes at mines is generally made once a week, when the ropes are run through some cotton waste held in a man's hands, and unless broken wires are found, the rope is said to be in good order. As a rule, little notice is taken of the flattening of the wires, and no effort is made to ascertain the amount of corrosion.

The chief causes of deterioration in winding ropes are wear, corrosion, and crystallisation. In a vertical shaft wear is caused by the rope coiling on the drum and passing over the pulley wheel. This causes a flattening of the wires, more especially when the drum is narrow, and the rope coils several times on it. Sometimes, also, a new rope is damaged by being put over an old pulley wheel which has been grooved by a smaller rope. In an underlie shaft there is, in addition to these causes, wear due to the rope passing over rollers in the shaft, wear caused by changes of grade causing the rope to bang up and down between the hanging and foot walls of the shaft, and in a flat and underlie shaft wear is caused by the rope being dragged along the footwall. In many vertical shafts in metalliferous mines the shaft traffic is so small that after three or four years' work the wires show hardly any flattening. Corrosion may be either internal or external. If the former, it is probably caused by water getting into the heart of the rope. This kind of corrosion is extremely difficult to find out, and frequently it is not suspected when the rope breaks. Water often finds its way into the centre of a rope owing to neglect to properly clean and oil it, and owing to the use of a stiff lubricant, which does not penetrate beyond the outside of the rope. Some lubricants that act very well in a cold climate become gummy in Australia. Outside corrosion may be caused by bad lubricants failing to protect the wires, or by water in the shaft.

When selecting a lubricant for a rope, the greatest care is necessary, as acids in a lubricant may do great damage. Formerly a favourite lubricant consisted of Stockholm tar and castor oil. Since it was discovered that Stockholm tar generally contains acetic acid, its use in lubricants has been largely dispensed with in Queensland. A graphite preparation, which seems to be fairly satisfactory, has been much used. Water is, of course, generally met with in sinking shafts, and in some cases acids are associated with it. In wet-sinking shafts, and in other shafts in which much water is hoisted, the ropes are often quickly rendered unsafe owing to corrosion.

After ropes have been in use for some time they begin to show signs of crystallisation, probably to a considerable extent due to vibration; and as a rule it is found that the lower end of the rope becomes brittle first. It frequently happens that a rope has to be subjected to the torsion test several times until it is cut at a point behind the pulley when the cage is at the top of the shaft. A case recently came under my notice at a mine where the engines were kept bailing water continuously. Some 300 or 400 ft. were kept coiled on the drums, as the bailing was not carried on from the bottom of the shaft. The engine-driver one day found a great many wires broken in that part of each rope that travelled in the shaft. A few days afterwards he found that the wires in the ropes coiled on the drums were also badly broken. Not having been much used, these wires showed very little flattening or corrosion. These ropes were of simple construction, with a rather high ultimate stress, and the wires were of a large diameter, becoming brittle quickly, as is often the case in this class of rope."—ROBT. HUNTER.—*Queensland Government Mining Journal*, Feb. 15, 1909, p. 64. (C. B. S.

TRANSVAAL IRON ORES AND STEEL PRODUCTION.—"A memorandum has been issued by Mr. R. N. Kotzé, Government mining engineer, on iron and steel industries. He points out that such industries are among the most valuable assets of any country, on account of the extent to which iron bulks in the materials required by almost every trade, its permanence when once established, and the numerous subsidiary enterprises to which iron and steel works give rise. Discussing the possibility of establishing such works in the Transvaal, he writes that the principal deposits of ore likely to be of economic importance are as follows:—

(*a*) Magnetite.—This ore occurs in very large deposits in various localities. The best-known and most easily accessible is probably that on and about De Onderstepoort, some nine miles north of Pretoria, where the ore is found in the immediate vicinity of the railway line. The quantity probably runs into many millions of tons. The deposits in the Steelpoort Valley, and north of it at Magnet Heights, are on a still larger scale, and may be looked upon as almost inexhaustible. The percentage of iron in this class of ore is high, varying roughly from 55 to 65%, but it suffers from the disadvantage of containing an excessive percentage of titanium, which ranges up to 12%. Owing to this latter ingredient, the ore is refractory and unsuitable for reduction in the blast furnace. It may be possible by electro-magnetic separation to diminish the percentage of titanium, but this is a matter of experiment, and nothing definite can at present be said about it.

(*b*) Haematite.—This is found on and about the farm Bosmanskop No. 277, twenty-two miles south-east of Middelburg. The deposits are found occurring irregularly in the coal formation, and the total amount of ore available is apparently not large. The haematite is very pure, running about 65% of iron (70% ore is the possible maximum), and contains neither phosphorus, sulphur, nor titanium worth mentioning. It is undoubtedly the best iron ore in the country.

(*c*) Ferruginous quartzite occurs in the Pretoria beds in a well-defined geological horizon. The well-known ironstone bands on the hills immediately south of Pretoria belong to this class and are a prominent feature in the landscape for many miles east and west. Another easily-accessible point where these bands occur is close to Airlie Station, on the Delagoa Bay line. Here there are three well-defined bands, of which the two upper are six feet and eight to twelve feet wide respectively. These bands are continuous over long stretches of country, and would yield enormous quantities of ore. The percentage of iron is generally low, probably between 40 and 45%, but portions of the seams run as high as 55% of iron. Injurious ingredients are practically non-existent in this class of ore.

(*d*) Limonite.—This ore is found in seams in the dolomite, and has received perhaps less attention than the others mentioned. Some of the ore is of fair quality, assaying up to 40% of iron. Owing to the softness of the ore, the deposit is not very prominent, and has thus hitherto escaped much attention, but it might become of importance if found, on prospecting, to exist over large areas. A deposit of this class is found close to Lyttleton Junction, near Pretoria.

(*e*) Chrome Iron Ore. — Although not strictly speaking an iron ore, this mineral should be mentioned here, as it may play a considerable part when an iron industry becomes an established fact. The mineral is found in deposits as large as those of magnetite, that is, in apparently inexhaustible quantities. The most notable occurrences are found along the Pretoria-Rustenburg line, at De Kroon and further to the north-west, and in the Lulu Mountains, north of the Steelpoort Valley, to the Olifants River. The latter deposit seems to be the richer, containing from 40 to 50% of chrome oxide, while the ore on De Kroon assays from 35 to 40%.

Unfortunately, the other two prime requisites—coke for smelting and limestone for a flux—are not so readily available. But the memorandum expresses the hope that further research and experiment may prove the existence of suitable coal and limestone in the Transvaal.—*Transvaal Leader*, July 13, 1909. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

TIMBERING AND MINING: A TREATISE ON PRACTICAL AMERICAN METHODS. By WILLIAM H. STORMS, E.M. Imp. 8vo., pp. viii. and 279, with 125 illustrations. 8s. 6d. (New York, N.Y., U.S.A.: McGraw-Hill Book Company.)

"The title of the book is rather misleading. The actual subject of the author is timbering: the mining contained in the book is merely illustrative of, or explanatory of, the several methods of timbering described. It must also be said that, considered as a monograph on timbering, the treatise is not so comprehensive—much less it is so nearly exhaustive—as one might expect. For example, it contains no reference to the art of timbering and securing a longwall face of coal—a very vexed question at the present day—at any rate, so far as the British aspect of the case is concerned. It must, in justice to the author, be remembered that he professes to expound only practical American methods; and the foregoing criticism is not to be taken as a reflection on the author, but simply as an illustration to our readers elsewhere that they may not find in this volume exactly all that its title might lead them to expect.

Though not by any means exhaustive, the work is not to be described as an elementary treatise, notwithstanding the author's preliminary remark to that effect. He states at the outset that: 'This work is not for the enlightenment of those who are experienced and, therefore, in no need of such a treatise, but rather for those who feel the need of a work which will give them practical ideas of what should be done under given conditions of mining, and how to do it.'

Some of the matters treated in this volume are:—the selection and preservation of timber; the timber-of drifts and shafts in running ground; the framing of shaft and other timbers; timbering stations at shafts; the securing of raises; the square-set system and its modifications; the timbering of large stopes; stoping without timber where timber is scarce; the design of head frames, etc.

The work exhibits a happy blending of theory and practice, the latter largely predominating. There is also an almost entire absence of mathematical treatment, although the fundamental principles are always well in evidence. For example, although the author does not specifically deal with longwall work, he touches the root principle of the timbering methods required even there. In other words, he recognises the fact that in certain ground the pressure is irresistible: that no timber or method of timbering can entirely resist such pressure, and yet he demonstrates that we can safely and successfully timber through the irresistible pressure by frankly bowing to it. He also suggests very clearly (though possibly unconsciously) that 'systematic timbering,' to be efficient, must embrace much more than a mere placing of the props, stulls, bars, etc., at a uniform distance apart."—*London Mining Journal*, Sept. 25, 1909, p. 431. (A. R.)

Bagshaw, Walter. Elementary Photo-micrography 2nd Edition. Cr. 8vo., pp. 104. *Iliffe.* Net 2s.

Caven, R. M. Systematic Qualitative Analysis for Students of Inorganic Chemistry. Cr. 8vo., pp. 250. *Blackie.* Net 3s. 6d.

Davis, A. C. Portland Cement. 2nd Edition. Revised and Enlarged. 8vo., pp. xix.-410. *Woodford Fawcett.* Net 15s.

Easdale, W. C. The Practical Management of Sewerage Disposal Works. A Handbook for those in Charge. 12mo., pp. 56. *Sanitary Pub. Co.*

Fenton, Henry John Horstman. Outlines of Chemistry with Practical Work. First Part. 8vo., pp. 384. *Camb. P.* Net 9s.

Godfrey, Hollis. Elementary Chemistry. Cr. 8vo, pp. 470. *Longmans.* Net 4s. 6d.

Horner, Joseph G. The Principles of Fitting for Engineers, Apprentices, etc. 4th Edition. Revised and Enlarged. Cr. 8vo., pp. 434. *Whittaker.* Net 5s.

Sloane, T. O'Connor. Elementary Electrical Calculations. 8vo., pp. 314. *C. Lockwood.* Net 9s.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 470/09. Boiler Scalers, Limited (1), Frank Gilman (2), George Fredrick Clements (3). Improvements in percussive tools for scaling tubes and for rock boring and the like. 15.10.09.

(P.) 471/09. William McIlwaine Robinson Cusha-Machere. Renewable seating for vacuum cylinder release and other valves. 15.10.09.

(C.) 472/09. Bernard Alexander Firth (1), Herbert Henry Ashdown (2). Improvements in or relating to shoes for mining stamps and the like. 16.10.09.

(P.) 473/09. Robert Airth. Improvements in safety jack bars for mining purposes and the like. 18.10.09.

(P.) 474/09. Alfred Thom Banks (1), Louisa Ann Hyde (2). The dandy machine and hand detachable rock drill head. 18.10.09.

(P.) 475/09. Hans Charles Behr. Improvements relating to pneumatic ore stamps, hammers and the like. 22.10.09.

(P.) 476/09. Hans Charles Behr. Improvements in pneumatic stamps and like percussive apparatus. 22.10.09.

(C.) 479/09 James Irvine MacNair. Improved apparatus principally intended for use in sterilising water. 22.10.09.

(C.) 480/09. Ela Bert White. Improvements in systems of and apparatus for washing and refilling locomotive boilers. 22.10.09.

(P.) 481/09. William McIlwaine Robinson. Binder for attaching telegraph, telephone and other wires to insulators or other supports. 22.10.09.

(C.) 483/09. The New Oil Refining Process, Ltd. (1), James Noad (2). Improvements in the treatment of hydrocarbon oils or the like for the production of volatile spirit. 26.10.09.

Abstracts of Patent Applications.

(P.) 495/08. T. H. Mutch. An improved plant for use in cyanide or like processes of extracting metals from their ores. 18.12.08.

This is an application for an improved plant for use in the cyanide or like processes of extracting metals from their ores. The whole of the plant is built of canvas or other suitable textile material, including the treatment vats, settlers, and extractor boxes. Means are also devised for strengthening the different structures and are as such described in the patent specification.

(C.) 424/09. George Lester Whitcombe. Improvements in or connected with concentrators or apparatus for separating or grading metals, ores and other substances in wet recovery processes. 17.9.09.

The foregoing application describes an apparatus for separating or grading metals and the like, when these are contained in a liquid or pulp. It consists of two vessels concentrically arranged and connected by a helical partition leading from a supply launder and discharging into a horizontally revolvable turbine or blade wheel, the shaft whereof carries a screw conveyor located within the inner vessel and communicating with a discharge launder. The vanes or blades of the turbine travel over an annular space or trough formed in the bottom of the large vessel where the concentrates are collected, and from whence they are led to chambers in a receiving box for further grading of the material. Means are provided for removing a layer of the liquid flowing in the launder from the remaining stream in case of any float material being present, and conducting it to a filtering device where the float material is separated out.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Blaine, G. R., *l/o* Kokstad; Village Deep, Ltd., P. O. Box 1145, Johannesburg.
Clennell, J. E., *l/o* London; Creston Colorado Co., Torres, Sonora, Mexico.
Darling, G. A., *l/o* Johannesburg; New Kleinfontein Co., Ltd., Benoni.
Hallimond, W. T., *l/o* Cleveland; Rose Deep, Ltd., P. O. Box 6, Germiston.
Krause, H. L., *l/o* Maraisburg; Afrikander G. M. Co., Ltd., P. O. Box 121, Klerksdorp.
Leyson, W., *l/o* Florida; P. O. Box 2718, Johannesburg.
Philp, H., *l/o* Lebong Donok, Sumatra; 22, Amess Street, North Carlton, Melbourne, Australia.
Robertson, T., *l/o* Germiston; 37, Brewland Street, Glasgow, Scotland.
Sharpley, H., *l/o* Broken Hill; Ulceby R.S.O., Lincolnshire, England.
Surmon, C. B., *l/o* Shangani; P. O. Box 67, East Rand.
Way, E. J., *l/o* Benoni; P. O. Box 2927, Johannesburg.
Weiskopf, E. H., *l/o* Modderfontein; Japanese Explosives Co., Ltd., Hiratsuka, Japan.
Yates, Arthur, *to* Lebong Soelit, West Coast Sumatra, Dutch East Indies.

Erratum.

October *Journal*, 1909, p. 139, Mr. F. W. Watson's remarks on Mr. L. J. Wilmoth's paper, "Experiments on the Assay of Acid Washes," should read as under instead of as previously printed.

Mr. F. W. Watson (*Member*): Why does Mr. Wilmoth add alkaline sulphide to a strongly acid solution? If H_2S in the nascent state is required, the author should have stated this, since it seems to me that it would be as efficient to pass H_2S slowly through the solution.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. DECEMBER, 1909. No. 6.

Proceedings
at
Ordinary General Meeting,
December 20, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, December 18th, Dr. James Moir (Vice-President) in the chair. There were also present:—

46 Members: Messrs. W. R. Dowling, C. B. Saner, K. L. Graham, E. J. Laschinger, A. Richardson, G. O. Smart, R. Stokes, H. A. White, Prof. J. A. Wilkinson, R. G. Bevington, W. A. Caldecott, Andrew F. Crosse, H. A. Adams, W. Beaver, F. W. Bentley, W. Broom, E. Browne, W. M. Coulter, M. J. Doyle, G. G. Ferris, A. D. Gilmore, B. J. Hastings, Dr. J. McC. Henderson, A. J. Herald, C. B. Hilliard, J. H. Johnson, T. Johnson, G. A. Lawson, J. Lea, R. Lindsay, W. D'A. Lloyd, R. MacGregor, M. T. Murray, S. Newton, F. D. Phillips, J. F. Pyles, E. T. Rand, D. G. F. Ross, O. D. Ross, A. Salkinson, A. H. Scarf, A. L. Spoor, S. H. Steels, W. A. C. Tayler, and J. F. Walker.

11 Associates and Students: Messrs. E. Brennan, J. Cronin, W. J. R. Hunter, A. King, C. J. McCaffrey, R. Murdoch, J. M. Robinson, J. Sharp, H. Stadler, W. E. Thorpe, and W. Waters.

9 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the November *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. A. Caldecott and A. F. Crosse were appointed scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been unanimously elected, as follows:—

Barry, Richard Alan, Nourse Mines, Ltd., P. O. 32, Denver. Mining Engineer.
Campbell, John McPherson, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Mill Foreman.
Crichton, Charles, Afrikander G. M. Co., Ltd., P. O. Box 121, Klerksdorp. Analytical Chemist.
Dodd, Michael, P. O. Box 1197, Johannesburg. Mining Engineer.
Sharwood, William John, Ph.D., A.R.S.M., Homestake Mining Co., Lead, S. Dakota, U.S.A. Metallurgical Chemist. *(Transfer from Associate Roll.)*
Shepherd, Simpson, Village Deep, Ltd., P. O. Box 1145, Johannesburg. Surveyor.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Moffitt, Charles Edward, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Amalgamator.
Sharp, J. E., Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Foreman.
Strange, Ernest Harold, 156, Guilford St., St. John (West), New Brunswick, Canada. Miner.
Whaley, John, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

As Student.—

Vogts, William Thomas Laing, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Learner.

General Business.

METALLURGICAL AND MINING TERMS.

The Chairman: I may mention that at the last meeting of the Council, Mr. H. A. White raised the question of what may be called a standard spelling to be used in the *Journal*. He considers that a number of terms now used in mining in the plural ought to be used in the singular, such as "sands," "concentrates," "tailings," etc. It caused an animated discussion in the Council, and a sub-committee was appointed to go into the question. What we should like now is suggestions from members as to what terms should be revised in this sense, and, generally speaking, what the members think of the revision.

Mr. H. A. White (*Member of Council*): This sub-committee has been appointed to consider the possibility of improving the nomenclature of mining and metallurgical terms. For instance, it is considered that it would be better to use such terms as "slime," "sand," and "residue" in the singular where only one such product is referred to. Again, such words as "mullock" and "dirt" which are workmen's terms, might be avoided in scientific literature.

Mr. A. F. Crosse (*Past-President*): I have much pleasure in supporting the suggestion. It is quite right that we should use the singular in using the word "slime" when we are talking about the slime of one mine, but when we are talking about the slimes of the Witwatersrand, for example, we should use the word in the plural.

The Chairman: I hope you will send in your suggestions to the Secretary soon so that the sub-committee may be able to deal with them as speedily as possible.

Mr. A. Richardson (*Member of Council*) read the following on behalf of the author.

ENDLESS ROPE HAULAGE.

By H. G. Kay (Associate).

A very considerable sum of money is expended annually by mining companies in the transport of rock to the reducing plants and waste and tailings to the dumps. The most common form of transport for this purpose is by means of endless rope haulages, and it is proposed in this paper to give a few notes on the system of hauling as practiced on diamond mining properties, where large quantities of ground are handled in open cast workings.

The principle of the system consists of hauling by means of an endless rope to which the trucks can be attached or detached at certain fixed points without the necessity of stopping the rope, thus keeping up a constant supply of trucks to the working places. The power to work the haulage can be transmitted by a stationary engine, electric motor, or by an ordinary clutch and drum arrangement attached to a moving shaft, or a sub-haulage can be driven off a main haulage rope by means of a shaft carrying a double drum with clutch attached.

The speed of the haulage depends largely on the state of the running track; a well laid line with properly constructed onsetting point can be worked quite satisfactorily up to a speed of 5 miles an hour, and at that rate 1,000 to 1,200 trucks per hour can be handled. With a rough track, as is usually laid on stone tips and tailings dumps, the speed should be reduced to about 2 miles per hour, at which rate trucks can be handled with ease and safety.

Where curve wheels are used to work haulages around angles a high rate of speed is not desirable, as even with the best constructed track there is always a great deal of wear and tear on both the truck and the rails in negotiating the small curves which the radius of the curve wheels necessitates. The amount of deflection off the straight for a curve wheel should be between 15° and 30 ; if the angle at the intersection is too flat the rope is likely to leave the wheel and, on the other hand, should the angle be too acute the curve, which by necessity has usually about a 3 ft. radius, is too sharp for the truck to work around with any degree of safety: as a support to the wire, a roller fitted to a swinging arm should be placed on the on-coming side of the wheel, and, as a further precaution against the rope leaving the wheel, the latter should be slightly elevated on the off-going side of the rope.

In running inclines a grade of 1 in 8 (12½%) is about the limit for a fast travelling haulage, although at a lower speed inclines of 1 in 6 can be worked: the grade to a large extent is limited by the speed of the haulage, as with a fast rope and a steep incline the trucks are likely to up-end as soon as hooked, and also at the starting and stopping of the rope. The usual method of attaching a truck to the rope of an endless haulage is by means of a "jockey," a **V**-shaped iron fork fixed to the top of the truck body, into which the rope is forced until the necessary grip is obtained. The form of "jockey" largely used on the Rand in earlier days was bent out of ½ in. or ¾ in. round iron into the form of a **V** with a long arm projecting downwards from one corner to form a means of attachment to the truck. An improvement on the above is the wrought iron fan-tail jockey, made in the shape of a **U** with the opening slightly tapering towards the bottom, the jaws are also broadened at the top so that when the truck receives the pull of the rope and the jockey turns slightly in its bracket the slot holding the rope is partly closed up at the top owing to the overlie of the fan shaped jaws, thus locking in the rope and preventing it, to a large extent, from rising and pulling out of the jockey.

A very successful jockey (Pack's Patent) in use at the Premier Diamond Mine is constructed with renewable jaws. Fitted to the top of the pillar is an oblong cast iron box into which the two jaws are set, these being held in position by means of a wedge driven down between them and through the bottom of the box, this wedge being also held fast by a small wedge and split pin. With this arrangement each jaw can be given a quarter turn, thus allowing four wearing faces before new jaws are required.

In laying a haulage track the rail on the side to which the jockey points must be slightly elevated so that the rope is brought directly over the centre line between the two rails, and to prevent unnecessary wear and tear on the rope, rollers or hard wood rubbing blocks are placed at intervals between the rails to carry the sagging rope.

The most important item in the haulage is a properly constructed onsetting point. The grade feeding the trucks to the onsetter should be such, that when the truck reaches the rope for hooking the speed of the rope and truck are exactly the same: should the speed of the truck be either greater or less than the speed of the rope a sudden jerk will take place on hooking, which is very detrimental to the life of the rope.

At a fast feeding onset, brakes to regulate the run in of the trucks should be used, also a contrivance to turn the jockeys automatically into the right position for hooking, and as a further safeguard against the trucks leaving the rails at the point of hooking, check rails or boxing should be placed for a short distance on each side of the onsetting point. The catenary curve connecting the down grade to the onset with the up grade of the incline requires special attention in laying out, as upon this the rate of hooking largely depends; the rails must follow a line which is parallel to the natural curve taken up by the rope, and to maintain the rope in this position a balance weight or tension is employed. This frame also takes up the amount of expansion or contraction in the rope due to stretch or change of temperature; a well balanced and sensitive rope is absolutely essential to a fast feeding haulage.

At the point of disengaging the trucks an "apex" is formed with a down grade sufficiently steep to cause the truck to overrun the rope, this change from a pull to a push on the rope causes the jockey to come square with the rope and easily free itself. On a long steep incline, or in fact on any haulage which is at all permanent, disengaging frames shou'd be used and "slog-out" boys dispensed with. The frames are made in various patterns, a cheap and effective one is in the form of a double cross frame carrying two long timber runners (see Fig. I.) placed parallel to the track and just sufficiently high to clear the top sides of the truck body; fixed to the top side of these runners and at the off-going end is a bar with a roller carrying the rope; as the truck reaches the frame the body is slightly lifted until it comes into contact with the runners when the rope is pulled out of the jockey and the truck released automatically from the rope.

Another arrangement for a disengaging frame consists of a double frame carrying two lengths of channel iron set parallel to the rope and just sufficiently wide apart to allow it to pass freely between them and up over the roller at the off-going end; as the truck reaches the frame the rope passes up through the slot formed by the channel irons, and as the opening is wide enough to admit the rope only the jockey is pulled off the rope and the truck disengaged.

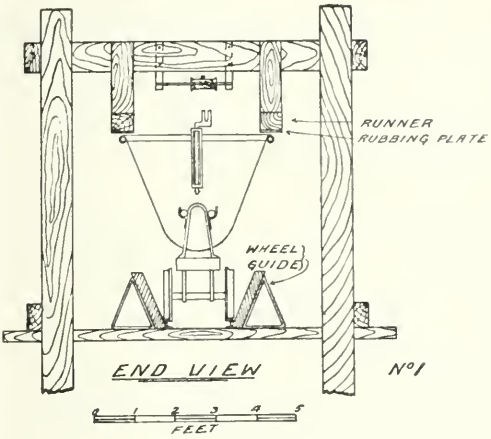


Fig. I.

A form of haulage very suitable for tailings dumps consists in having the jockey fitted to the side of the truck carriage instead of on to the body as is the usual practice: with this arrangement the truck can be tipped without being released from the rope, and consequently there is no time or labour lost in disengaging or rehooking. A haulage of this description is very economical as from the time the truck is filled at the bin and hooked to the rope there is no need to handle it again until it is returned to the filling bin, the tipping, righting, and disengaging all being done automatically. With this arrangement trucks will negotiate curves of 12 ft. radius or over by a series of rollers set about 2 ft. 6 in. apart and just on a level with the rope: these curve frames (see Fig. II.) are held in position by ballasting them down with sand bags or stones and can be easily shifted bodily forward as the dump extends.

A serviceable truck tipper (see Fig. III.) consists of a frame carrying a length of rail or pipe set at a height to just catch the top of the truck body, this tipping bar is placed diagonally across the track with the on-coming end about 1 ft. outside the rails and the off-going end over the centre line of the track; as the truck reaches the tipping bar the body runs along it until it is overbalanced and tipped, a check bar is placed over the truck wheels to prevent the carriage capsizing also. The same principle is applied to the "truck righter," a length of rail or pipe is bent into the form of an incline plane and set alongside the track so that the lower end of the incline will just catch under the down turned lip

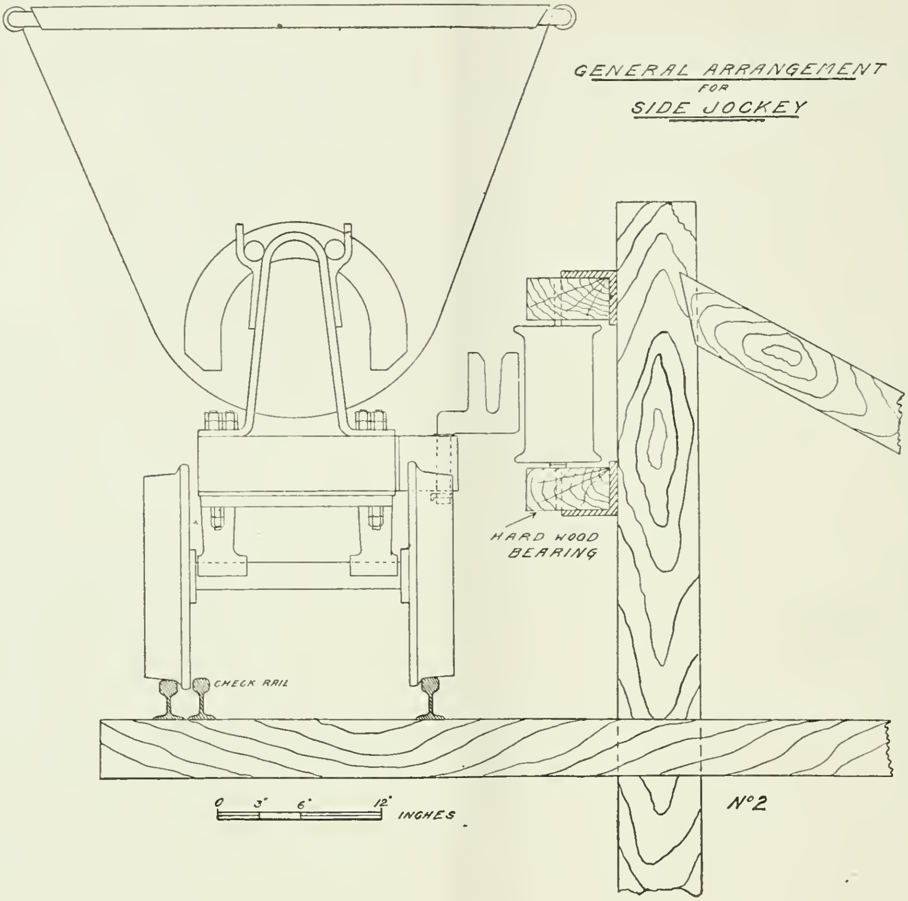


Fig. II.

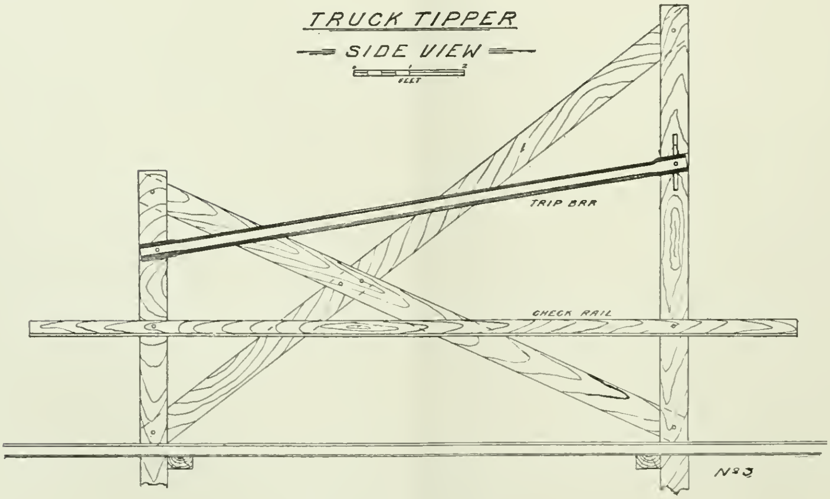


Fig. III.—Side View.

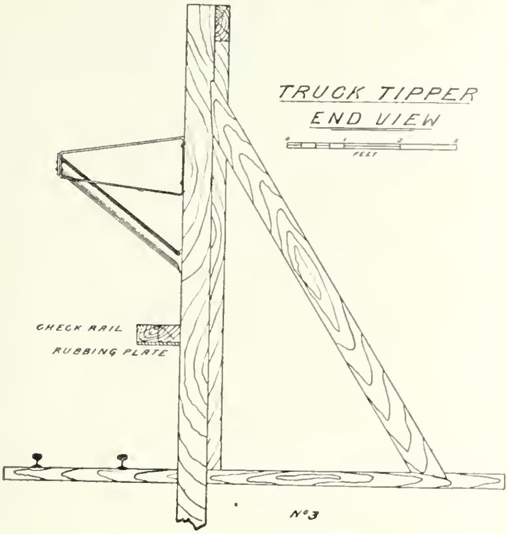


Fig. III.—End View.

of the truck body, and as the truck is drawn along the incline the body is raised until it regains its upright position (Fig. IV.).

At the Premier Diamond Mine endless rope haulages are used throughout the workings; the main haulage to No. III. gear has three changes of grade, the steepest being 1 in 8, or 12½%, the tracks are laid with 40 lb. rails set on wooden sleepers, width between up and down tracks 6 ft. The rope is of plough steel 1¼ in. diameter, travelling about 4½ miles per hour. The rate of hooking on this haulage is from 1,000 to 1,200 loads per hour, although over 26,000 loads have been hauled in a working day of 21 hours, and over 140,000 loads in the week. The length on incline from onset to apex is 3,123 ft., and the vertical lift 239 ft.

The new incline to No. 4 gear on the same mine has two changes of grade 1 in 8·9 (11·2%), and 1 in 8 (12½%) respectively, the tracks are also laid with 40 lb. rails, 8 ft. centres between tracks, the rope is 1¼ in. diameter plough steel, and travels at about the same rate and has the same capacity as the previous haulage referred to. The length of incline is 2,425 ft., and the vertical lift 270 ft.

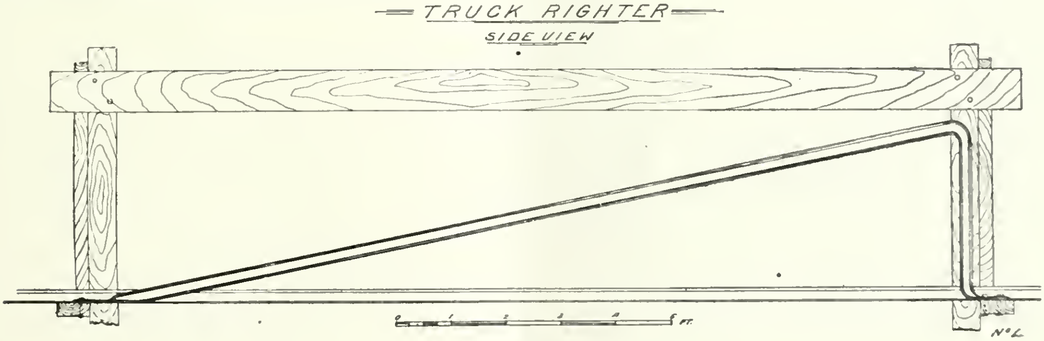


Fig. IV.

The Chairman: I think we owe a very hearty vote of thanks to Mr. Kay for his interesting paper. It is entirely out of my line, so that I do not propose to comment on it. The only thing that strikes me at present is that a paper of this sort seems to go far towards solving the native labour question, by making things more automatic than they are at present.

Mr. R. G. Bevington (*Past-President*): I should like to second the vote of thanks to Mr. Kay. On the occasion of our Society's visit to the Premier Diamond Mine, this was one of the devices that struck me and interested me very much, and it is certainly of advantage to have a description of it on record in our *Journal*. I was interested in watching the way the trucks were hauled along, automatically tipped and automatically righted.

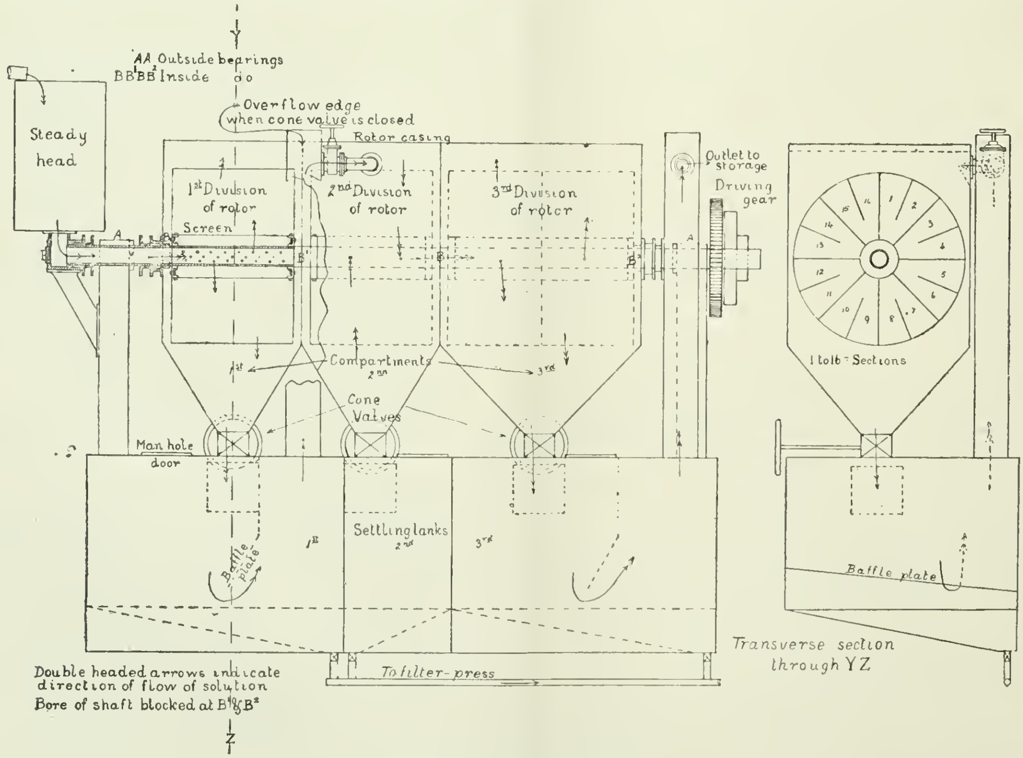
Mr. E. T. Rand (*Member*) read the following paper:—

A ROTARY EXTRACTOR FOR PRECIOUS METALS FROM SOLUTIONS.

By W. D'Arcy Lloyd and Ernest T. Rand (Members).

From time to time attempts have been made to precipitate gold and silver from their cyanide solutions whilst the precipitating agent was in motion, with more or less success. For example, zinc shavings have frequently been subjected to mechanical motion whilst in contact with the gold and silver cyanide solutions, and in most cases as far as actual precipitation of the precious metals was concerned, some success has been achieved, but mechanical difficulties have developed due to the method of applying motion by attrition, or an insufficient contact of the causing the breaking up prematurely of the zinc solution with the zinc. The main idea underlying all the attempts to perform the operation of precipitation with the precipitating agent in motion, is to avoid the fouling of the zinc, which is so painfully evident in the first compartments of all our extractor boxes after a three days' run from time of dressing. It is well known that after thorough washing and redressing, the zinc will go on precipitating as briskly as ever until fouling starts again. The natural inference from this is that if the zinc could be kept continuously in motion, thus preventing fouling, the highest efficiency would be maintained. Having this in view, we have evolved the machine of which we endeavour to give some idea.

The precipitating agent used was zinc in the form of shavings and clippings, the latter being cut from sheet zinc and approximately $\frac{1}{4}$ in. to $\frac{5}{8}$ in. square The machine is a rotating one, with a water tight casing of comparatively light steel plate, the casing being fitted with plates to divide it into compartments. In the machine with which the experiments were carried out, there were four of these compartments, but we have reason to think that the result required would be attained with three, the third being one-third larger than the first and second (see sketch). In the centre of these division plates are securely fixed bearings, in which a hollow shaft rotates. Two circular discs in each compartment distant from each other almost the full length of the compartment, are securely bolted to flanged collars, which in turn are fixed to the shaft. The shaft with these discs and other attachments form the rotor. The rotor, besides being divided into compartments in length, is subdivided radially, in appearance much as a transverse section of an orange. There are sixteen radial divisions, which number was arrived at experimentally, the pressure of solution during rotation of a smaller number tending to press the zinc against the wall of the divisions leaving a space between the zinc and the next wall. The use of the larger number of radial divisions obviated this tendency. These radial divisions are removable, and are lightly and strongly constructed of perforated ($\frac{5}{16}$ in. holes) steel plate. Between and attached to the discs a suitable framework is fixed to contain the sections. There are many simple methods of retaining the sections when in position. Each section has a door of the same material as that of which the section is constructed, so that each is independent of its neighbour. Around the shaft, and fixed securely



THE ROTARY EXTRACTOR.

to the flanged collars to which the rotor discs are fixed, is a strong screen of $\frac{1}{8}$ in. mesh leaving a space of about 2 in. between itself and the shaft, to prevent any fine material entering the shaft. The shaft, as before stated is hollow, the bore being of ample size to allow of the free flow of the solution. The parts of the shaft enclosed by the flanged collars are perforated by inch holes to allow of the free egress and ingress of solution. At the required position the bore of the shaft is tightly closed to regulate the flow of the solution to wherever it is required. At the inlet for the solution end of the rotor is a small tank to act as a steady head, the top of this tank extending about 18 in. above the top of the rotor casing. At the outlet end of the casing, the pipe for carrying away the effluent is so placed that a constant level of the solution is maintained within the casing, which insures that the rotor will be completely submerged. The bottom of each of the casing compartments is made in the form of a cone, and to this is fixed a valve which connects the cone with a settling tank, whose total capacity is equal to one and a half times that of the casing; this is divided into compartments to correspond with the number of those in the casing. All the solution passes from the casing into the settling tank from both the first and third compartments. The solution passes from the second compartment into the third compartment of the casing along the hollow shaft. The compartments in the settling tanks have a sloping bottom to one point in each, at which point a valve and pipe connection is made communicating with one main pipe which leads to a filter press. To the first compartment of the settling tank a stand pipe of ample capacity is fixed, and stands outside and close to the rotor casing and projects a few inches above it. A pipe from this stand pipe delivers into the second compartment of the rotor casing. A similar stand pipe is supplied to the third compartment of the settling tank, and the outlet pipe from this delivers to the sump, the position of this pipe determining the level of the solution in the rotor casing, as before mentioned. In the case of the first and third compartments of the settling tank a baffle plate is placed between the inlet of the rotor and casing and the exit to the stand pipe. Each compartment is supplied with a manhole door. A screen basket is placed underneath each inlet to the tank to catch any fine zinc that may have passed through the rotor casing. A suitable gearing for rotating the machine is fixed to an extension of the shaft at the exit end of the machine and about 1 h.p. is absorbed in a machine precipitating 250 tons a day.

The operation of the machine is as follows:—The solution enters the steady head, passes into the hollow shaft, and as the bore is blocked at the position corresponding to the end of the first division of the rotor, it passes out through the perforations, coming in contact with the zinc and leaving the rotor peripherally. The solution next passes into the settlement tank, and beneath the baffle plate, rising to the constant level in the stand pipe to the second division. From here it again enters the perforated shaft passing inwards through the zinc, along the shaft, and outwards through the zinc in the third section, again downwards, through the settlement tank (where the speed of the flow is greatly lessened by the size of this receptacle) beneath the baffle plate and up the stand pipe thence to the storage sumps.

By the arrangement of settling tank it will be seen that there is but small chance of slimes being carried over to the other division. The rotor revolves from one to one and a half times per minute, and this is fast enough to allow of a slight motion of the zinc in their sections which is as much as is desired.

All gold slime loosened from the zinc passes to the tank beneath: this can be removed at any time without stopping the machine by closing the valve connecting the rotor casing and the settlement tank, and allowing the solution to overflow the division plate forming the first and second compartments. The contents can then be pumped into a filter press, the solution from this being returned to any rotor desired. By means of the manhole before mentioned the inside of the tank can be washed out, this closed and valve slightly opened until settling tank is full of solution, then the valve opened full.

With reference to zinc consumption, it may be stated that the only zinc taken to acid was that which passed through the rotor in the form of shorts during its operation.

The following is a summary of the results obtained over a period of from November 4, 1908, to May 26, 1909.

Runs were made with strong, weak and slime solutions respectively, the results obtained being as follows:—

	Tons of Solution per cubic foot of zinc per 24 hours.	Value of Solution Entering. dwts.	Value of Solution Leaving. dwts.	Percentage of Extraction. Per cent.
Strong	1·01	2·221	0·053	97·6
Weak	1·84	0·383	0·029	92·3
Slimes	1·53	0·460	0·031	93·2

The above figures represent the mean of assay results over the individual runs.

At the clean-up the solution is shut off, the belt on step cone shifted to faster speed and rotor run on this for a few minutes, then stopped. (We might here mention that the sections on our experimental machine were fixtures, also there was no settlement tank below). The sections on the top side were opened up, the zinc taken out and rinsed though it was found that there was but a small quantity of loose slimes entangled. The zinc was returned and the section filled up with zinc from the second division. We found that finely cut filiform zinc broke up into shorts when using strong solution so that coarsely cut zinc would be preferable.

When coarse zinc or clippings are used, the adhering gold may be removed in an acid bath as it was found in the case of clippings that 94% of the gold was removed in a 5% acid bath in 15 minutes, any deteriorating effect on the clippings being negligible. After the acid bath the zinc is thoroughly washed in clean water, the sections opened and examined, and fresh zinc added where necessary; it is found that the zinc in this form can be used in the rotor for a long time with a minimum consumption of both zinc and acid. To remove the gold from the zinc the sections are placed in a drum which is revolved in the acid bath. We experimented for three weeks in our rotor with clippings and found the precipitation as good as under the conditions of the same space packed with filiform zinc (finely cut), the weight of clippings being 11 lb. to 1 lb. of shavings.

Our experimental machine was somewhat crude, and devoid of perhaps the greatest mechanical essential to its success, *i.e.*, the settling tanks, the solution passing directly from one division to another without separation of precipitated slimes. The advantages of the rotor may be summed up as follows:—(1) Ready access by authorised persons to the gold slimes, without stoppage of precipitation, enabling a daily clean-up to be made if required. (2) With clippings and coarsely cut filiform zinc, absence of breaking down and readiness of removal of adhering gold in an acid bath without consumption of all the zinc, with a consequent saving of zinc and acid. (3) Clean-up quick and cleanly with a consequent advantage of a minimum of loss. (4) Shortness of time rotor is out of action during dressing. (5) No fouling of zinc. (6) While in operation no danger of theft of gold slimes, as all valves would be locked.

Before closing, we must record our thanks to the Management and to the Consulting Metallurgist, Mr. E. H. Johnson, of the East Rand Proprietary Mines, for the interest shown, and encouragement and assistance rendered while conducting our experiments.

The Chairman: I see you appreciate the interest of this paper. There is no doubt there is some scope on the Rand for a process of this sort. On theoretical grounds, any rotary process will expose a larger surface of zinc, but, of course, it remains to be seen whether any particular device is successful commercially. I would like to know whether any gentleman has had any experience of it, and in the meantime I would like to move a hearty vote of thanks to the authors for their interesting paper.

Mr. A. F. Crosse (*Past-President*): I think there is a great deal in this paper. The authors claim several advantages for their method but they have left out one most important advantage, namely, the mechanical driving off of the hydrogen bubbles from the zinc; this removal of hydrogen helps precipitation.

Mr. E. T. Rand (*Member*): In connection with that I may say that the bubbles of hydrogen on the surface of the solution are very evident indeed. There is a very violent action going on continually.

Mr. G. O. Smart (*Member of Council*): I have pleasure in seconding the vote of thanks to Messrs. Lloyd and Rand for their paper. They have certainly shown considerable ingenuity in the invention of their plant. From the reading of their paper, I have not been able to follow the whole process, but it appears to me that the clean-up will be more under the control of the responsible man, and further, the collection of the gold bearing slime ought to be a much cleaner operation, than obtains with our present plants.

NOTES ON CORROSION, WITH SPECIAL REFERENCE TO THE CORROSION OF STEEL WINDING ROPES.

(Read at August Meeting, 1909.)

By M. Thornton Murray, M.Sc.

DISCUSSION.

Prof. J. A. Wilkinson (*Member of Council*): Before the author gives his reply to the criticism on his paper, I should like, with your permission, to make a few remarks. The subject of this paper interests me very much, and I had hoped to have been able to have done some experiments on this subject. Just, however, as the vacation was beginning, and hence my opportunity, a sudden thirst for knowledge on the part of the mine employes has deprived me utterly of the time required, as many of those present will be fully able to realise. The author dealt in his contribution with the effects of corrosion as shown by micro-photographs of some typical specimens. He did not consider the power of resistance to corrosion of new wires, a phase of the subject which, in my opinion, is well worth investigation and can easily be carried out. The methods of attacking this problem is too obvious to require explanation here. Further, the question of the action of lubricants was merely mentioned, and here again I am of the opinion, that there is a wide field for investigation on definite lines. I hope, however, that this is not the last which we shall hear on the subject, and I am sorry to see that the subject has not met with that discussion which its importance to the mining industry warrants.

Mr. Andrew F. Crosse (*Past President*): As Professor Wilkinson has treated this paper in a scientific way may I ask him if the University College staff could not make some experiments on the state of iron or steel wire containing occluded hydrogen. It has probably a very deleterious effect on the tensile strength of the wire. I believe that it has been proved that hydrogen in this state has a specific gravity eight times as high as it should have by calculation. This figure was obtained in using palladium. Perhaps Professor Wilkinson will make some experiments and publish the results.

REPLY TO DISCUSSION.

Mr. M. Thornton Murray (*Member*), in reply, said: I regret that there has been so little discussion on my paper. I had hoped that some of the members would have brought forward the results of their practical experience, for it is only by practical experience of such questions as corrosion that we can really test our theories: laboratory experiments can never quite reproduce working conditions.

With regard to Mr. Crosse's suggestion, I think it is an excellent one. The phenomenon he refers to is not well understood, and I will certainly endeavour to carry out the experiments he mentions, when I have time.

I must thank Prof. Wilkinson for going into further detail. At the same time I have to take exception to one or two of his remarks. I think he has forgotten the title of my paper. Now, I expressly entitled it: "Notes on Corrosion," intending to deal more with the *practical* than the theoretical side of the question. On the other hand, Prof. Wilkinson has justly drawn my attention to the fact that I omitted giving credit to Dr. Whitney. Dr. Whitney has certainly done much good work on the subject, but it was Dr. Cushman who first demonstrated Whitney's theories in so excellent a manner, that his name has eclipsed that of Whitney as pioneer of the electrolytic theory. If Prof. Wilkinson will refer to the *Locomotive Engineer* of (I believe) August, 1908, he will find a summary of a paper which I had the honour of reading before my own University Metallurgical Society, wherein full credit is accorded to Dr. Whitney.

Prof. Wilkinson observes that he would prefer "the CO_2 theory," as a name for the original theory of corrosion. I am surprised that Prof. Wilkinson should make use of such a phrase; it is scarcely academic. "The CO_2 theory" would never be used by chemists; "the carbon dioxide theory" might. Then Prof. Wilkinson has drawn attention to the matter of lubrication, and the effect of the dissociation of oils in promoting corrosion. I think such a phenomenon (if it exists) is scarcely of practical importance, because, after all, we find that lubricants retard corrosion in almost every case, and the case of wire haulage ropes is not so very different from other cases of lubricated surfaces.

I certainly regret that Prof. Wilkinson has not been able, owing to the claims of the "Reef," to carry out the experiments he outlined, so that he might have placed the benefit of his very great chemical knowledge at our disposal.

The Chairman: I think I must congratulate the author on his spirited reply. One point I would like to refer to, is the protective action of alkali on driving wires. I think if Mr. Murray has any spare time he might look into that question a little more fully. Theoretically, I suppose the one which is most alkaline, namely, caustic potash, ought to be the best, but there are obvious difficulties in using it.

Prof. J. A. Wilkinson (*Member of Council*) read the following reply:—

PRECIPITATION FROM CYANIDE SOLUTIONS BY ZINC SHAVINGS AND DUST: A COMPARISON OF RESULTS AND COSTS.

(*Read at January Meeting, 1909.*)

By Allen J. Clark (Associate)

REPLY TO DISCUSSION.

Mr. Allen J. Clark (*Associate*): In replying to the discussion of this paper, I find it necessary to refer also to Mr. Linton's paper in the *Journal* for August, 1909, in which he replies to the discussion of his earlier paper. Indeed, with the exception of this article, there is little to which I can reply, as my paper met many of the points raised by Mr. Pearce.

If I have read Mr. Pearce correctly, and have expressed myself intelligibly, then Mr. Linton has misinterpreted us both in his attempt to compare our results. My statement had reference to but one type of solution, and that (viz., the first and last drainings, or "low solution") of minor value and importance. Mr. Pearce stated that no figures were available for zinc consumption for slime solution—the solution most nearly resembl ing that with which the Homestake experiments were conducted—and gave figures for the whole plant.

I had aimed to confine myself to the discussion of the relative efficiency of zinc shavings and zinc dust upon identical solutions, and sought to avoid any generalisations therefrom, other than as to the Homestake plant, inasmuch as the attempt to compare results achieved at different plants, precipitating, probably by different methods, solutions of widely varying composition, has always seemed to me to be without profit ; and while Mr. Linton's later figures only confirm me in this belief, I feel it only fair, in view of his statement, to say, following literally Mr. Pearce's form, that at the Homestake, for the eleven month period ending November 30, 1908, "taking it over the whole plant, it works out, on an average, to ·096 lb. zinc dust per ton treated, irrespective of the grade of the material treated. If expressed in relation to the gold recovered it would be 2·13 lb. zinc dust per ounce recovered," or, if in terms of the gold and silver, it would be 1·55 lb. zinc dust per ounce Au + Ag recovered. This from tailing assaying $1.20 per ton.

With Mr. Linton's review of Mr. Pearce's experiments I am in general accord. To his suggestions as to the factors influencing the results, I would add these : alkalinity of solutions, rate of flow, continuity of flow, and design of press—the latter inasmuch as the Merrill triangular press marks a great forward step in the development of the zinc dust process. I would also suggest that Mr. Pearce might have done better had he reduced the air agitation to a minimum, or even dispensed with it entirely, and given all possible attention to preventing the oxidation of the zinc dust precipitate.

Since reading this article, however, I have made a few brief runs at strengths lower than ·02% KCN, and these have indicated that, though when the average strength is ·02% some few hundred tons of a strength of ·015% or even ·01% may be successfully precipitated, the precipitation of solutions of a lower average strength than ·017% would be difficult, and of an average strength of ·01% well nigh impossible. Our present practice has brought us a long way below Mr. Pearce's minimum strength of ·05%, but it has not brought us all the way necessary to meet his requirements.

The influence upon the zinc consumption of the proportions of gold and silver and of the strength and grade of the solutions to be precipitated, is of course great, whatever form of zinc may be employed. It may be interesting to elaborate Mr. Linton's table, noting that all these results are from zinc dust.

Oz. Au + Ag per ton Solution.	Ratio Au : Ag.	Pounds zinc dust consumed per oz. Au + Ag, Precipitated.	Remarks.
·02	2 : 2	6·60	Homestake 'low solution'
·15	2·2 : 1	·91	,, 'weak solution'
·47	1 : 4	·59	Cerro Prieto.
·49	1 : 4	·57	,,
·70	1 : 4	·42	,,
1·84	1 : 19	·19	Montana, W. J. Sharwood
3·29	1 : 99	·16	An American mill.

As to the cost of refining precipitates the same difficulty of comparison exists ; at the Homestake we have never found trouble in acid treating the zinc dust precipitates since we learned the lesson of protecting the precipitate from oxidation. Such plants as have had this difficulty, so far as I am aware, owe it to insufficient attention to this point.

I submit to Mr. Linton that as between the processes the difference in bulk of precipitate can by no means be gauged by comparison of the zinc consumed. Again to elaborate his figures, we have

Plant	Method	Zinc consumption, lbs. per oz Au + Ag, precipitated.
Cerro Prieto	Dust	·4
Liberty Bell	Shavings	·29
Montana	Dust	·18
Homestake	Dust	1·55 (or 2·13 gold only.)
Rand	Shavings	— (1 to 3, gold.)

From these premises, without supplementary data, it would be difficult to arrive at any conclusion ; indeed, "that way madness lies."

It may be interesting to note that zinc dust precipitates have carried up to 65% of precious metal, and that the Merrill presses at the Homestake have yielded a product over 42,000 times as rich as the solution from which it was derived.

Mr. Pearce has made clear to us several features of the African practice. I would like to ask him for light upon one more point. He states that the amount of zinc used over "the whole plant" amounts to from 1 to 3 pounds per ounce of gold recovered. Are these the figures for different months at the same plant, or do they represent the extremes found at different plants in the district? And if the former, what causes can lead to such a great variation in the efficiency of the zinc?

The Chairman : We are bound to thank the author for his very interesting paper, and for his still more interesting reply.

CONTRIBUTED.

Mr. S. H. Pearce (*Past-President*) : In reply to the above, I have to state that the variation in the zinc consumption here, from 1 to 3 lb. per oz.

recovered, represents extremes on the field, and not month to month results on the same plant. The chief variation in the consumption as expressed in lb. per oz. of gold recovered, is caused by the value of the solution precipitated, *i.e.*, rich solutions will show a low consumption of zinc, and *vice versa*. The consumption of zinc per ton of ore treated will vary according to the amount of solution precipitated, and this again will vary according to the strength of solution employed.

Another factor of some importance is the proportion of zinc removed from the extractor boxes at the clean-up.

As, however, I do not propose to deal with the matter at any length, I might answer Mr. Clark's query briefly as follows :—

1-lb. zinc consumed per oz. of gold recovered represents the best work on rich material, such as 5 dwt. mill pulp and over.

2-lb. consumption represents economic work on 2·5 dwt. mill pulp.

3-lb. consumption will probably cover the average of the least economic work on the same value or lower, and is generally due to a closer clean-up than is necessary to preserve effective precipitation.

It will probably interest Mr. Clark also to hear that we intend to make another attempt to use zinc dust, and that we hope to be able to chronicle more encouraging results than were obtained on the former occasion.

Prof. J. A. Wilkinson (*Member of Council*) read the following reply :—

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

(*Read at May Meeting, 1909.*)

By the PORTLAND METALLURGICAL SOCIETY (edited by THOS. B. CROWE).

REPLY TO DISCUSSION.

Mr. Thos. B. Crowe: In answer to Dr. Moir's criticism upon my article, appearing in your July issue, I am afraid that Dr. Moir has drawn conclusions which are incorrect.

Possibly he is right as to ammonium persulphate not giving off SO_2, however, we are thoroughly acquainted with the fact that our leading authorities differ upon the subject.

As for the liberation of I from KI in an alkaline solution, I ask him to try a simple experiment before making further statements. Make a NaOH solution of one to two-tenths per cent. strength, or about the same strength in alkali as an ordinary working cyanide solution, add a little KI and persulphate and then test for I by any method preferred. (I think that he will find it possible to collect the I with CS_2.) After being satisfied as to the existence of free I, take some of the solution and be satisfied as to its alkaline condition.

We were aware that it was a well-known fact that the halogen salts of cyanide could not resist alkali and it was the following out of this principle (keeping the protective alkalinity at the lowest possible point), that we were able to obtain good results.

As for the replacement of KI by NaCl, the fact that the Na and Cl of NaCl have a stronger affinity for each other than the K and I of KI, makes it impossible to liberate the Cl under the conditions demanded in this solution. Again, if it were possible to liberate Cl from NaCl under these conditions with the same ease as I is liberated from KI, I doubt very much that we would be able to form ClCN, as we have experienced considerable difficulty in making this compound. We have repeatedly tried replacing the KI with KBr as well as with NaCl but with no success.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution :—

A SKETCH OF THE SMALL CYANIDE PLANT AS ERECTED AND WORKED IN RHODESIA.

(*Read at September Meeting, 1909.*)

By F. J. THOMAS (Member).

DISCUSSION.

Mr. R. Murdoch (*Associate*): Having had a few years' experience of Rhodesian cyanide plants, mostly small and of corrugated iron, I venture to supplement Mr. Thomas' paper with a few observations.

For sand plants I have invariably used 24 gauge iron for the sides with 22 gauge iron for the bottoms, on tanks up to 18 ft. diameter and 4 ft. 6 in. sides. Joints must be perfectly true, all other details being as specified by the author. As soon as a tank is in position it is charged and solution pumped on till saturated and full to the brim. After 12 hours' standing leaks are carefully looked for, and if absent, residue sands are allowed to pack round the outside until level with the top. This ensures uniform support, protects the tank and adds to its life. As far as the stability of good corrugated iron tanks go, I might here say that I have run two slimes plants in this country built of that material. Dissolving vats had the usual paddle stirring gear, and were built of 22 gauge iron throughout, the dimensions in one case being 16 ft. diameter by 8 ft. deep.

Small units are obviously the rule on such plants, being readily moved from one place to another, while the shallow depth of sands, and the handling they get tend to an ideal extraction. Even slimy charges from quartz ore (that is, containing 30 to 40% of very fine sand, practically slime), well dug over daily and well drained will give an 85% extraction within four days on a solution ratio of 1 : 1 or 1·5 : 1 put on with a hand pump. I think the average extraction on these plants is high, and I have checked the working of a few run mostly by men whose knowledge of cyaniding was of the poorest. On one occasion I was called upon to locate a constant discrepancy between theoretical and actual extraction, and discovered that the assayer was using grain assay-ton weights for his samples and ½ mgm. riders on a decimal beam for his prills.

Furnaces are certainly primitive, but with a little care can be made quite efficient even in the open air. The method I favour personally is to excavate in a sloping bank, build fire-brick round a suitable dop cask, run a flue out for a few feet and then stick on a good long chimney. When charged with imported coke there is not a better or quicker fire. I once had some petrol smelting furnaces, which were delightfully clean and effective. The cost for petrol was about the same as coke, but we could only renew spares by ordering direct from Los Angelos, and were reluctantly compelled to discontinue their use for that reason. For muffle work in the assay office the same type of furnace cannot be excelled, but it is somewhat noisy.

After the cyanide clean up is over, the small worker very often insists upon doing the regular pot assays in the same furnace for the rest of the month. Charges were weighed, fluxed and put into the pots under the lee of a buck-sail. The furnace was carefully dusted and the fire lighted, then, before the temperature rose, the pots were carefully put in by hand and packed round with coke. Results were repeatedly checked by bank assay and found correct. 30% of the assays were 2·5 A.T., residues going anything up to ·5 dwt., but what was most to the point, on the cyanide assays actual extraction corresponded with theoretical.

Straining zinc slimes on calico frames is still very common, but many ingenious modifications are also to be seen. The favourite is a box with a false bottom, and a suction attached to a hand pump, which removes the liquid percolating through the linen and causes a vacuum. The most effective I ever made was the false bottomed box, as above. The linen was strongly supported on wire screening again supported on a perforated board. Instead of a pump I made a steam ejector out of piping, and on passing steam through, it caused a most effective vacuum, and gave a thin cake of slimes as dry as though it had come from a press. Not only small workers but small mines run by rich companies are often guilty of calcining slimes in the open air. But here again, care counts for a deal. When the tray is filled, sprinkle the slimes with borax and press sheets of filter paper or newspaper well on to the soft mass. As soon as a good heat is obtained a skin is formed and the paper charred. Stirring, of course, is not to be thought of, but if too thick a layer of slimes be not put on the tray the roast is quite effective. Again, in charging pots the stuff is placed straight in on top of the flux with more as a cover. As the author states, I have found that it makes no difference to the product, and is a consideration under the conditions described.

The Rhodesian small worker very seldom has any inducement to put up a slimes plant. It can be safely assumed, that most of these small mines treat free-milling ore. Very often it is rubble and soft surface stuff, which will produce 50% slimes through a 600 screen, yet good amalgamation will leave very small values in either sands or slimes. It may be news to many that 3 dwt. rubble can be made to pay on a 5-stamp proposition, and a glance at the Rhodesian output will show numbers doing well on 4 to 5 dwt. One large mill I know runs its original slimes to waste. The cyanide manager places their value at 2 or 3 gr., but the figures are not guaranteed. I do know of a heap of accumulated tailings assaying from ·6 to ·8 dwt., which has returned a profit of £75 a month consistently. Would any member like to treat the slimes from that ore? When slimes values get to 3 and 4 dwt. or even more they are refractory and need skilled treatment, which the small worker is loth to seek in the proper quarter. When he wants a cyanide plant he calls in the local plumber, and then casts around for a cyanide expert (we are all experts up here!) to run it. To save tanks, he favours collecting in pits and gives you stuff like streaky bacon to leach. On one such plant I tearfully pointed out that with 48% slimes, in the tanks through badly shaped pits and 1 to 3% copper, which might be reduced by blanket strakes my extraction was at least 12% lower than it might be. After much thought on the part of my superior, the final result was my dismissal for losing 12%!!

Mr. H. A. White (*Member of Council*) read the following contribution:—

A METHOD FOR THE RECOVERY OF ZINC FROM SOLUTIONS OF SULPHATE.

(*Read at September Meeting, 1909.*)

By WM. CULLEN (Past-President) and G. F. AYERS (Associate).

DISCUSSION.

Mr. C. E. Meyer (*Member*): I should like to add my appreciation of the effort made by the authors of the above paper for solving the difficulty, which in my opinion is a question of importance and which before long will require a remedy. As I have been employed on a similar process having the same object in view (namely, the saving of zinc going to waste with present day cyanide practice) I may be able to give the authors some figures and facts to aid them or otherwise in their proposed new venture. I am not wholly of the opinion expressed by Prof. Stanley, that precipitation processes have no scope in this particular branch of the cyanide works, but will prove by figures that the waste proposed to be treated by the authors, is only a small part of the total waste, the greater portion being chemical loss and therefore not recoverable, rather than mechanical loss.

The sources of waste of zinc may be divided into two distinct classes:—A. Chemical loss; B. Mechanical loss.

A. *Chemical loss.* — This is zinc going in solution to the sumps from precipitation boxes. Also a small portion as double compounds of zinc and cyanide insoluble in a 10% sulphuric acid solution and loss from insufficient washing in the filter press.

B. *Mechanical loss.*—This is caused by a considerable quantity of zinc having to be dissolved in the acid vats to obtain the precious metal before smelting.

For the purpose of dividing these losses I will give you some figures from the October returns from the mines, and I will take four of our largest producing mines for this purpose:

FOR MONTH OF OCTOBER.

	Tons Cyanided.	Stamps No.	Gold from Cyanide.	Duty per Stamp.
Crown Mines -	120,466	675	14,983	7·13
E.R.P.M. - -	157,200	820	23,610	7·07
Randf. South -	92,812	400	14,028	8·44
Simmer & Jack	69,000	300	7,096	7·75
Average - - -	109,869	549	14,929	7·58

Average consumption of zinc in lbs. per ton of sands and slimes treated = 0·33 lbs. = 36,256 lbs. of zinc used in one month = 2·428 lbs. of zinc for every oz. of gold precipitated = 66·02 lbs. per stamp per month = £1·03 per stamp per month = 38·848 parts zinc to one of gold obtained.

The total number of stamps running on the Rand during the month of October, 1909, were 9,200. This gives £9,476 sterling as value of zinc used in one month.

Quoting the author's figures of £3,000 recoverable zinc in one month, we get

£3,000 value = Mechanical loss.
£6,476 value = Chemical loss.
The total zinc used = 38·848 to 1 of gold.
which gives us 12·3 approx. = Mechanical loss.
26·242 = Chemical loss.
or less than 1-3rd.

I do not think that this figure of £3,000 was arrived at correctly, as the other base metals such as iron, lead and copper were neglected in this calculation. This would tend further to reduce the recoverable zinc. If this saving of £3,000 were calculated on the basis of tons cyanided, after deductions for treatment, labour, etc., it would be seen that this figure would not be near ¼d. a ton cyanided, and would hardly afford facilities for its treatment with a satisfactory margin of profit. This process would also be subject to its universal adoption by all the crushing mines. With the variations also which occur in individual cyanide works from month to month, which are between 2d. and 3d. per ton cyanided, there seems very little hope of its success, and would hardly warrant the erection of a plant.

The only process which in my opinion is bound to succeed is the one which embodies the recovery of the zinc, decreases the mechanical loss, and if possible recovers the solvent used during the process of cleaning up, and, which is the most important, consequently decreases the loss of gold:—by handling of the rich product, by loss through heavy solutions from filterpress, by calcination. This loss has been quoted at a very high figure in the past, and should therefore be worthy of further investigation. The present-day process for cleaning up not only leaves the zinc in a bad form for recovery, but has so many objections that Messrs. Cullen and Ayers' paper is one to be highly recommended, and it is only by combined efforts and interchanging of views on the subject that a solution is likely to be found.

The Chairman: We thank the author for his contribution, which I am sure will give Messrs. Cullen and Ayers something to think over in their reply.

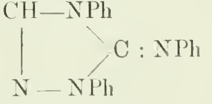
ANALYSES OF GASES FROM BURNING NITRO-GLYCERINE EXPLOSIVES.

(Read at September Meeting, 1909.)

By W. Cullen (Past-President) and D. W. Greig (Associate).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I had intended to discuss this sooner, but find I have no comments to make on their ingenious process of analysis. I just wish to put on record the process I myself used 3½ years ago (this *Journal*, 1906, p. 307), as it has the advantage of greater directness. This method is not mine, being essentially the same as one invented by M. Busch about 6 years ago. The fumes are caught in excess soda and treated (cold) with $KMnO_4$ as long as turned green and acidified. This converts the nitrogen (originally present as a mixture of nitrate and nitrite) entirely into nitrate. The solution (which should be pink) is boiled with a little sodium acetate filtered from MnO_2 if necessary, and precipitated by *nitron* acetate, allowing three hours' cooling. [*Nitron* is a remarkable yellow base :



which gives an insoluble nitrate and is becoming very useful in many technical analyses] and weighing as nitron nitrate $C_{20}H_{17}N_4(NO_3)$. I find there is a discrepancy, however, in our results as regards blasting gelatine, the authors having found about five times as much NO_2 per gram of *gelatine* as I did, although as regards *gelignite* we are in agreement (I found 186 c.c. per gram, they find 202 c.c.). In the former case I consider that if *gelatine* burns vigorously (finely divided) it does not give off so much NO_2 as it does when it burns slowly from a lump, and this point ought to be gone into again: and while they are at it, the authors ought to determine CO_2 and CO in the gases to see if they can confirm my statement, that CO_2 is the chief product from gelatine whereas CO predominates in gelignite fumes.

The Secretary read the following contribution :—

THE BARBERTON GOLDFIELD.

(Read at October Meeting, 1909.)

By A. Richardson, M.I.M.M. (Member of Council).

DISCUSSION.

Mr. E. Browne (*Member*): Alluvial gold was first found on Barrett's in 1881, and considerable quantities were recovered from the various creeks. The Barrett G. M. Co. was formed in 1883, and owns the Barrett's Berlin farm, situated on the plateau between the Kaap and Elands Valley. Crushing started with 6 stamps, and afterwards 30 stamps were erected, but these were discarded in 1894 in favour of the present treatment on account of the excessive amount of slime, which carried most of the values and could not be treated at that time.

The ore, which is a kind of rotten reef, is found in irregular leads and deposits in decomposed diorite over large portions of the farm, and is mined from open workings which vary from a few feet to 100 ft. in depth. The ore is trammed about 4 miles and lowered by self-acting inclines to the mill, which is about 1,000 ft. below the level of the mine, and passed through trommels of ¼ in. mesh, the fines going to the fine ore bin and the coarse to the crusher. The crusher is a Whittaker mill, consisting of a revolving pan with two mullers and a perforated bottom. The grates have holes of ⅛ in. and are replaced when these wear too large. The mill is driven by a Pelton wheel working under a 350 ft. head. From the crusher the ore passes to the fine ore bin and is then trucked to the vats. The vats are 21 in number, and from 30 ft. to 35 ft. in diameter, but only 4 ft. deep, this shallow depth being necessitated by the slimy nature of the ore, amounting to 40% as it comes out of the mine. When a vat is filled, a strong solution of ·15% KCy is run on and the leaching cock opened; percolation is very slow, the solution taking about two days to run through ; weak solution and water are run as usual, and finally the vats are sluiced out through bottom discharge doors into the creek. 3,000 tons a month of an average assay value of 2·25 dwt. are treated, the treatment lasting 18 days, and the extraction being 91%. The treatment presents no difficulties although the ore carries a good deal of copper and manganese and sometimes arsenic, cobalt, and nickel ; the use of lead acetate has been found to be superfluous and no white precipitate is formed in the boxes. The gold slime is treated with acid and smelted in Salamander crucibles with clay liners, and an analysis of it gives :—

Au	44·44
Ag	0·88
SiO_2	8·88
ZnO	20 30
Cu	11·40
Pb	4·97
Fe_2O_3 (by diff.) ...	9·13
	100·00

The low working costs, about 7s. a ton, which are rendered possible by the simplicity of the mining, and a plentiful supply of water for power and other purposes enable this low grade ore to be worked at a profit. I have to thank Mr. Acutt, the manager, for permission to send these notes.

Mr. W. A. Caldecott (*Past-President*): The reference in the last speaker's remarks to the trommels used at the Barrett G. M. Co. is of interest, inasmuch as these were mentioned in one of our earliest papers.*

The Barrett ore is typical of those suited for dry crushing and direct cyaniding, since it carries very finely divided gold, and is of a porous friable nature, not requiring fine crushing to expose the gold contents. With hard pyritic ore carrying some coarse gold, direct cyaniding is impracticable on account of high operating and maintenance cost when crushing dry to the requisite fineness: such fine product is unleachable under ordinary conditions, and the exposed gold cannot wholly dissolve in a practicable time of treatment.

As illustrating the influence of conditions on working costs, I believe the lowest total mining and milling costs in South Africa, all under 10s. per ton, have been those of the three companies —the Wanderer (Rhodesia), Pigg's Peak (Swaziland), and Barrett (Kaapsche Hoop)—all of which had soft oxidised surface ore, and adopted a similar method of dry crushing and direct cyaniding to that described.

THE ASSAY OF CYANIDE SOLUTIONS AND SLIME RESIDUE CARRYING DISSOLVED GOLD.

(*Read at October Meeting, 1909.*)

By A. Whitby (Member of Council).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I have made some experiments to clear up the theory of Mr. Whitby's process. I made solutions of $KCuCy_2$ and $KAuCy_2$ and mixed them and, of course nothing happened; on acidifying a mixture of CuCN and $CuAuCy_2$ (cuprous aurocyanide) was precipitated and the filtrate from this was entirely free from gold when tested by evaporation with aqua regia and adding $SnCl_2$. On the other hand, when $KAuCy_4$ was used instead of $KAuCy_2$, although the result was the same to the naked eye, I found unprecipitated gold in the filtrate, the reason being presumably that $CuAuCy_4$ (cuprous aur*ic*yanide) is not totally insoluble in water. What happens in Mr. Whitby's process is this: when $CuSO_4$ is added to cyanide solution we get potassium cuprous cyanide and free cyanogen — $CuSO_4 + 3KCy = KCuCy_2 + K_2SO_4 + CN$. In presence of acid, we get CuCN precipitated because the acid $HCuCy_2$ (contrary to $HAuCy_2$) is extremely unstable. If we add sulphite, the free cyanogen is removed, and consequently the second reaction can take place unimpeded: $KCuCy_2 + KAuCy_2 + H_2SO_4 = CuAuCy_2 + K_2SO_4 + 2HCN$. But if sulphite is omitted, the cyanogen converts some or all of the gold into $KAuCy_4$ and we get imperfect precipitation, as I have shown above, and as Virgoe found long ago in practice.

EXPERIMENTS ON THE ASSAY OF ACID WASHES RESULTING FROM THE CYANIDE "CLEAN UP" BY THE USE OF BISULPHATE.

(*Read at October Meeting, 1909.*)

By L. J. Wilmoth (Member).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I had long suspected that acid washes contain not only suspended Au but also Au present as $HAuCy_2$. I have made the latter and find it surprisingly stable. Its dilute solution can be boiled for some time before it decomposes into AuCN (yellow) and HCN. It was prepared as follows:

(1) $Ba(OH)_2 + H_2S + 2AgCN \rightarrow BaCy_2 + Ag_2S$ etc

(2) $2BaCy_2 + 2Au + O = Ba(AuCy_2)_2 + BaO$.

(3) $Ba(AuCy_2)_2 + H_2SO_4 = BaSO_4 + 2HAuCy_2$.

This fact is very interesting, seeing that both $HAgCy_2$ and $HCuCy_2$ decompose as soon as made.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

Part I.—Crucible Experiments.

(*Read at November Meeting, 1909.*)

By Prof. G. H. Stanley (Member of Council).

DISCUSSION.

Mr. A. F. Crosse (*Past-President*): On looking up my old notes on the analyses of titaniferous iron ore we received at Messrs. Johnson, Matthey & Co's., in London many years ago, I find that the ore was more impure than the titaniferous ore we have in this country. The Norwegian ore contained 1·56 % of pyrites, which in addition to the titanic acid was certainly a great objection.

The Chairman: Were these ores successfully worked !

Mr. A. F. Crosse: I cannot say what was done with it. It merely passed through our hands for sampling and analysis.

---

* See this *Journal*, Vol. i., August 21, 1894, p. 23.

Mr. A. Adair (*Member*): Prof. Stanley's paper has interested me very much as a former ironworks' chemist, and again as having myself experimented with titanium ores. As Prof. Stanley is working for a definite practical purpose, it is a pleasure to think one may be able to support him in a task that is not an easy one. The real trouble with titanium ores, is white iron, loss in the slag, and the tendency to form *bears* and *scaffolds*.

In the following remarks and suggestions, one has always to remember, that however careful one may be to provide against contingencies, one can never trust a blast furnace to understand what is required of it, when dealing with new ores. The results are sure to be surprising in some way, and seldom agreeably so. In changing twyers we often get little glimpses of what is going on inside, and many curious samples. I often noticed lumps of ore originally hæmatite or other varieties which accidentally got to the twyers without melting, or even being touched by slag. The porous ores were invariably saturated with carbon, ilmenite would come through apparently unchanged. On testing the former with weak acid, the strong odour of carburetted hydrogen was very marked, with ilmenite scarcely at all. When a scaffold was forming, pieces of it were procurable also from the twyer house—peculiarly sticky lumps (when hot) of slag and semi-fused wrought iron. The *bears* were masses of iron as tough as, and similar to, wrought iron, with abundance of "kish" and some pieces of fire-brick from the lining, also cyano-nitride of titanium, mostly adhering to the metal. Soft hæmatite would (other things being equal) always give a good grey iron with rough fracture and high content of graphite. Compact and dense ore would give an iron with finer fracture and less graphite, whilst ilmenite mixed with the ore would always give white iron and (incidentally) trouble.

From these and other observations, I venture to suggest that the physical character of the ore, viz., its porosity or the lack of it, is the principal factor to take account of in dealing with iron ores. I suggest also that titanium is a hindrance, only so far as it may prevent thorough carbonising of the ore. Further, one may presume that it is the lack of sufficient carbon in this way, which is responsible for the production of wrought iron, instead of the easily fusible carbon compound, and therefore of scaffolds, bears, white iron, and losses in the slag. I judge that with a porous ore, and sufficient time in the reduction zone of the furnace, satisfactory results would be obtainable. Such results might be realisable in practice, by using a slow fusing flux, say a ferruginous clay (for its alumina) and dolomite, to get 50% of lime and magnesia in the slag.

If we take grey iron as the index of success in treating titanium ores, then Prof. Stanley would seem to have achieved success, or near to it, in his experiments 4 to 7, especially in the last case. With 45% of lime or its equivalent, the slag would not be so fusible therefore, a longer time available for carbonisation would be given before melting.

The experiments with the silicious slag are also of great interest as it seems as if the author might be successful in this direction. I would however suggest that a silicious slag would require a special lining, approaching Dinas brick or gannister. The Vereeniging fire-clay with admixture of quartz would perhaps meet the case.

I shall look forward to Prof. Stanley's next contribution, and most heartily wish that his efforts may be rewarded with an unqualified success.

The Chairman in closing the proceedings wished the members a Merry Christmas and a good New Year.

Contributions and Correspondence.

STOPE MEASUREMENTS.

Thanks are due and have been expressed to Mr. Tonnesen for his paper on Stope Measurements, and his detailed description of his Stereometer and its working.

I would add my appreciation, and hope that his instrument may be the means of increasing the efficiency and easing the work of the mine surveyor, as I quite agree with his statement that stope measuring is a cause of "hardship and sometimes of danger" to the mine surveyor, though I am not in accord with the natural inference, that in his work there is an excess of hardship or of danger over that undergone by others whose work lies underground.

I must, however, endorse Mr. Johnson's remark, that, given competent and painstaking men, accurate work can be done, and undoubtedly is done with the present methods. In this connection I would instance a case in which a miner disputed his measurements. Two Government surveyors, both accustomed to mine work, were called on to check the mine surveyor's work, one by the miner, and the other by the mine authorities. They checked the areas mined from the two stopes in dispute, both working by tape triangulation, the mine surveyor having worked with a transit theodolite. The dip of one stope was 30°, with an area about 200 fathoms, of the other the dip was 42°, with area between 600 and 700 fathoms. The agreement between these two checks and the areas totalled from the duplicate cost sheets in the mine office, was between 1% and 2%.

In his objections to theodolite stope measurement the author mentions the difficulty of setting up under the peg in a steep stope. I have found that two 18 in. holes, in suitable positions, drilled and blasted in the footwall, under the peg, gave all the required convenience for setting up and standing room, and, of course, served as long as that peg was in use.

As to the office work, the preliminary reductions of inclined to horizontal distances is obviated by plotting with Bristol's protractor, one of which I had the benefit of using for some time. The stage in the office work described by the author and Mr. Coaton as the chief objections to this method—that of converting "horizontal fathomage to inclined fathomage" depends, of course, on the correctness of the observations for dip and change of dip.

I endorse Mr. Coaton's descriptions of the best method of doing this, and would only add that, in observing the strike line, the candle at the face must be held at approximately the same distance from the same hanging or footwall as is the telescope of the theodolite.

With most of the author's objections to the Tape Triangulation method I am in agreement. The tape sag error can be largely obviated by using more lines, and those approximately dip lines, from which to offset; but the angular error of offsets, which is largely a personal error, and partly due to difficulty in climbing about steep stopes, is a strong objection to the method.

B. Travers Solly.

Germiston,
18th December, 1909.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Testing of Galvanized and Other Zinc-Coated Metals.—"Three methods are now commercially used to coat iron with zinc. The oldest and for many years the only commercial method is the '*hot-galvanizing*' process, in which the iron article is simply dipped into a bath of molten zinc. The second is the '*wet*' or '*cold-galvanizing*' process, in which zinc is plated on the iron article by electrolysis from a solution of a suitable zinc salt. The third is the '*sherardizing*' process of Sherard Cowper-Coles, in which the iron article is heated in a closed chamber in the presence of finely divided metallic zinc and zinc oxide. (See our Vol. VI., page 189.)

Preece Test.—Although there is a marked difference in appearance and in structure between these three kinds of zinc-coated iron, only one method of testing them has been used, namely, the 'Preece' or copper-sulphate immersion test.

As standardized by the American Telegraph and Telephone Company this test is essentially as follows: A standard solution of copper sulphate is prepared by agitating a nearly saturated solution of this salt with copper oxide until it is perfectly neutral. It is then diluted to 1·186 sp. gr. at 65 Fahr. When a rod of zinc and one of iron are placed in this solution the metals pass into solution as soluble sulphates, an equivalent weight of copper being at the same time deposited on the metallic rods. The copper deposited on the zinc surface is black, spongy and easily removed, while on the iron it is dense and of a bright color.

When this reaction is used in testing galvanized iron it is assumed that the speed of solution of the zinc is a direct function of the time of immersion, and by the number of one-minute immersions necessary to dissolve the zinc from the iron and to deposit a film of bright copper thereon, the thickness of the zinc can be estimated. It will be seen that this method measures the thickness of the coating only at its thinnest point, and that the assumption is made that no bright copper will be formed until the iron base is reached. To appreciate the limitations of this test, it is but necessary to note the structure of the three classes of zinc-coated iron.

A polished oblique section of ordinary hot-galvanized iron etched with iodine reveals three distinct layers. First, a coating of zinc, which varies in thickness in accordance with the temperature of the zinc and the amount of squeezing or wiping applied to the article before the zinc solidifies; second, a distinct layer of a zinc-iron alloy, termed alloy B, which varies in thickness with the temperature of the zinc bath and the length of time the iron was subjected to it; third, the iron base itself. Between the zinc and alloy B there is generally a second alloy, alloy A, which is a thin, discontinuous layer, richer in zinc than alloy B, and between the iron and alloy B is a third alloy, relatively unimportant, which is very rich in iron, alloy C.

The structure of wet-galvanized iron is comparatively simple, the layer being practically pure zinc.

Sherardized iron, on the other hand, presents a relatively complex structure. The metallic zinc penetrates the iron, forming deep layers of the alloys B and C, and in place of alloy A there occurs a number of compounds, as yet unidentified. Upon the surface there is generally a layer of relatively pure zinc, although frequently the process is carried to the point where only a deep layer of alloys is formed. When examined under the microscope this alloy is seen to be covered with deep cracks or fissures as though the alloy in forming had contracted. It is thus apparent that in testing galvanized iron made by these different processes we are dealing with three very different materials.

Desiderata for a New Test for Determining Durability or Resistance to Corrosion.—It is the purpose of this paper to treat of zinc-coated iron with regard only to its durability or resistance to corrosion, and not to its tensile strength. From this standpoint a test should, if possible, indicate the following:

(1) The uniformity and thickness of the zinc coating; (2) the continuity of the coating with reference to pin holes; (3) the purity of the zinc, and (4) the toughness and ductility of the coating.

The corrosion of pure zinc in water is very slow. If, however, the zinc is in electrical contact with any material upon which the hydrogen can be liberated, the corrosion of the zinc is relatively rapid. The iron-zinc alloy *B*, and iron itself, are both surfaces on which the depolarization action can take place, and hence, so long as neither the iron nor the alloy is exposed, other things being equal, the zinc coating will not corrode. The importance, therefore, of maintaining a uniform coating of metallic zinc upon the iron can be appreciated.

To determine whether the ratio of the time of immersion required to produce a bright copper surface was in reality a measure of the thickness of the zinc coating, samples of galvanized sheets were obtained and tested. The weight of zinc per square inch area was determined by dissolving the zinc from the sheets without attacking the iron-zinc alloy. This can readily be accomplished by heating the galvanized sample, together with a piece of metallic iron, in boiling caustic soda until the generation of the hydrogen ceases. As might be expected, the ratio between the time of immersion required to show the presence of bright adherent copper and the amount of zinc present per unit area depends upon the uniformity with which the coating of zinc is spread upon the iron base. In the case of 14 pieces of galvanized sheet iron, for example, the average ratio found was 23, while the greatest variation in either direction was but 20. This test, therefore, so far as indicating the uniformity of the coating and measuring the thickness of the zinc are concerned, is fairly satisfactory; but alloy B is always indicated as iron, and hence, when the test is applied to sherardized articles, very erroneous and misleading results may be obtained.

The presence of the channels or pin holes caused by the free iron surface coming in contact with the zinc and causing it to pass into solution cannot be detected by the copper-sulphate method, as these pin holes down through the zinc to the iron fill up with black, spongy copper and cover up the bright copper spot at the bottom.

The purity of the zinc cannot be determined, as the bright copper particles precipitated by this method are so small that they are lost in the mass of black, spongy copper.

Finally, the fourth factor, the toughness or the ductility of the coating, cannot be determined, due to the same tendency of the spongy copper to cover up the bright copper deposits, which indicate cracks extending down through the zinc coating to the alloy or iron underneath.

Test for Detecting Pin Holes and Cracks.—The relation between the presence of pin holes in the zinc surface and cracks due to a brittle coating and the durability of the structure as a whole has not heretofore been studied, largely because no method of detecting these imperfections has been available. The following phenomenon serves as a basis for such a test. If a piece of zinc be placed in a strong solution of caustic soda heated to about 100° C. no action is noticeable. If now the zinc be touched with a piece of iron, hydrogen is liberated in great volume from the iron. Small cracks in the zinc coating may be easily detected in the same way.

Ordinary hot-galvanised ware is generally very free from imperfections of this kind, while wet-galvanized ware, on the other hand, is frequently very porous, generating hydrogen at numberless small points over its entire surface. Theoretically, the best electroplated surface should be that carrying the greatest weight of zinc per unit area, deposited at the slowest rate. That is, the lower the current density at the cathode in plating the less porous will be the deposited metal, and the thicker this dense deposit is the better protection will it be. A great many tests on electroplated zinc-coated iron were made, the samples varying in both these particulars. In every case, those deposited most rapidly were the most porous, and the results showed that the time of plating was more important than the weight of zinc per unit area, although there was a minimum below which it was not safe to go.

Tests for Determining the Combined Effect of Imperfections.—As the rapidity of hydrogen generation from zinc is a function of the presence of an alloy of zinc and iron, hence the more impure the zinc coating, the more impure will be its solution in acid. Accordingly, the impurity of the zinc, the presence of pin holes and cracks, and the thinness of coating are all factors which act in an accumulative way to increase the rate at which hydrogen is generated when the zinc surface is exposed to acid. This method has been developed by Mr. Charles L. Campbell in his thesis for the B.S. degree at the Massachusetts Institute of Technology, and found to give very concordant results. The apparatus employed may take any convenient form, it being but necessary to expose a known area of the zinc coating to a standard acid solution under uniform conditions and to measure the hydrogen generated per minute. In almost every case a sharp maximum is reached giving the resultant of all the different factors which make for rapid dissolution of the zinc. Thus, in a series of wet-galvanized sheets, those electroplated for 15 minutes reached a maximum of 7 cub. cm. in three minutes; those plated for 30 minutes a maximum of 14 cub. cm. in 10 minutes, while those plated for 45 minutes showed a rather poor maximum of 6 cub. cm. in 30 minutes. In many samples of sherardized product there were two maxima on the curve showing the amount of hydrogen generated per minute. This indicates that there is first an action between the outside zinc coating and the iron-zinc alloy, and later a well defined action between the alloy and the iron.

Service tests with galvanized iron are of necessity very slow at best, and the relation of the above phenomenon to the real durability of the material can be determined only after a number of years. Something can, however, be learned by the study of ware which has already been exposed for sufficient time to determine its durability. Most of the material available for the purpose was in the form of fence wire; from a study of these the following conclusions are drawn:

First, thickness of coating,—In every instance where a very durable fence was found, the coating of zinc proved to be relatively very thick. On the other hand, the wire fences which showed marked corrosion in from one to two years proved to have almost no zinc on the iron, the zinc color being due to a layer of alloy alone. It is doubtless true that the purity of the iron used in the wire itself plays an important part in determining the ultimate durability of a fence, yet in the opinion of the writer the thickness of the zinc coating calls for more immediate attention. The modern method of close mechanical wiping the wire as it emerges from the zinc-bath produces too often a wire covered not with zinc but only a thin layer of electro-negative iron-zinc alloy, which affords but little protection to the iron. To produce a wire with a liberal coating of zinc would, of course, cost more both on account of the extra zinc and on account of a somewhat smaller production per machine.

Second, purity of zinc,—It is probably a necessary consequence of hot galvanizing that the zinc becomes somewhat contaminated with dissolved iron It is apparent that this should be kept at a minimum in order that the coating should be of maximum durability.

Third, flexibility of the zinc coating,—The important objection to placing a thick coating of zinc upon wire used for fencing is that such a coating cracks off when running through the machine. This objection applies only to those wires which are subjected to very sharp bends or turns. It is possible that by passing the zinc-coated wire through dies or under grooved rolls that the crystalline condition of the zinc could be destroyed and the flexibility and ductility materially increased.

To sum up the results of this paper:

1. The copper sulphate immersion test is of value in determining the uniformity of coating and relatively the thickness of coating in hot and in wet-galvanizing products; but it is worthless in the case of sherardized products, and gives no idea of other important factors involved in the durability of the structure.

2. By immersing a galvanized product in hot, strong caustic soda, the presence of any unprotected iron may easily be detected, however small such area may be.

3. The weight of zinc per unit area apart from the zinc-iron alloy may be analytically determined by dissolving the zinc from the plate through treatment with hot caustic soda, while in contact with metallic iron.

4. Theoretically, the rate at which hydrogen is evolved when the galvanized product is immersed in dilute acid should indicate its relative durability; inferior products should reach a maximum in a few minutes compared with a much longer time for better products.

5. In the very important matter of fencing wire, while the purity of the iron used is of consequence, a more immediate necessity is a heavier coating of zinc on the wire.

The flexibility of a zinc-coated wire may possibly be increased by mechanically working the wire in rolls or dies to destroy crystallization in the zinc coating.'—Dr. W. H. Walker.—*Electrochemical and Metallurgical Industry*, Oct., 1909, p. 440. (H.A.W.)

METALLURGY.

The Development of Heavy Gravitation Stamps.*—"Mr. W. Fischer Wilkinson introduced the paper by summarising its salient points.

Mr. S. J. Truscott said that in a previous paper, read in October, 1904,† Mr. Caldecott had enlarged upon the necessity and advantage of finer crushing. The paper that evening might be taken in continuity of that idea, especially as the author had expressed a desire in his former paper that it, being more or less general in treatment, should be followed by others giving details of actual achievements.

The author stated at the commencement of the present paper that the history of ore crushing, by means of gravitation stamps, showed a progressive increase in their weight and in their corresponding efficiency. In that advance South Africa must be taken to have maintained a very forward place. Although to those in this country the Rand appeared as a whole, it was nevertheless the fact that the mines there were distributed in well-marked groups, of which the Consolidated Goldfields was one, and it had to be stated that the credit for that advance in the direction of greatly increased weight of stamp was at the outset largely due to the engineering staff of the Consolidated Goldfields, of whom Mr. Caldecott was one.

Before making that step of moving forward to greatly increased weight, a series of trials was carried out at the Knights Deep battery, and they, as engineers, must acknowledge that trials made under such conditions must have their great regard. A portion of the battery was set apart, and a special staff was deputed to carry out the trials, and everything was arranged with the highest intelligence, so that the results were in every way trustworthy.

The first lines taken in those experiments were to try the effect of a double discharge as against a single discharge, and the effect of a preliminary very fine crushing. The results obtained were tabulated on p. 3, from which it was seen that with the same weight of stamp, the same drop, and the same number of drops per minute, under those conditions there was no advantage in either departure from the ordinary practice. Apparently there were other trials made which were not detailed, but it was stated that after those experiments it became obvious that, in order to crush more rock, a simple increase of stamp weight was the best course.

It appeared to him that that was a change of direction, that the experiments so far were after greater efficiency; in fact, on p. 2 it was stated distinctly that the object of the trials was to discover some means whereby stamp milling efficiency could be increased. But it appeared to him that the idea of crushing more rock was a departure from seeking after efficiency, and it seemed to be greater capacity which was now the object.

They must regard greater efficiency as consisting in the attainment of more work for the same consumption of power, but greater capacity arose from obtaining more work with a correspondingly greater amount of power consumed.

Then a series of tests was made to show the effect of increasing the weight. The results of those tests were, as was naturally to be expected, that with the heavier stamps there was more ore crushed. But the efficiency must be expressed in terms of the power consumed. It was only reasonable to consider that there would be extra power consumed in almost exact proportion to the extra weight lifted. Therefore they would expect that the extra amount crushed would be also proportional to the extra weight lifted.

To the statement on p. 5 he would add a new column, and set it parallel with that in which the duty was expressed. In that column he had put the figures which should be obtained if the extra amount crushed was proportional to the extra weight. It would be obvious that the first figure (5·88) remained the same.

Tons of Ore Crushed per Stamp per 24 Working Hours.

Actual.	Calculated.
5·88	5·88
6·58	6·20
6·74	7·53

Going to the second series of experiments, the 4·26 remained the same.

Tons of Ore Crushed per Stamp per 24 Working Hours.

Actual.	Calculated.
4·26	4·26
4·29	4·51
4·55	4·53
4·96	4·68
5·17	5·47
6·02	5·62

---

* See this *Journal*, vol. x., Sept., 1909, pp. 108-115.

† *Trans. Inst. M.M.* xiv, pp. 48—59.

The figures given were the results of the extra crushing which one would naturally expect by the increase of weight. There was no increase of efficiency; they were simply the results which one would expect by the increase of power consumed. It would be noticed that those figures were somewhat variable. Sometimes the calculated one was higher and sometimes lower than the actual one obtained.

In order to eradicate those variations the average was taken. Of the first series of experiments, the average of the actual tons crushed was 6·66; the average of the theoretical was 6·86. That was, in that case, they would have expected 6·86 to have been crushed, whereas 6·66 only was obtained. In the one below, the average came out for the actual duty 5·00, and what they would have expected was 4·96. That was slightly less. It would be seen that there was no increase of efficiency. There was an increase of capacity without increase of efficiency.

It was also to be remembered that there were other varying factors connected with it. Especially in the lower series, the amount of water had been increased considerably along the line. He thought it must be accepted that the flow of water through the box had influence upon the duty, upon the efficiency. The author agreed with that, because on p. 15 he said that the ratio of water fed had influence upon the tonnage crushed.

It might also be seen from this statement that where the amount of water used had been more, the actual figure crushed was greater than the theoretical calculated: and the reverse was also the case. That held through the whole series, and was exemplified in the last two cases on the page.

In the last figure but one 5·05 tons of water were used per ton of ore; that was the lowest of the whole series, and in that case the actual crushing was at its lowest when compared with the theoretical crushing. Taking the very last figure, where 6·30 tons of water were used, that was the very highest, and the actual crushing was at its highest as compared with the theoretical crushing.

It might be taken also that the increase of water was hardly measured in tons of water per ton of ore crushed; it was rather in the tons of water going through the box. Taking it in that way, there were 23 tons of water going through with the lightest stamp and 38 with the heaviest; that was an increase of more than 50%.

It had also to be mentioned, as bearing upon the work done, that if one referred to the last column in the statement on p. 5, where the amount per cent. +60 was given, it was seen that in the case of the last figure, where 6.02 tons of ore were crushed, the amount per cent. remaining of the +60 was 11·66, whereas with the lighter stamp with which everything had been compared it was 5·16. That was, in the case where the actual duty was comparatively high, it had to be discounted by the fact that it contained a good deal of stuff, more than usual, above the +60 mesh. He thought from that statement they could not say there had been any increase of efficiency; in fact, if there was a difference, there had been a decrease in efficiency. There had been an increase in capacity.

There was another point, connected with the preliminary fine crushing. He thought it was certainly the case, and conceded, that the long-established practice on the Rand was an approach towards efficiency, and that any departure from that practice must be also regarded as a departure from efficiency. So that, when they saw, as in column B, on p. 3, that ore of such fine stuff as was passed through rolls, set to half an inch, was crushed under stamps of more than usual weight, they must say that there was such a departure from general practice as must necessarily be accompanied by a departure from efficiency.

From that point of view he thought the experiment under column B could hardly be said to represent a fair case. He thought it more reasonable to assume that if in crushing that fine stuff a lighter stamp had been used the same duty might have been obtained, and a correspondingly greater efficiency.

It seemed to him also that that was a line to which more consideration might be given, that the ore as it left the crushers, where it was crushed say to 2 in., contained a good deal of stuff which was very small, and a good deal which was fairly big. If efficiency was sought, it might be along the line which took the coarser stuff to the heavier stamps, and the finer to the lighter stamps.

There were many mines upon the Rand now which had started with light stamps and which had recently added, or were adding, heavy stamps, so that there they had already a differentiation between heavy and light stamps.

It must not be gathered from those remarks that he disputed in any way the great advantages of the heavy stamps. He was as much in favour of the heavy stamps as most of those present. The advantages were greatly economical. If one could get 100 stamps to do the work previously done by 200 stamps, one had economy in all directions—of shafting, of administration, of supervision—and that economy was so large that naturally heavy stamps would probably be extended.

They had probably heard that the most recent achievement on the Rand was the case of the stamps of the West Rand Consolidated, where a trial run was made recently, and a duty of 15 tons was obtained, producing stuff for the tube-mill to work upon. He even thought that the latest report was that a duty of 17 tons was obtained. Those stamps weighed 1,890 lb.

In the City Deep there were now stamps of 2,000 lb. Their mill duty was guaranteed at 11 tons, with a probable ultimate amount of 20 tons, though that was not guaranteed. It seemed to him that the limit of heavy stamps had not been reached. Possibly it would be determined by whether the cam shafts and cams remained satisfactory.

He thought that stamps would become heavier until at last the cams and cam shafts were thrown out, and some other lifting device was obtained, or a heavier blow achieved in some other direction.

That statement, favourable to heavy stamps, would seem to be somewhat in contradiction of the fact that it was generally recognised that for the ordinary run of stuff going to the battery there was a blow which was most effective; but it was not so contradictory as it might appear, because, although the statement was true, it was dependent upon the condition that the diameter of the shoe and die was fixed. This had practically been the case; the diameter of the shoe and die was many years ago about 9 in., and he believed in the case of the City Deep the diameter was 9½ in., which represented about the same thing.

It seemed to him that now they were erecting heavy stamps, some trials should be made in the direction of increasing the diameter of the shoe and die, in which direction he thought some greater efficiency lay. If one increased the stamp without increasing the die, one got a needlessly strong blow, because it was an admitted fact, he thought, that there was a lighter blow which was equally effective.

It was interesting and opportune to say that in the case of the Dolcoath Mine, where pneumatic stamps were employed, they were used at first with an ordinary-sized die and were not satisfactory; the die was then increased to 12 in. diameter, and the stamps were now quite satisfactory.

Mr. H. S. Denny said that so much had been heard of the increased efficiency attaching to the corresponding increase in weight of the gravity stamp in the past few years, and so many of the data had emanated from the mines with which Mr. Caldecott was associated, that it was fitting he should come forward with a pronouncement on the subject before the Institution.

It might surely be taken as axiomatic that there was a critical point in the weight of a stamp which would satisfy all the contingent factors of capital outlay, operating cost, horse power consumed, crushing efficiency, maintenance, and repairs, short of which point or beyond which point there would be losses in economy.

Mr. Caldecott in his notes undertook to show, not that he had established this critical point, but that the increases in weight already carried out had resulted in very appreciable improvements in economy.

Whilst he, the speaker, did not for a moment dispute this statement, he must say that a close scrutiny of Mr. Caldecott's paper did not appear to provide the data essential to the proper substantiation of his contentions. They were told that with 1,605 lb. stamps 6·02 tons of ore were crushed per twenty-four hours, yielding a pulp product of which 11·66% remained on a 60-mesh screen. This was in itself a very extraordinary performance, but they were given to understand that the operating cost per ton crushed was less than that obtaining with lighter stamps, although in his summary of the advantages of heavy stamps Mr. Caldecott did not specifically make this plain. One could take it, however, that as the whole question was purely one of ultimate economy, Mr. Caldecott meant them to understand emphatically that the results obtained from the 1,605 lb. stamps did show a great economic improvement over the 1,250 lb. stamps.

The mere facts that there was a reduction in the size of the mill building and that 30% less shafting and other moving parts were needed and that less labour was required, did not in themselves constitute a final and convincing case for the heavy stamp. There were a great many other things that formed important counters in such a computation and from his point of view he would like to see Mr. Caldecott's statement supported by supplementary figures in which the following points would be dealt with.

1. Comparative statements of capital cost of a 1,250-lb. stamp battery as compared with a 1,600-lb. stamp battery of equal capacity, complete. The number of gravity stamps in the respective batteries would remain the same, whilst the cost of fine crushers for the lighter stamps would of necessity come into this statement.

2. A power curve illustrating the fluctuating efficiency of the respective batteries on varying numbers of stamps running, together with a statement of the actual consumption of power per ton of ore crushed and a grading analysis of the pulp produced.

3. Data as to the comparative time and labour involved in carrying out repairs to either battery, taking into consideration the heavier units of the bigger mill.

4. A statement showing the comparative effect of loss of time in hanging up a battery of the heavy stamps as compared with lighter ones.

5. A general statement showing that the advantage still lay with the heavier stamps, taking the lives of respective batteries into consideration and taking into account the initial capital outlay with redemption and interest charges properly apportioned, and figuring on the respective capacities of the two mills on equal products, and finally calculating in all those other factors of time and labour interwoven in the question.

It must not be forgotten that the framing and foundation and driving gear and component moving parts must all be correspondingly heavier and more expensive, and renewal units more unwieldy, and whilst one could readily grant that these points were of minor importance they must not be entirely overlooked.

Two points of the greatest importance that had always been a subject of discussion, and which were fraught with the greatest bearing on the capacity of the gravity stamp, were the size of the ore being fed into the mill and the coarseness of the screens used for the discharge. If the stamp mill were asked to do the work of either the stone breaker or the grinding machine, that work would be done at an economic disadvantage, as there were other machines in which these two operations could be much more efficiently carried out.

Mr. Caldecott said "fine breaking before milling on heavy stamps does little good," and he (the speaker) must confess that the figures in column B of the first table rather surprised him. Of course it was clear that with the heavier stamp it would not be economical to resort to so fine a preliminary crushing as that found most profitable with the 1,050 lb. stamp, but he was not prepared to find that all the advantage disappeared when the weight of the stamp was increased to 1,342 lb. According to Mr. Caldecott's figures, the duty per stamp per twenty-four hours with equal weights of stamps was higher in the case where 54·3% of the material remained on a 1⅜ in. screen than in the case where only 1·1% of the material remained on the same screen. This result seemed most extraordinary, and in order to show how remarkable it was he herewith appended some results obtained from a Rand mill by virtue of the interpolation of a fine-grinding plant between the coarse crushers and the battery.

The primary object of the installation of the fine-crushing plant on this mine was to permit of the mill being overhauled, 20 stamps at a time, without interfering seriously with the output per month. Two small jaw-crushers, electrically driven and designed to reduce the ore from 2½ in. to 1½ in., were installed. It became quite evident, after the first few days' work, that the introduction of this plant would have the desired effect. The plates and launders banked up with the additional material, and it was found necessary to make considerable alterations to the mill water service before the increased quantity of ore crushed in the mill could be properly handled. Without the fine breakers the mill duty on 400-mesh screens averaged under 5 tons per stamp per day. After the installation of the fine breakers the duty increased to 5·6 tons per stamp per day, equivalent to a 12% increase. The figures were calculated very carefully and might be taken as accurate. The average cost of milling under ordinary conditions over a period amounted to 2s. 11·657d. With the fine breaker plant the average was 2s. 11·177d., being a reduction of 0·48d.

per ton milled. The objects of the installation of the plant were achieved most successfully, and the whole mill was rehabilitated without prejudice to the monthly returns.

These figures were all carefully compiled and the results were so satisfactory as to leave no room for doubt regarding the efficacy of fine crushing when used in conjunction with light stamps. It might be argued, however, that the ore was in any case being fed in too coarse a condition to the mill, and he thought that when Mr. Caldecott said "hence the heavier the stamp the coarser the preliminary breaking admissible, and *vice versa*," he had hit on the real truth. *Vice versa* would then read: *the lighter the stamps the finer the preliminary breaking admissible.*

To follow this statement to its logical deduction, one found then that as Mr. Caldecott believed in the use of fine crushers with a light stamp, he must further believe that the use of the heavy stamp would do the same work that the crushers were designed to do for the lighter stamp, more economically. It appeared to the speaker that a great deal of the problem lay in this point, and personally he had the feeling that the limitation of the weight of the stamp would be met at the line of demarcation between the crusher and the stamp mill principles. It was difficult to imagine that ore could be reduced from 2½ in. to 1½ in. as efficiently and economically in a stamp mill as in a jaw or toggle breaker, because in the one instance one had anything but a positive discharge and positive feed, whilst in the other, one had both of these points absolutely defined, and furthermore, the power consumption in the case of the crusher was in direct ratio to the work performed, whilst in the case of the gravity stamp it was not necessarily so.

He made special reference to this matter because a reader of Mr. Caldecott's remarks might unwittingly get the impression that Mr. Caldecott intended to say that preliminary fine crushing was in no case necessary; but this was not so, and as there are many remote places where moderately light stamps would still be used on account of great decrease, the value of a preliminary fine crushing should not be lost sight of, even from this point of view.

Mr. Caldecott made no reference to the high-speed stamps of the Morrison type installed and run on the Meyer and Charlton mine for many months in the year 1903. These stamps had a weight of over 1,600 lb., and, owing to their being operated by a crank shaft, it was possible to run them at as high as 135 blows, and they registered a duty of over nine tons per stamp per day for a considerable period.

It was quite clear that the increased weight was resulting in a more than proportionate increased duty, but there were other considerations that prevented one from arriving at any settled conclusion in the matter.

Mr. Caldecott said "it will be evident that the day of the stamp as a unit of crushing capacity has passed away," and in this everybody will agree with him for the reasons he had stated. Whether or not the principle of the gravity stamp would ever be discarded universally, was doubtful, but he, the speaker, did know that many attempts had been made in small plants to discard it, with the greatest success; that is to say, there were numbers of plants in the United States to-day in which the stamp mill found no place. The principle followed was coarse crushing, fine crushing, edge runners and tube mills, and, when considered from the all-important aspects of capital outlay and operating costs, it had been proved in more than one case that the stamp mill was not absolutely indispensable. So far, however, the results had been obtained only from small plants, and he knew of no case in which the engineer had the temerity to lay down a large capacity plant without the gravity stamp.

In conclusion, he thought that Mr. Caldecott, now that he had started the ventilation of this important matter, would do a very great service if he continued the subject in some of those more abstruse directions which the speaker indicated in his remarks, and as Mr. Caldecott was well placed to carry out the necessary investigation work, it was just as fit that he should continue the good work begun.

He, Mr. Denny, would suggest, however, if the result of increasing the weight of stamps were merely to do the work which would be done by the crusher in conjunction with the lighter stamp—and this was what in effect Mr. Caldecott claimed—then the matter required very grave consideration before they went further; for he, personally, was not fully satisfied as to the wisdom of such a step, and until they had further details he did not consider that they should be justified in assuming that the heavy stamp was an unmitigated good.

Mr. Hugh F. Marriott said that it must be noted that the paper was really a record of the practice of one of the largest companies on the Rand, and did not include, especially so far as the history was concerned, a great deal of what had gone on concurrently amongst other companies. It must be seen now that the race for heavy stamps had begun in good earnest.

It would go on until the component parts of the machinery would not hold together any more. Since the author had written his paper, stamps had been designed up to 2,000 lb. in weight, and there was a a still further variation from what was shown on the diagram. The concrete, which had already replaced the wooden mortar blocks, was creeping up the superstructure until it had got half-way up the king-posts, and doubtless, before they had finished, wood would be eliminated altogether.

It was evident that stamp duty alone conveyed but little meaning to-day. It had already been stated by previous speakers that the West Rand Consolidated had crushed 15 tons per stamp per day, and in one case 17 tons had been crushed. It was not stated what mesh was used, but it was doubtless a very large mesh indeed.

This method of increasing the output brought to his mind a story which went round South Africa during the war. At a certain military hospital there was an Irish doctor, who was told by his superior officer that all the water given to the patients had to go through a Berkefeld filter. As this had only a limited capacity, which was not great enough, they did not know what to do. The doctor at length put a brawny native on to the filter, who worked at it till he blew the bottom of the candle out, and then they got as much water as they wanted, and also satisfied the authorities.

That was really what the stamp duty amounted to to-day. A screen with ten holes to the linear inch would give a certain amount of tonnage, and if the mesh were enlarged to three holes to the inch, as he understood had been done, a great deal more would go through without there being any real increase in efficiency.

There was only one way in which a true comparison could be made in the future, now that they had tube mills, and that was to consider the stamps and the tube-mills combined as one unit. The power required, the tonnage put through and the size of mesh to which the pulp was reduced, could then each be taken to demonstrate the standard of efficiency attained, provided that two out of the three were reduced to a fixed constant.

A point of interest was mentioned by the author on p. 14 with regard to the screen, because screening played a very important part, in fact the most important part, in all those discussions as to what stamps would do. The author coupled the work of that Institution with that of the Chemical, Metallurgical and Mining Society of South Africa, which had elaborated another system for the standardisation of screens.

It seemed to him a pity that the local society there had not yet recognised the decided superiority and simplicity of the standards that the Institution of Mining and Metallurgy had brought out. To compare the two types, the difference between them was that the South African Society measured with a micrometer the size of the hole in the screen, and reached their minimum of 50 holes to the inch, presumably the limit of accurate reading power.

The Institution of Mining and Metallurgy took a standard type of screen in which the diameter of the wire was equivalent to the linear width of the hole, *i.e.*, the open spaces totalled 25% of the entire screen surface. By that means the standard screen could be carried down to any degree of fineness possible in commercial screen manufacture. They were already made down to 150-mesh, and the 200-mesh standard would shortly be issued.

Now the South African Society, stopping at 50-mesh cut out altogether all the work done on the cyanide side of the question: the finest sands and slimes could not be measured by their standard at all. He therefore failed to see why they had adopted it, because the most important part of their work could not be measured by it.

The author also mentioned that by their (the South African Society's) system of screening, they could get a duplication of previous orders from the screen maker. This meant that the screen maker would be bound, within certain limits of error, to manufacture the size required to pass the micrometer test. By the Institution's system, which was simplicity itself, it was open to purchase any good screening in the market, and then to test it against a standard set of sieves in order to obtain its standardisation number It did not matter what size it was called by the maker. For the purposes of the user the standardisation number was all that was required. Therefore, screening could naturally be bought in a much cheaper market than if the method adopted by the South African Society were followed.

Mr. Walter McDermott said it struck him that there was not sufficient information given in the paper as to grading tests below 60-mesh, as this was necessary in comparing the efficiency of different crushing operations. The pulp which passed from a set of rolls through a screen of given mesh, was very different from the discharge of a stamp battery through the same screen. The paper had relation throughout to crushing for cyanide treatment; and therefore slime production and fine grinding of the sulphides was important: the stamp was to be followed by a tube mill.

It was interesting to see figures of careful tests given against two ancient fallacies: fallacies which many men, in many countries, for many years, had exposed; but which kept coming up fresh at intervals in different places. He referred to fine breaking before stamps, and increased screen surface. It has occurred to large numbers of men that it would be an advantage to lessen the work of stamps by breaking finer; but the practical experience of all the world is against it as a general proposition. So in regard to screen surface: it is obvious, it is scientific, and it is wrong, that this is of primary importance.

He remembered fully twenty-five years ago seeing an improved stamp mill in Colorado with double screens, of corrugated form, concentric to the dies, which quickly destroyed all illusions as to increased capacity expected. The very same idea was patented in the Transvaal ten years later, tried and abandoned. Over and over again men have built double discharge mortars; and after trial have blocked up one side, and returned to single discharge. The mere fact that the same mortar with the same screen will crush 10 tons per day of one ore, and 40 tons of another, shows that screen area is not the controlling factor which many imagine it to be.

The success of the heavy stamp seems to be more a result of the introduction of grinding after stamps than of any special application lately of science to ore crushing. Steam stamps, so largely used for over twenty years on Lake Superior, were tried in Dakota on gold ores, and in Montana on copper ores, and given up. High-speed pneumatic stamps were used on Lake Superior over twenty-five years ago. The utility of heavy stamps had increased with the use of coarser screens; that is, the stamp was only doing part of the work of fine crushing.

He did not agree with the conclusion that the day of the stamp mill, as a unit of crushing capacity, was past. It might be so in Johannesburg, from conditions affecting finances and ore treatment, but such conditions varied greatly over the world. In Australia, for much grinding work, pans had displaced tube-mills. Water classification and tube mills meant very fine grinding of the sulphides, desirable for cyaniding, but undesirable for concentration, as almost universally employed, for instance, in California.

He would have liked to see the screen sizes in the paper expressed in terms of the I.M.M. standard, and grading tests on the same standard given on the minus-60 products.

Mr. R. E. Commans said that he was pleased to hear Mr. Truscott's remarks, because he thought it very desirable to emphasise the fact that it was not increased efficiency that was gained by increasing the weight of the stamps, but economies in the erection and running of a mill caused by a reduction in the number of units.

He felt that, with a weight of 2,000 lb., the limit of cam-lifted stamps had almost been reached, and that, if the weight was to be further increased, some other form of lifting device would have to be employed. As stated in the paper, with a drop of 7 to 8 in., 100 drops a minute was the maximum number practically obtainable by cam-lifted stamps, and he was rather surprised that the author had not made any reference to the Morrison high-speed stamp, to which Mr. Denny had just referred, and which was installed and had been running continuously for over two years at the Meyer and Charlton mine.

These stamps were no freak, or the design of a man with no knowledge of the special requirements of a stamp mill, but were the outcome of a long series of trials and experiments on a large scale by Mr. Morrison, one of the leading mechanical engineers in this country, and Mr. Bremner, who had had many years' practical experience of the running of stamp mills on the Rand.

That more had not been heard of these stamps was, he considered, due to an unfortunate combination of circumstances, and not to any defect in the design and working of the stamps themselves, and he sincerely hoped that more and heavier stamps of this type might soon be installed by other companies. Mr. G. A. Denny, who was for many years manager of the Meyer and Charlton mine, stated that when some small improvements, which experience in running the trial 5-head battery had suggested, had been made, he was satisfied these stamps would more than favourably compare with cam stamps, and in his paper* on 'The Design and Working of Gold Milling Equipments,' spoke most highly of them.

The main object of the Morrison stamp was not to accelerate the fall of the stamp, but to obtain a greater number of falls by securing a quicker pick-up of the stamps. The distinguishing feature of the high-speed stamp was the lifting device, which enabled the number of drops to be increased some 30% to 35%, and which consisted of a hydro-pneumatic cushioning cylinder lifted by a crank and connecting rod. The stamps fall freely by gravity alone, and therefore, so far as the blow on the ore was concerned, it no way differed from a cam-lifted gravitation stamp of the ordinary type.

At the Meyer and Charlton, double the amount of ore was crushed per head by the high-speed stamps. The weight of the Morrison stamp was 1,600 lb., and the cam stamp 1,100 lb., and if he remembered rightly, the high-speed stamps crushed 9 to 10 tons, and the cam stamps 4·7 tons per head per day with an 8-inch drop, through 500-mesh screens, the horsepower per ton crushed being about the same in both cases.

Pneumatic stamps had been designed to increase the number of drops per minute, as well as the energy of the blow, but they had not come into general use, and required a large piston area to lift a heavy stamp, which makes them rather cumbrous.

With regard to the author's remark as to the need of some effective means of testing abrasion hardness, he thought that this might be done by means of the sand blast. The ordinary tests for hardness by scratching by the rebound of a small pointed steel cylinder, or the indentation of a polished specimen of the metal by the pressure of a hardened steel ball, were theoretically of interest, but were of no practical use in deciding the hardness and suitability of steel for ore crushing.

Mr. **Alfred James** said that possibly the greatest point in the paper was that it seemed to mark the passing of the stamp mill as a fine crusher. Only three or four years ago they were accustomed to hear of screens with 1,600 holes to the sq. in., they were told that they were now crushing through a 3-mesh, or nine holes to the sq. in. In fact, one gentleman had told him that he was now passing 1½ in. cubes through his tube mills. So that it seemed that the stamp was now taking much the place that Mr. Denny was proposing to take with his rolls, that of coarse crushing.

He need scarcely say how much they were indebted to the author for his valuable and classic research, which would be useful not alone for the particular type of ore on which he had conducted his experiment. The title of the paper was 'The Development of Heavy Gravitation Stamps.' Perhaps it would be more accurately stated as 'The Development of Heavy Gravitation Stamps on the crushing of Rand Ores,' because he was sure that the author would not desire to use very heavy stamps to crush light porous ores.

One point which struck him, and which had been unnoticed, was where the author endeavoured to give the crushing efficiency of the blows, on page 10. He thought the author had omitted to take into account in his mathematics, the fact that the crushing power of the stamp, crushing a 2 in. cube of quartz, was not the duty of a stamp striking through 1½ in., because he thought he might support his 2 in. cube of quartz under that weight of 2,190 lb. per sq. in. for a long period without crushing it.

But if they took it that the elastic limit of that cube was one-twentieth of an inch that was, if they pressed it to one-twentieth of an inch it then cracked, then they had the duty of that stamp expending itself in one-twentieth of an inch instead of 1½ in., and therefore the force would be approximately 65,700 lb. instead of the 2,190 lb. per sq. in. which the author gave. It was a very moot point as to whether the striking (not the pounding) duty of the stamp was not done in less than one-twentieth of an inch. He thought they might take it that 2,190 lb. was not a correct statement of the force of the actual blow on the cube.

He would like to refer also to the statement of the author as to the remarkable efficiency that had been attained by the adoption of the new method of crushing. If they looked at the work for May, 1907, of the Simmer East, it would be found that whereas formerly they crushed, with 250 stamps with 8 to 1 of water, 5 tons per diem containing 11% of + 60, they now crushed, with only 6½ to 1 of water, no less than 8½ tons per diem, with only 1·61% of + 60, for exactly the same cost. So that they were getting a higher tonnage and a higher efficiency, as shown by the finer crushed material, at the same cost. That, he thought, was an advance of which the pioneers of that new method of crushing on the Rand might well be proud.

Mr. G. H. J. **Hooghwinkel** wished to draw attention to two points. First, with the increased weight of the stamps, the number of stamps on each cam shaft would probably decrease. The second consideration was the variation in the speed of the engine. Both would be met in the future by electrically driven stamps. The three-phase motor had a constant speed, and was therefore well adapted for work of that class, and electrical driving was also easy to apply to a sub-division of the cam shafts by means of several motors.

He had seen a new stamp mill in Cobalt about a fortnight previously on one of the larger mines, and they had their cam shafts, with five stamps on each shaft, driven by an independent motor. The weight of each stamp was about 2,000 lb., and the speed was 100.

Although he did not know the exact results, they were using hard metal there, and it would be noticed that the running was much smoother than with a steam engine. If the weights were still further increased, the probable solution would be found in a combination of the electric driving and pneumatic methods of lifting stamps, just as was being done with the Temple drill. An electric motor would probably drive the compressors for each unit, in which compressed air would be used as lifting power, and they would therefore get the advantages of the concentration of the power in the power station and also the advantages of the pneumatic lifting power to help, if still heavier stamps were to be used.

Mr. W. **Fischer Wilkinson** said that the part of of the paper which had especially interested him was the comparison which the author had made between crushing with stamps alone and double-stage crushing with stamps and tube mills. He remembered when the Knights Deep and Simmer East joint plant was put up, one-half of the plant was devoted to stamps alone and the other half to stamps and tube-mills, and that it was very difficult at that time to say which was the better practice.

---

*Minutes of Proceedings Inst, C.E., vol. 166, 1905-6.

It was interesting, therefore, to find now that the author approved of double stage crushing, because for the same cost he obtained a finer product, and therefore, of course, a better extraction. Now the question as to which was the better really depended very much on a correct estimation of the cost of crushing, and he would like to ask the author whether interest on the capital expenditure of the different plants was included in these costs or not, because it was certainly very important that that should be considered in making a comparison between the two systems of crushing. One might get a higher extraction by spending more money on plant, but the interest on this additional expenditure might exceed the benefit gained by increased extraction.

Another smaller point in connection with costs to which he wished to draw attention was that the costs of the year 1907 were compared with those of 1909. It seemed to him that it was quite possible that in the interval between 1907 and 1909 the conditions on the Rand had changed, and therefore that the costs of 1907, in order to make a fair comparison with those of 1909, ought really to be adjusted to the new conditions. Labour and supplies were, he believed, considerably cheaper now than they were two years ago, and if that were so, Mr. Caldecott was doing an injustice to the single-stage crushing system of 1907.

The popular practice now on the Rand was stage crushing, crushing with stamps and tube-mills, but it must be remembered that there were engineers on the Rand who did not accept that as the best system. Mr. E. J. Way, for instance, the consulting engineer of the New Kleinfontein, was, last year at all events, in favour of single crushing, when one did not want to get a product finer than one containing from 10% to 14% coarser than +60. Under such conditions Mr. Way maintained that the best and cheapest practice was to use fine screens in the mortar-box and do without tube-mills, though, if finer grinding was required, he thought a combination of stamps and tubes or similar grinding machines would perhaps be right.

After studying Mr. Way's views, he (Mr. Wilkinson) thought he would examine some of the returns from the New Kleinfontein, which was now milling on the single-stage system, and compare them with the costs of other mines on the Rand using tube-mills. He had looked up the last report of the New Kleinfontein and found that the reduction costs, which included all crushing and cyaniding, amounted to 3s. 3d. per ton milled, and that they obtained an extraction of 94·95%.

Then he looked up the costs of a neighbouring mine, the New Modderfontein, which was using tube-mills, and found that their reduction costs were just under 5s. and their extraction 97·4%; in other words their costs were considerably higher than those of the New Kleinfontein, but they had a better extraction equivalent in value to about 9d. per ton milled. The net result between those two systems was that the New Kleinfontein system of single crushing by stamps alone without tube-mills was about 1s. a ton cheaper than that of the neighbouring mine using tube-mills. The returns of several other mines using tube-mills that he had examined gave reduction costs of from 4s. to 5s. per ton with extractions of from 93% to 94%.

He was not prepared to say that there might not be some special reasons for the low costs at the New Kleinfontein, and he merely put these figures forward as an argument that they ought not to accept the coarse screen and tube-mill practice as the best without careful consideration. In comparing costs due regard must be paid to the interest on capital expenditure as well as to the percentage of extraction recovered.

Mr. Hugh F. Marriott said he might point out that the New Kleinfontein mill was burnt by the Boers during the war, but they had saved Modderfontein.

The President said he hoped that those who held views upon the matter and had information, but who had not spoken that evening, would let the Institution have the benefit of their views by correspondence, as it was a subject to which much importance attached.

CONTRIBUTED REMARKS.

Mr. Wm. H. Shockley: A point that does not seem to have been brought out during the discussion of this paper—though very probably familiar to most of the members—is that a decrease in the weight of a stamp may greatly increase the crushing capacity, even though all the other conditions remain unchanged. The first man to prove this to me was my mill superintendent, George Fleming, when he was in charge of the 10-stamp dry-crushing mill of the Mount Diablo Mining and Milling Co., at Sodaville, Nevada, about 1890. At that date we were trying to get through as much ore as possible, and had increased the weight of our stamps to 1,250 lb. by the use of top tappets.

Fleming noticed that the replacing of the old shoes by new always caused a greater increase in the crushing than the difference in weight seemed to justify, and suggested that we should true up the irregularly-worn old shoes and try them in place of new shoes. We tried this and found that the old shoes, after being cut off in a lathe so that their bottoms were flat, crushed almost as much as did the new shoes. After this we faced the shoes about every two weeks, and greatly increased our output by this simple and inexpensive change. We estimated the loss in crushing due to cupped shoes at nearly 20%. Our maximum output with this mill was nearly 1,350 tons per month, crushing through a 30-mesh brass wire screen.

Mr. W. Truran: I should like to endorse the remarks made by Mr. Walter McDermott—that it is not possible to make a broad statement as to any one class of crusher being the most efficient, so much depends on condition. What gives most satisfactory results in South Africa, might turn out to be a dismal failure in South America, for instance.

As an example of this I should like to mention the pneumatic stamp. This machine, as I think Mr. Commans pointed out, is by no means a new idea. I believe I am correct in stating that the pneumatic stamp generally in use now was the invention of the late William Husband, of the firm of Harvey & Co., Ltd., of Hayle, Cornwall; and at the Inventions Exhibition, held at South Kensington in (I think) the year 1886, one of these machines was exhibited, of which exhibit I was in charge. The first machine made had an oscillating cylinder, which has now been done away with.

These machines were, I think, first used abroad in any practical manner by the late William Smeddle, at the Babilonia Mine, in the Libertad District, Nicuragua, Central America. Some few years since

I was in charge of this mine, and the crushing machinery consisted of a pneumatic stamp; and I am bound to state that it was a source of endless trouble.

The complicated and delicate nature of the stamp necessitated constant supervision and very careful handling, and this in a country where no mechanics are available was the cause of a great loss of time and considerable worry. For the efficient working of this stamp it is absolutely essential that all working parts should be kept vertically and horizontally true, and in order to maintain this condition no alterations or repairs could ever be entrusted to a native, or even a white man who was not a trained mechanic. The packing rings in the cylinder and the side arms needed very careful attention, for once the stamp got the least bit out of truth, trouble and delay ensued.

In the case of repairs or replacement of wearing parts, work was greatly hampered by the cramped construction and lack of room in the machine, and around the frame, the enormous weight of the stem head, etc., and the number of disconnections necessary, all of which matters rendered the work a matter requiring considerable thought and care.

The stamp we had was, I must admit, not of the latest pattern, as no lubricators were fixed on the cylinder, and in order to lubricate the interior of the cylinder, melted pork fat had to be blown in through the air-holes in the cylinder by means of a kind of pea-shooter and in most cases the stamp *bleu first.* In conjunction with amalgamation, I do not consider the pneumatic stamp a success, the plate surface being altogether insufficient for the capacity of the machine. In places such as Cornwall, and as a crushing machine where skilled labour is ready to hand, it is quite another matter, and the pneumatic stamp, I can well believe, is a most satisfactory and efficient machine."—FISCHER WILKINSON AND OTHERS.—*Bulletin of the Institution of Mining and Metallurgy.*—Nov. 11, 1909, pp. 9-28. (H. A. W.)

PRACTICAL NOTES ON DRY CRUSHING MILLS (W. Australia) —"*Milling.*—Three No. 5 Krupp ball-mills, running at 26 rev. per minute, and taking from 16 to 20 h.p. each, crush the ore. They are fed by a shaking feeder worked by a single armed cam on a countershaft, and giving up to ¾-in. stroke. There is nothing like a steady feed for a ball-mill, and this feeder does it well. Each mill is loaded with 2,350 lb. of steel balls, and will crush 40 tons daily through a 27-mesh screen. I have had experience with dry crushing stamps, rolls, and studied the Griffin mill, but the Krupp mill is undoubtedly the best dry crusher when capacity, wear and tear, power consumed, and ease in getting at parts are taken into consideration. Using Krupp steel liners, a mill will run for eight to ten months without renewals, excepting, of course, such minor repairs as broken bolts, bent scoop plates, and riddles, a plate off, and fine screens. English and German steel plates have been tried, but the latter are the best by far. We have a mill newly lined with Melbourne made liners, and they appear to be doing well. Four grinding plates are bolted to each of the ten mantle plates, also the long perforated plate. These are got ready on a spare set of mantles beforehand, and in twelve hours six men can strip a worn-out mill of outside casing, screens, mantles, and side liners, and then practically renew the mill. The steel consumption in balls is 3½ oz. per ton crushed. Three are added each week. A remarkable point about the balls is that, on emptying a mill, seldom or never does one find a ball smaller than 1½-in. diameter; I have three specimens of ¾-in. balls, and these are the only ones found during the last three years in our mill. What becomes of the smaller balls of under 1½ ins.? Are they smashed up? And yet there are no traces of fragments. It is rather a mystery. Once daily—*i.e.*, providing the ore is dry—the fine screens are brushed with a wire brush, but oftener if damp ore is fed in. About 3-in. size is the right feed for a ball mill, and if the ore is very fine and soft the crushing is not so good, as the balls have a tendency to "bed" in the fine material. My opinion about the crushing is that a certain amount of ore is broken by the balls falling from one plate to another in the mill; but a great deal of grinding goes on by rubbing and rolling, as every stone taken out of a mill is quite smooth and rounded. The working of a ball-mill is entirely by sound, and a poor millman can easily cause a lot of trouble and lose tonnage by neglecting the feed. All bearings are lubricated by a light grease, and give no trouble. The spur wheels should run three years at least, the pinions about half that time. A strong fan should be running in every dry crushing mill to draw off the dust from the ball-mill; and the suction has a tendency to keep the fine screens clear. Dynamite fed into Krupp mills with the broken ore is apparently ground up, as it seldom explodes, and then only a fine screen might be blown out. There is so much room in the mill when anything explodes; far different to the Griffin mill, which generally has the bottom blown out. Milling costs 1s. 8½d. per ton.

Grinding.—Much controversy has passed about the merits of pans and tube mills, but in all roasting plants here the pan is in use, while in the wet mills the pans and tubes are generally found together. In a dry crushing mill, the pan serves a double purpose —*i.e.*, as a grinder and amalgamator. It is a very efficient machine in both ways, as a ball mill product of which 40 per cent. will pass a 150-mesh screen, 90 per cent. will pass the same screen after passing the pan; and by feeding in the necessary Hg every day the coarse gold will be caught before the slime goes to the cyanide plant. The pan is an interesting machine. A 5-ft. pan running at 47 revolutions per minute absorbs about 5 h.-p.—*i.e.*, a pan with heavy shoes. Some people prefer each pan driven separately by belt and two pulleys, but the friction clutch is very satisfactory. Several pans may be driven by one belt, and little inconvenience is experienced in this connexion; whereas a lot of money is represented in, say, 16 belts driving 16 pans. The muller plate is attached to the yoke by means of a bridle, permitting the former to swing with ease. Eighteen hematite iron shoes are pegged on to the plate, and there are a like number of dies fitting into the bottom of the pan. The yoke and plate may be raised and lowered whilst in motion by the usual screw and hand-wheel. We find it beneficial to clean each pan twice a month, as the lime in our ore is deposited as a hard scale in all the spaces between shoes, dies, &c., and on the sides. A set of shoes and dies lasts from four to six months on roasted ore, and when half worn down a compensating weight of some 600 lb. is put on top of the plate. A pan can be cleaned under two hours, and this includes chipping the above-mentioned scale. A pan can be cleaned and newly shod in about four hours by two men. In many mills the sides of the pan are badly scoured by the sand, but after 5½ years' run ours are as good as ever. A great deal of the good working of a pan depends on the man in charge, overloading meaning a big discharge of sand, and too little feed just waste of power and material. We don't separate our product before grinding, as is usually done, and experiments have proved just as good work by passing all the pulp through the pans. They grind somewhere about 10 tons each per day, at a cost of 1s. 7·8d. per ton milled. I don't see much advantage in the central feed, and the side feed is very satisfactory; in fact. it is almost universal.

Settling the Pulp.—A plain long **V** box is a nuisance if not divided up by partition plates, the single **V** settlers, say, 6 ft. square at top and 7 ft. deep, with a tap at the bottom being the best. The latest settler in this centre has a plug in the bottom worked by short lever, while on one mine the settlers are worked by a rod and plug worked from an eccentric on a slow running shaft above. Taps at the best are rather a bother. The first row of settlers gets the fine heavy sand, and this should be returned for regrinding, or, if not, then the taps on this row should be slightly opened all the time to permit the stuff to escape to the agitators. Every two hours will suffice to run out the thick slime from the other settlers, but, of course, this just depends on the feed and settling capacity. It is necessary to rake down the slime on the sides of the settlers once a shift, or else it will set hard. The slime runs to the agitators by means of a launder, which is best if made of wood; iron launders soon scour and rot away.

Agitating.—Now-a-days we have the Brown tank, the open monteju, the A. Z., and the ordinary type of agitator. The latter is a tank, 22 ft. by 6 ft., with gear driven spindle and arms attached, running at 5 to 8 revolutions per minute, absorbing 2 to 3 h.-p., and taking 12 to 20 hours to treat a charge of pulp and is commonly in use. This agitator has its weak points, and will soon be superseded by others. The worm wheel and friction clutch is not so satisfactory as the bevel gearing, but still such gear runs for years without renewal. We run the slime into the agitators with about 1—1 consistency, and as it is running in lumps of KCN are put into the launder, thus the strength of the solution, ·08%, is in contact with the gold the moment it enters the vat. It is customary in large mills to have a lot of strong solution in a special tank, and run it in with the pulp as required, but the way mentioned is just as good, if not better. We find a good deal of concentration going on in the bottom of the agitators, the arms acting somewhat as a buddle; the heavy sand contains a little Hg from the pans, and has assayed up to £20 per ton. This stuff we re-grind and re-treat as usual. Our KCN consumption runs 1 lb. per ton of dry slime, and the costs of agitation is 1s. 4·7d. per ton milled.

Filter-pressing.—Many will have it that the filter-press is doomed to make way for the vacuum process; but not on this field in the treatment of fresh mill slime for years to come I am thinking. In a well-arranged plant, agitation, filter-pressing, and disposal of residue costs 2s. 8d. per ton, and the figures given in vacuum treatment for old dumps locally are about 2s. 6d. per ton, yet the question arises, will the vacuum process deal successfully with current mill slime? There doesn't appear to be much reason against this, but nobody is inclined to rush into change, and it would take much experimenting to prove that it would pay to discard the press and install the vacuum plant, although it is well known that the Ridgeway plant on the Boulder is doing this with good results; one mine here is now conducting tests in this connexion.

The filter-press is a costly machine, requires a lot of attention, has its weak points, but yet does good work, and its value in connexion with treatment in Kalgoorlie has been incalculable; in fact, it saved the mills from enormous trouble. The ordinary press made by Dehne-Martin has 50 frames for the slime cake, and a like number of solid plates covered with filtering cloth, which costs 1s. 9d. per yard. Only in large plants is the hydraulic closing apparatus in use; in fact, one mine with 20 presses has not got it working, the majority being screwed up by back nuts and hand wheel. To the layman the flow of pulp, solution, and air in the press is somewhat puzzling, but there is really nothing in it, as the various channels are easily traced. Each plate is provided with a tap over the launder, but these are a great bother, and are now discarded, the vent being plugged. A pair of men on contract can easily dump eleven presses per shift, and keep an eye on the cloths, &c., filling presses by monteju is out of date. We fill with a 3-throw pump with 12 by 10 in. plungers, and running at 20 revolutions per minute, the time taken being about 12 min. with pulp 1 to 1 thickness. This pump could deal with five presses at a time. At 60 lb. pressure a safety valve blows off so that no accident can occur. The pump gives little trouble. The plungers are of cast iron, and last about twelve months, while with a clean water jet greasy hemp packing lasts some time. The rich KCN solution from filling the presses runs to a special tank, through a clarifier, then to the extractor boxes. With a similar pump running at 13 revolutions per minute, the slime is washed for 30 minutes at 90 lb. pressure. The wash being shut off, compressed air at 90 lb. pressure is admitted to dry the cakes. This takes about 5 minutes, the slime then holding 20% moisture. From time to time plates are found to be broken in the presses. This is most likely caused during the filling when one frame is filled faster than the next one, through the channel being choked; also a shock, such as turning on the wash quickly, may possibly do it. These plates cost from £3 to £5 each. A badly filled press will not wash properly, and the solution follows the least line of resistance. In sampling, take a section of a cake about every tenth frame. Pressing costs 1s. 1d. per ton.

Disposal of Residues.—Under the line of presses is a horizontal belt, feeding an inclined belt elevator at 26°, in turn feeding a boom distributor. The main belts travelling at 300 ft. per minute give little trouble, yet at times they take fits and run to one side and throw off the slime where not wanted; even the wind affects them. The boom belt running at 950 ft. per minute throws the slime far out over the dump. This is driven by a 10 h.p. 3-phase motor running at 300 revolutions, and driving the boom by a short belt without gearings, and the way in which this machine behaves in heat and cold, bad dust and rain is really astonishing. In dumping 120 tons daily, 40 units at 3d. are used. To minimise the dust on the dump we hose it with salt water, which forms a crust on top. A belt may run for a day or two without trouble, then you may have days of constant attention needed. The canvas composition belts are greatly affected by heat and cold, the rubber belts stretch a good deal, and joints need much watching. The steel lace makes a good joint, but it wears out quickly, and if one side should wear through and catch anywhere when working it is likely that the belt will be ripped from end to end; all classes of belts will do this. To get over this trouble we make new joints at regular intervals, and also wrap some tough cloth over the ends of the belt at the joint, and the whole laced together as usual.

Canvas belt costs 7s. per foot, and the rubber from 10s. to £1 per foot. The hand type tightener is of little use, the automatic sliding weight being very good. Belt conveyors are easy to erect, but it is advisable to have plenty of attention devoted to them when running.

Precipitation and Clean-up.—Of course this is a big subject, yet my experience has always been without bother in this line. Good roasting in the mill, a little lead acetate added at the pans, a fairly strong solution in treatment, a trickle of extra strong solution as the rich solution enters the boxes, and a dose of acetate every other day or so, the cells kept well filled with zinc, and all is well. Occasionally a little lime comes down in the top boxes, and the acetate does not affect the bullion to any extent. In cleaning-up it is quite a matter of judgment as to how much zinc is to be put into the acid direct and how much to be washed. Acid of about 5 water to 1 acid is strong enough. The lead-lined tank should be connected with a fan to carry off the fumes. Heat is not necessary, and three hours in the acid is quite sufficient. From the tank the sludge should run into the monteju, and then into a lead-lined centre filling press. An hour's wash will do, and hot water is beneficial. Twelve to eighteen hours drying, roasting, and a good stirring at the end is all the better. About 50% borax and a little sand are sufficient fluxes. The tilting furnace is very handy for large lots, and the pot should last three or four smelts. A flat topped furnace is good to do the final refining in, as one can stand right over it and skim off base, etc. In sampling bars a drill bore in about ¼ in. at top and bottom at opposite ends is satisfactory enough.

Conveyors and Elevators.—The Archimedian type of screw conveyor works well on fine ore from the mills, and being enclosed makes no dust. The push conveyor also works well on this ore and that from the furnaces, but makes much dust. The double push conveyor will take along the feed for the roasters, and return any excess not required. The carriers are of the frictionless style, and the flights may be taken out or replaced while in motion. In roasting, the roast may be poor at times, but by the time it has reached the end of a conveyor it will be much better, due no doubt to the great rabbling, pushing, and exposure to the air. The power used varies a good deal according to the feed. For cold dry crushed ore the rubber belt with steel buckets is very serviceable. The belt is butt jointed, and the buckets placed 15 in., staggered, and is examined once a month. In elevating hot ore, the chain and bucket working in a steel housing is very satisfactory, though at the best of times it requires much attention in greasing, etc., and breakdowns are accompanied by much heat and dust. I hear lots of complaints about the elevators, but my experience with them is on the good side. The chain must be kept reasonably tight by the gear provided at the top. The wear and tear is remarkably small, and recently we renewed the bottom shafting after five and a half years' run. The elevator is examined about twice a year.

Driving Belts.—There are so many makes in use, and most of them of a good character, but of all the rubber, raw hide, leather, composition, and balata, I prefer the latter. It is made of the best cotton duck impregnated with balata, is expensive, stretches very little, and lasts a long time. If not run carefully it will open out, but the inside ply can be torn out if it gets too rough, although this takes some of its strength away. Where there is much heat it is not advisable to use it. Many millmen believe in butt joints made by steel laces or clips of various makes. I don't. The lap joint properly made is very strong, and the ends of the belt do not tear out as happens with the butt. Even in driving a dynamo, or a motor driving anything by belt, the lap joint makes no bump if planed down and laced neatly, although this is contrary to general practice. When a belt is stretched the laces may be cut out and the lap rivetted with copper rivets. Large belts are generally laced in place by means of clamps and rods. A little castor oil, which makes a good dressing for belts, is necessary in a dry crushing mill. In rope driving, a 2 in. cotton rope with a long splice will run for years without trouble, and little dressing is required.

In conclusion, it might be said that although a wet crushing mill is the healthiest and easiest to keep clean, yet a dry plant can be kept fairly free from dust by having fans, all crushing machines and conveyors well housed, plenty of windows and doors, and open roof where possible, but, of course, the machinery in the latter cannot be expected to last as long as in the former.—W. M. VON BERNEWITZ. *London Mining Journal*, Sept. 4, 1909, p. 299. (A. R.)

PURIFICATION OF MERCURY.—"The constant use of the mercury cathode in this laboratory necessitates the frequent purification of very impure mercury. According to the methods described in the text-books a great deal of time is required if anything like complete purification is desired. Furthermore, little is known of the relative efficiency of the methods, so that the worker simply chooses the one that strikes his fancy. It therefore becomes desirable to know how to obtain the purest mercury in the least time and with simple apparatus.

The usual methods are, first, to draw air through the metal contained in a wide inclined tube bent up at the end; second, to shake with mercurous nitrate, ferric chloride or potassium dichromate, in a separatory funnel; third, the method of L. Meyer,[*] to let fall in a fine stream through a long column of mercurous nitrate; fourth, to purify electrolytically by making the mercury the anode in a nitric acid solution;[†] and fifth, to distil. No one who has tried the first of these will think very highly of its speed or efficiency. The shaking in a separatory funnel requires no construction of apparatus and is probably the favourite method. The surface of contact between the metal and the solution is, however, relatively small and long shaking is obviously required to dissolve the impurities present in the mercury. Washing through a column of mercurous nitrate containing free nitric acid is excessively tedious, provided the opening of the funnel from which the mercury falls is sufficiently fine to give a thin enough stream. Moreover, this opening frequently becomes choked and the funnel must be removed and cleaned. Various automatic arrangements have been used to return the mercury to the funnel,[‡] but they are more or less complicated and have no advantage in point of time. The electric purification is also time-consuming, requires efficient stirring and is not particularly effective.

For the removal of silver and gold, of course, distillation is necessary. This is usually done under reduced pressure in one of the various forms of still which have been proposed. Hulett and Minchin[§]

---

[*] *Z. analyt. Chem.*, *2*, 241 (1863).

[†] Wolff and Waters, *Bull. Bureau of Standards*, *3*, 623; *4*, 1 (1907)

[‡] *E. g.*, Desha, *Am. Chem. J.*, *41*, 152 (1909).

[§] *Phys. Rev.*, *21*, 388 (1905). The oxidation of the zinc or cadmium by the air is an important factor in the purification.

have shown that unless the mercury is distilled in a current of air and with no bumping, the more volatile metals such as zinc or cadmium are not removed. The distillation apparatus they describe yields mercury of a high degree of purity, as shown by their careful testing.

This is a modification of the L. Meyer method, which is very rapid and, as recent measurements show, very efficient, a degree of purity being obtained comparable with that of distilled mercury. Instead of the mercury being delivered from a funnel drawn out to a fine point, a separatory funnel is used, the delivery tube of which is slightly narrowed 0·5 cm. from the lower end. Over this end is bound with twine a piece of rather closely woven muslin. The mercurous nitrate solution is contained in a piece of glass tubing 1·5 m. long and 2 cm. in diameter. The narrow delivery tube at the bottom is bent up 25 cm. This is more than usual, as it is necessary to have the height of the mercury in the wide tube about 5 cm. so that the tiny globules which fall will have time to coalesce before they reach the small tube, otherwise the mercury will not be delivered dry, or it may even be swept out of the tube followed by the entire solution of mercurous nitrate.

The end of the tube of the separatory funnel over which the cloth is bound should dip below the surface of the mercurous nitrate solution. This diminishes the surface tension of the mercury so that the fine streams running through the cloth breaks up into very minute globules. Unless the surface of the mercury is clean the funnel cock should be closed just before all the mercury has entered the stem of the funnel, as the dirt thus carried into the stem would clog the cloth.

By this arrangement, instead of one stream of mercury, as in the ordinary form of the apparatus, we have several hundred streams, finer than is practicable with the usual drawn-out funnel, and a number of them may be choked by particles of solid amalgam or dirt, before the flow of mercury is sensibly diminished. The rate of flow need only be limited by the rapidity with which the globules coalesce on reaching the bottom. The rapidity of the washing and the large surface of mercury exposed are obvious.—J. H. Hildebrand.—*J. Amer. Chemical Society*, Aug., 1909, p. 934. (A. McA. J.)

Gold Mining in Korea.—"Korea is a highly mineralised country, possessing an excellent climate and an abundant supply of good and cheap labour. Its chief gold quartz mines are included in the American concession at Unsan, which was granted originally in 1895 for 25 years, and, as modified and extended, may be operated until 1954. At the Unsan mines a series of large quartz lodes, occurring in granite, are being mined successfully and profitably under American management.

The Unsan Gold Mines (Oriental Con. Mining Company) constitute the most important mining property in this part of Asia. While other Korean mining properties are now recognised as possessing dividend-paying resources, the Unsan mines have heretofore been the only properties operated continuously on any extensive scale by modern methods in the Empire. This company is situated in north-western Korea, about 60 miles south of the Yalu River, and at the present time has a total of 230 stamps in five different plants, crushing some 30,000 tons per month of low-grade gold ores. The character and value of the ore varies considerably at the several mines. It averages about 5 dwt. gold per ton, with very little silver. This low-grade ore is quite base, containing from 5% to 6% of gold-bearing sulphides.

The Kuk San Dong plant (40 stamps) treats an ore averaging 12s. in value per ton, being quite base and containing 8% sulphides of arsenic. Maibong plant (20 stamps) treats £2 ore, containing 3 or 4% sulphides of arsenic, iron, lead and zinc. Candlestick plant (10 stamps) treats £3 to £4 ore, containing as high as 15% sulphides. Taracol plant (80 stamps) mills 16s. ore, with 5% sulphides of iron, lead and zinc. Tabowie plant (80 stamps) treats 20s. ore, with 5% sulphides.

All the plants wet crush, amalgamate and concentrate about 1,000 tons per day of £1 ore. The total extraction on all values equals about 81% at the present time. About 2,500 tons of concentrates per month are cyanided at the several plants by the old 18-day leaching method, giving a total extraction of 83%.

Cost of Mining and Milling for 1908.—30,000 tons per month (20s. 10d. ore).

	s.	d.
Mining and development ...	5	0
Milling and concentrating ...	2	3
Cyaniding concentrates ...	0	6
Ore transportation	0	1
General expenses	2	0
Total expense	9	10

All of 4s. out of a total expense of 9s. 10d. is for cordwood and mining timber alone. Although we have cheap labour our wood and timber problem is serious. The company is now making arrangements to operate the two 80-stamp mills and plants by hydro-electric power, and through these changes reduce the cost of milling considerably. A railroad is also being constructed into the Yalu River timber country, and should, when completed, reduce the costs for wood and timber. During the coming year the costs, with electric power and cheaper timber, will, no doubt, reach a total of 8s. We will be able to handle any ore at a profit that will assay more than 2 dwt. per ton. It is very doubtful if the Oriental Con. Mining Company could carry on operations in America with only 230 stamps, where the cost of labour is high. The Alaska-Treadwell and Homestake Companies, U.S.A., with their 8s. to 16s. ore, are only able to operate through handling immense quantities of ore (3,000 to 4,000 tons per day) in their large mills.

Labour.—Our native labour consists of Koreans, Chinese and Japanese, about 8,000 on the concession being dependent upon the company for a living. Japanese are employed as carpenters, foundrymen and machinists, at 3s. to 5s. per day, and at this work are far better than the other Orientals. The Koreans are the most efficient miners and machine men (two Koreans being equivalent to one average European or American miner). As miners they receive 1s. and as machine men 2s. per day of 10 hours. Ordinary Korean coolies receive from 10d. to 11d. per day, and out of this they are expected to live. Their food (rice and fish) costs from 5d. to 8d. per man per day. Chinese are employed on the rock-crushing, tramways, and stone masonry work, and earn from 12d. to 15d. per day. By far the majority of Orientals employed are Koreans. The Korean miner possesses all the good qualities of a miner, with very few of his defects, being physically strong and very easy to handle. Generally speaking, a Korean miner's father and grandfather were miners before him, and his son will, as a rule, be a miner. They withstand hardship, such as water and bad ventilation, much better than the average white miner. When practicable, it is customary to employ Korean miners on contract work, and when this is done the results obtained per day by them are equal to the average American or Cornish miner.

Gold Extraction.—The ores treated contain only a fraction of an ounce of silver per ton, and therefore can be classed as low-grade base gold ores. With a 16s. ore, and with our present plants and treatment, the following results are obtained:—Milling 30,000 tons per month on 16s. ore—Values won by plate amalgamation, 46½%; values won by concentration, 41%. Cyaniding 2,500 tons per month on £4 concentrates—Values won by cyaniding (18-day leaching), 83%. Milling and cyaniding—Values lost in milling, 12·5%; values lost in cyaniding, 6·1%; total loss, 18·6%. The present method of treatment gives an actual extraction of 81·4 per cent. of the values. I have recommended improvements in concentration and cyaniding concentrates, resulting from my past four years experimental work in Korea. At the end of my tour to the various mining districts I shall remodel the old plants and bring them up to date. I have through all-sliming of our concentrates (no roasting), air agitation, vacuum filtering, obtained a total extraction of 93% with a 48-hour treatment. Average cost by this method per ton of concentrates is 13s. (per ton ore crushed 7d.) The average cost by the 18-day leaching method equal about 11s. (per ton ore crushed 6d.) Improvements in our concentration methods will lower our mill tails to about 18d. per ton, when milling 16s. ore. The improvements will show an increase on the present total extraction (81·4%) of from 6 to 7 per cent. Presently the company shall start retreating its old cyanide concentrate tailings dumps which have accumulated during the past six or eight years. They will average from 4 dwt. to 6 dwt. per ton, having been previously cyanided by the 18-day leaching method for an extraction of 80 to 83 per cent. These dumps contain 40 to 60% quartz sands, the remaining product being sulphides. The new plant for these dumps will reconcentrate into sands assaying 1 dwt., and sulphides valued at 12 to 15 dwt. per ton. The sulphides to be all-slimed in tube-mills in cyanide solution, air agitated, and vacuum filtered within my special combined agitator and vacuum-filter. This method will show a net profit of about 4s. to 5s. on each ton of dump as it stands. The extraction obtained on the sulphide equals 70%. The company at present is producing some £25,000 per month in bullion, and to date has produced about £2,400,000 from these low grade ores. I will say that this gold mining company is the only one operating in Korea at a profit at the present time. Its success has been largely due to the good management of Mr. H. F. Meserve, who has been its leader from the beginning. —By A. E. DRUCKER, Metallurgical Engineer to the Oriental Con. Mining Co., Korea. (A. R.)

TUBE MILL PRACTICE IN MEXICO.—"A tube mill was early recognized in Mexico for fine grinding. Krupp mills of standard types were largely used at first, but subsequently various American makes came into favour. There has been for some time considerable discussion as to the relative merits and efficiency of long tube mills and short tube mills, but the subject has not been demonstrated satisfactorily to any extent except to the partisans of the respective types. A few years ago at the El Oro mill the practice of introducing a ribbed lining, into which the pebbles became fixed and formed automatically a flint lining for the mill, was introduced and met with immediate success. This lining is quite generally used. Following this improvement the Ennis tube mill liner was invented and introduced at Guanajuato. The idea consists of removable bars properly secured by bolts which pass through the fixed lining in the shell. These bolts are countersunk so as to permit the wearing of the liner to the level of the fixed lining, which only becomes slightly worn between the ribs. These fixed plates have an indefinite life, estimated at at least five years. The obvious saving comes in the reduction of the weight of metal necessary for the restoring of the lining. In a tube mill 16 ft. by 4 ft., the old lining weighed 8 tons. The new lining will have the same weight for the first set, including plates and bars, but subsequently the bars will weigh only 3½ tons. The bars will last about 8 months, and about the same length of time as a set of the old time lining. Four of the tube mills at the Bustos mill, Guanajuato, are equipped with this device, and it is expected that others will be likewise equipped.

The use of quartz ore in place of flint pebbles has been introduced at El Oro and has been found practicable. It has certain disadvantages and limits, reducing the output and causing excessive wear on the silex lining. The best practice has been obtained by introducing a small amount of pebbles with the quartz pieces in the tube mills of the El Oro type, the pebbles by reason of their shape wedging between the ribs and forming the usual lining. The subject is being carefully worked out by H. B. Smith, the manager of the Guanajuato Reduction and Mines Co., and it is expected that some definite data on this important subject will be available soon.

The question of the best speed and proportion of pebbles to ore is being carefully studied in the different camps, and when this data becomes available it will furnish a definite standard in tube mill practice, which will be of interest everywhere."—*The Mining World*, July 3, 1909, p. 35. (C. B. S.)

MINING.

CAPACITY OF SINKING PUMPS.—"The capacity of a sinking pump is considerably increased by placing it very near the sump, so that it can readily maintain its supply of water, without drawing air."—*Queensland Govt. Mining Journal.* (C. B. S.)

RELATION BETWEEN ROCK TEMPERATURE AND AIR TEMPERATURE IN MINES.—"On the basis of an extensive series of calculations, E. Schmid (*Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, July 17) arrives at the conclusion that in very deep mines it will be impracticable to reduce the temperature of the ventilating current sufficiently by merely increasing its velocity, and that the only practicable method will be to cool the air by artificial means at the working face, as was done in driving the Simplon tunnel. — *London Mining Journal.* August 28, 1909. p. 276. (A. R.)

ACCIDENTS DUE TO ELECTRICITY IN BRITISH COAL MINES.—"The reports of H. M. Inspectors of Mines on coal mining show that in the various coal mining districts there is a disposition on the part of many owners and managers to extend the use of electricity as a motive power, especially in the Southern district, where it is used for pumping, haulage and the working of about a dozen coal-cutters. With regard to the latter, however, the Inspector advises erecting air compressors underground as far in-bye in the intake airway as it may be safe to work electrically and supply compressed air to the coal-cutters at the face. In the East Scottish district, where there were 126 deaths and 524 non-fatal accidents, two fatal accidents only were due to electricity. In the West Scotland district, of 85 deaths and 374 non-fatal accidents only one was supposed to be due to electricity. In the Newcastle district, where 42 electrically driven coal-cutting machines are used, there were no electrical accidents. In the Durham district 104 deaths and 570 injured were recorded, of which there was only one non-fatal accident due to electrical causes. In the Yorkshire and Lincolnshire district, with 111 electrical coal-cutters there was no accident attributable to electricity. In the Liverpool and North Wales district there were 155 deaths and 300 injured, one workman being killed by an electric shock of 500 volts. In the Midland district 91 electrical coal-cutters were at work with two fatal electrical accidents. In the Stafford district 111 men were killed and 423 injured, and only one non-fatal accident due to electricity occurred. In the Cardiff district 134 deaths and 423 injured were recorded, of which one non-fatal accident was attributed to electrical causes. In the Swansea district one fatal accident occurred resulting in the deaths of three workmen, due to electrical detonation of gelignite on refilling a partially blasted hole. In the Southern district, of 93 fatal and 277 non-fatal accidents one accident with one death occurred from the use of electricity. These data prove that the use of electricity underground is safe so long as proper precautions are taken, as many of the accidents which happen can be ascribed to ignorance or carelessness on the part of the workmen."—*Electrical Engineer*, Aug. 13, 1909, p. 182. (J. A. W.)

SINKING A SHAFT BY THE CEMENT PROCESS.—"Dobbelstein describes the sinking of a shaft through water-bearing strata, by the cement process, at the Liévin Colliery. On reaching the wet ground a concave steel bottom was built in the shaft, with covered orifices for eight boreholes round the periphery. Five of these holes were driven down to the solid sandstone and filled with cement milk, about 122 tons of cement being used. After seventy-five days the false bottom was removed and sinking recommenced; but, owing to the presence of uncemented fissures in the rock, and the fact that the cement had not set hard, the influx of water soon reached 3,300 gallons per hour, so that, by the time another 60 ft. of tubbing had been put in, the false bottom was replaced and supplementary boreholes were driven and cemented, 50 tons of cement being used. On continuing the sinking, it was found that the cement milk had penetrated the rock to a depth of 25 ft. below the bottom of the boreholes, and completely shut off the water from outside the ring of holes. The actual cost of the cementing process was about £1,000, or only one-eighth the expense of the freezing process" — *Mining Journal* (London). (C. B. S.)

ROPE CAPPINGS AND ATTACHMENT TO DRUM.—Ample strength in the rope itself is, of course, of no avail if its attachment to the drum on the one hand, or to the cage on the other, is not proportionately strong.

The former attachment to the drum is comparatively easily provided for by taking advantage of what is known as coil friction. It is usual to purchase a rope of considerably greater length than is represented by the distance from the drum to the bottom of the shaft, so that when the cage is at the bottom there are still three or four complete coils left on the drum. This extra length not only provides spare rope to make up for the pieces which should be cut off at each recapping, but they also serve to secure the rope to the drum.

The coefficient of friction between a greased wire rope and a wood-lagged drum will probably be about 0·35, which means that one coil on the drum would enable a pull of 1 lb. to resist a strain of 9 lb., two coils 9 × 9, three coils 9 × 9 × 9, and four coils 9 × 9 × 9 × 9, or 6,561 lb. With four coils on the drum, therefore, and the end passing through the lagging secured with a force of, say, 50 lb., the rope is securely held, even if the load amount to as much as 140 tons.

One frequently finds that the rope end which is passed through the drum lagging is given two or three turns around the drum shaft, in which case the security is increased to an enormous extent, and there is not the slightest necessity for the two or three strong clamps which generally complete the arrangement. Yet at the other end of the rope we have been satisfied with an attachment only giving half the strength of the rope.

Too much care cannot be exercised in the selection, design and application of rope cappings. The forged steel conical socket, in which the prepared end of the rope is secured by means of white metal, when properly and carefully made, possesses a greater strength than the rope to which it is fitted.—G. H. WINSTANLEY. *Queensland Government Mining Journal.* September 15, 1908. p. 465. (C. B. S.)

RECORD DRIVING.—"The Modderfontein B Gold Mines has achieved a notable record for the Rand in driving during the month of September. In a drive from the West Central shaft, a distance of no less than 334 ft. was accomplished in sixty-two shifts, which leaves far behind the previous best, which was achieved by the Van Dyk in sixty-two shifts just a year ago. Before the Van Dyk record was made, the Simmer Deep held the record, having with three machines driven 294 ft. in sixty-one shifts in April, 1908. In November of the same year 272 ft. were driven in the Vogel Deep with two machines. The Van Dyk record was won by one white man and five Kafirs working each shift, using two 3¼ in. machines. In the Modder B, two machines were operated by two machinemen; the average holes per round was 12·5; the average footage per day was 11·13 ft., and the size of the drive 6 ft. × 7 ft. In the Van Dyk, the average number of holes per round was twelve, and the average number of feet per round was 5·209."—*South African Mining Journal*, Oct. 2, 1909, p. 58. (A. R.)

MISCELLANEOUS.

POINTS ON ADJUSTING INSTRUMENTS. — "The following points on adjusting instruments are doubtless familiar to many field-men, but as they do not appear in any of the text or field-books that have come under the writer's observation, they may not be out of place. They possess the advantage that many of them can be made without the help of an assistant.

The Transit.—To Adjust the Standards.—Pick out some line, as the edge of a chimney, or the corner of a building, that should be plumb. Sight to a high point on it and depress the telescope, noting to which side of the line sighted, the line of collimation falls. Reverse the telescope, turn the slate around, and repeat. If the line of collimation falls the same distance to the other side the line sighted on is plumb, and the standard can be adjusted so the line of collimation follows it all the way down.

To Adjust the Line of Collimation.—Having adjusted the standards, set up over a point and set two others by foresight 200 and 400 ft. distant, thus giving three points in a straight line. Set up over the middle point sight to one of the others, reverse, and note the distance the line of collimation falls from the other. The vertical cross-hair is then moved over one-half the error, instead of one-fourth, as is customary, when the two outer points are set by a foresight and backsight.

Second Method.—Pick out some distant object for a foresight, reverse and find some near object on line. Turn the plate around and sight to the near object again. Plunge the telescope and note where the line of sight appears with reference to the distant object. This method of long and short sight makes the error more apparent, which should be taken into consideration in making the adjustment, and care used that the cross-hairs are not moved too much.

To Adjust the Telescope Level and Vertical Circle. Set the vertical circle so that its zero coincides with the zero of vernier. Pick out some object 300 ft. to 500 ft, distant cut by the horizontal cross-hair, reverse the telescope, turn the plate around, and again bring the cross-hair on the object sighted. If the zeros do not coincide, read the error, and move the vernier over one-half of it, repeating the operation until the zeros coincide in both positions of the telescope. When this point is reached the telescope is level, and the bubble should be adjusted to it.

The Level.—To Adjust the Horizontal Cross-Hair. —The bubble being adjusted so that it will reverse perfectly, drive two pegs in the ground 300 to 500 ft. apart, and set up as near one of them as possible to get a clear sight. Take a reading with the target and assuming and elevation for the peg of 100 figure and H.I. Move the rod to the other peg, take a reading and calculate its elevation. Move the instrument to the second peg, take a reading, and from it and the elevation found, calculate a second H.I. Move the rod to the first peg, and if the instument is in adjustment, the reading will give it an elevation of 100. If this reading is not obtained, set the target half way between the reading which is obtained, and the one which should have been, and adjust the horizontal cross-hair to it. This adjustment can also be used for the telescope level of the transit."—L. A. PALMER.—*The Mining World*, Aug. 21, 1909. (C. B. S.)

CORROSION OF REINFORCED STEEL.—"It has been said that reinforced concrete should be avoided as it is a treacherous material to use owing to the fact that the metal corrodes, and being covered by concrete the extent of the corrosion can never be ascertained, and therefore many well-known engineers up to the present have avoided the use of the material owing to this impression. If they were right in their assumption, and the steel did corrode, and there was no remedy for it, then reinforced concrete would soon have had its day, for its weakness in this respect would become generally known, and it would naturally be avoided ; but from a series of experiments which the author has recently made, he is in a position to state definitely that no such fears need be entertained.

The results of these experiments—21 in number—have led to the following conclusions ;

(1) That rusty steel imbedded in concrete will in a very short time become bright, regardless of whether the concrete is in water or air. This point has, in the author's opinion, been conclusively proved by his experiments.

(2) That the application of cement grout to steel is an effectual safeguard against corrosion, but that the greatest care should be taken in the grouting process to see that every portion of the steel is well coated, and that before the steel is embedded in the concrete the cement grout is allowed to become quite dry upon the steel.

(3) That if the aggregate used for the concrete is not porous and the concrete is well mixed, the reinforcement being well embedded, no cement coating is needed. (Seeing that the application of a coat of cement grout is such an inexpensive procedure the author makes it a rule in carrying out work of this kind to have all reinforcement coated in this manner.)

(4) That no porous materials, such as coke-breeze or slag, should be used in connection with reinforced concrete work, if such concrete is intended to be under water or exposed to the air.

(5) That linseed oil or turpentine or probably any other coating except cement or lime applied to steel before its insertion in concrete facilitates rather than prevents the rusting of the metal.

(6) That it is of great importance to see that the reinforcing steel is well embedded in the concrete, so that every portion of it is covered with cement.

(7) That the best results were obtained when the aggregate consisted chiefly of broken stone or brickbats. Gravel would no doubt answer as well.

The author was surprised that such a good result was obtained with an aggregate composed of brickbats.

(8) That river sand, generally speaking, is not satisfactory for reinforced concrete work, where such work is required to be water-tight." — E. R. MATTHEWS, Society of Engineers.—*The Mining World*, August 24, 1909, p. 223. (C. B. S.)

SOLUTION OF GOLD IN CYANIDE. — Gold dissolves in cyanide solution in direct proportion to the area of metallic surface presented. The area of a sphere equals the square of the diameter multiplied by 3·1416 ($A=\pi d^2$) and the volume equals one-sixth of the product of the cube of the diameter and 3·1416 $\left(A=\frac{\pi d^3}{6}\right)$. Therefore, to take the simplest conditions, if a gold particle has a diameter of 1 mm., its volume will be 0·5236 cubic millimetres, and its area 3·1416 square millimetres. For a sphere of 2 mm. diam., the corresponding figures are 4·1888 and 12·5664. The ratio of volume to surface in the larger particle is as 1 to 3, while in the smaller it is as 1 to 6. At ½ mm. diameter the ratio becomes 1 to 12, and so on. The economical limit of size is one to be determined in the light of results.—*Mining and Scientific Press*, March 20, 1909. (K. L. G.)

NON-CRACKING GLASS.—"The Baccarat Works (France) has manufactured a new kind of glass which does not crack at a temperature of 100 degs. C., when sprinkled with water at 15 degs. C. This glass has been accepted for safety lamps as superior to Jena glass. The composition is as follows : 75 per cent. sand, 13 per cent. bicarbonate of soda, 9 per cent. manganese carbonate, 6 per cent. zinc oxide, 5 per cent. lead oxide." — *Russegna Mineraria*, August 21.—*London Mining Journal*, Oct. 2. p. 13, 1909. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

RAPID METHODS FOR THE CHEMICAL ANALYSIS OF SPECIAL STEELS, STEEL-MAKING ALLOYS, AND GRAPHITE. By CHARLES MORRIS JOHNSON. First Edition. 12s. 6d. (New York, N.Y., U.S.A.: John Wiley & Sons. London: Chapman & Hall, Ltd.)

"Nothing but praise can be accorded to this thoroughly practical work, whose 220 pages include innumerable useful tips on special points in connection with the analysis of 'special steels,' and of special metals and alloys used in their manufacture.

The advance during the last few years in the use of such metals as chromium, nickel, manganese, tungsten, cobalt, molybdenum, vanadium, tantalum, niobium, etc., and in the production of such metals and their ferro-alloys, for producing special properties in steel, has revolutionised the special steel industry and, incidentally, the methods employed in their analysis. This work, whose author is the chief chemist to the Park Steel Works of the Crucible Steel Company of America, gives details, and especially notes on what to expect and what to avoid, which cannot be obtained elsewhere.

An important feature of the book is the fact that the author appears to have not merely selected methods of recognised value, but has tested most of them himself, and has only recommended those which he himself knows to be reliable.

Although dealing mainly with special steels, there is much of value to the ordinary iron and steel or general analyst. Methods are given, for instance, for testing tungsten, molybdenum, and other powders: special tips are given for carbon, sulphur, phosphorus, and silicon determination; and a few notes useful to those engaged in the assay of ores are added.

Apart from its interest to the analyst, a perusal of the work shows how largely the iron and steel industry is now connected with 'non-ferrous' metallurgy, on account of the increasing use of other metals in steel manufacture. Each manufacturer has his own fancies as to metals to be added, and as to their proportions, and, although one may doubt whether so great a variety in composition is necessary or advisable, the difficulties of the analyst are growing daily, and can only be met by the publication of such works as that under review, written by men of experience and kept well up to date." — *London Mining Journal*, Sept. 25, 1909, p. 429. (A. R.)

INDEX OF MINING ENGINEERING LITERATURE. By Dr. W. R. CRANE, Dean of the School of Mines and Metallurgy, Pennsylvania State School, State College, Pa. Price $4. 812 pages. (New York City: Wiley & Sons.)

"This volume includes a complete and carefully compiled index of some eighteen engineering publications, including the mining journals, and the transactions of societies. In many cases the references are complete for all volumes from the first date of publication. The book represents a vast amount of research work and will be of great value to the mining profession. The reviewer has had the privilege of using Dr. Crane's card catalogue before publication, and he therefore has no hesitancy in recommending this publication most highly. It is to be hoped that annual appendices will keep the index up to date."—*Mines and Minerals*, Sept., 1909. (A. R.)

Bailey, C. H. First Stage Inorganic Chemistry (Theoretical). Edited by William Briggs. 4th Edition. Cr. 8vo., pp. 257. *W. B. Clive.* 2s.

Ball, Sir Robert S. The Earth's Beginning. New Edition. 8vo., pp. 400. *Cassell.* 7s. 6d.

Bausor, H. W. First Stage Inorganic Chemistry (Practical). Revised Edition. (Organised Science Series.) Roy. 16mo., limp. *Clive.* 1s.

Belden, A. W., Delamater, G. R., and Grovees, J. W. Washing and Coaking Tests of Coal at the Fuel Testing Plants, Denver, Colo., July 1, 1907, to June 30, 1908. (Bulletin U.S. Geological Survey.) 2 Plates and Engravings. 8vo., sd. *Wesley.* 1s. 6d.

British Standard Specification and Sections of Steel Fish Plates for British Standard Bull Head Railway Rails, etc. Folio, sd. *C. Lockwood.* Net 10s. 6d.

Burrows, J. S., and Randall, D. T. Results of Purchasing Coal under Government Specifications. With a Paper on burning the Small Sizes Anthracite for Heat and Power Purposes. (U.S. Geological Survey Bulletin.) 8vo., sd., pp. 44. *Wesley.* Net 1s. 6d.

Clowes, Frank, and Coleman, J. B. Quantitative Chemical Analysis. 8th Edition. 8vo., pp. 590. *Churchill.* Net 10s. 6d.

Kelsey, W. R. Continuous Current Dynamos and Motors and their Control. 2nd Edition. Cr. 8vo., pp. 354. *Tech. Pub. Co* Net 7s. 6d.

Macfarlane, Walter. Laboratory Notes on Iron and Steel Analysis. Cr. 8vo., pp. 478. *Longmans.* Net 7s. 6d.

Marks, Lionel S., and Davis, Harvey N. Tables and Diagrams of the Thermal Properties of Saturated and Superheated Steam. 8vo. *Longmans.* 7s. 6d.

Maxim, James L. The Apprentice's Course of Experimental Physics and Mechanics. For Preliminary Technical Students and Secondary School Pupils. 12mo., pp. 126. *Longmans.* 1s. 6d.

Standard Specification for Ammeters and Voltmeters. Folio, sd. *C. Lockwood.* Net 2s. 6d.

Standard Specifications for Keys and Keyways. Folio, sd. *C. Lockwood.* Net 2s. 6d.

Standard Specification for Wrought Iron of Smithing. Quality for Ship Building. Grade D. Folio, sd. *C. Lockwood.* Net 2s. 6d.

Workshop Receipts for Manufactures and Scientific Amateurs. Vol. 2. New and thoroughly Revised Edition. Cr. 8vo., pp. 550. *Spon.* Net 3s.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 484/09. Henry James Shedlock Heather. An improved method of automatically isolating damaged electrical cables. 27.10.09.

(P.) 485/09. Richard Wild. Vacuum brake for mine cages or skips. 28.10.09.

(P.) 486/09. Charles Robert Bishop A new and improved lubricant mixture. 28.10.09.

(P.) 488/09. Richard Colson. Improvements in apparatus or means for use in surveying mine stopes and the like. 28.10.09.

(P.) 489/09. William Berthelson. An improved chuck for fastening drills in rock drilling machines and the like. 29.10.09.

(P.) 490/09. Fritz Wagner. Improvements in means for obviating the vibrations of main pressure actuated valves. 29.10.09.

(C.) 493/09 William Rees Bevan (1), Daniel Thomas Morley (2). Improvements in nut locks. 29.10.09.

(P.) 497/09. John Whitford (1), James Amos Mills (2). Improvements in means or devices for loading skips or other vehicles in mine shafts and the like. 1.11.09.

(P.) 499/09. Robert Drew. Improved holding jack for idle battery stamps. 1.11.09.

(P.) 502/09. Henry James Shedlock Heather. An improved method of automatically isolating damaged electrical lines or sections of lines. 2.11.09.

(C.) 503/09. John Scarisbrick Walker (1), Thomas Ashcroft Walker (2), Edwin Robert Walker (3). Improvements in or relating to valves for blowing engine compressors and the like. 5.11.09.

(P.) 504/09. Kenneth Oldham Powrie. Surveying instrument. 5.11.09.

(C.) 505/09. Charles Allen. Slime separator and classifier. 6.11.09.

(P.) 506/09. Walter Williams Wright. A gradual safety brake for use on mine cages, lifts and the like. 8.11.09.

(P.) 507/09. James Ernest Horsfield. Improvements in nozzles and the like. 8.11.09.

(P.) 509/09. Francis Joseph Watts (1), Fraser Frederick Alexander (2). Improvements in electrically actuated apparatus for cleaning and scouring amalgamating plates or surfaces. 10.11.09.

(C.) 510/09. Charles Desmarais (1), Michael John O'Brien (2). Core extractor for mining drills and the like. 10.11.09.

(P.) 511/09. Sidney Sherrard Osborn. Improvements in tube mill linings. 10.11.09.

(P.) 512/09. William Cullen (1), Eric Hartwig Weiskopf (2), British South African Explosives Co., Ltd. (3). Improvements in nitro-glycerine explosives. 10.11.09.

(C.) 513/09. Claude Albermarle Bettington. Improvements in and relating to apparatus for pulverising coal and other materials. 12.11.09.

(C.) 514/09. Harry August Stockman. Improvements in means or apparatus for disintegrating ores, minerals and other materials, and recovering the metal or other contents thereof. 12.11.09.

(C.) 515/09. Howard Robard Hughes. Drills. 12.11.09.

(C.) 516/09. Donald Barnes Morison. Improvements relating to cooling towers. 12.11.09.

(C.) 518/09. Frank Humphris. Improvements in and relating to toothed driving gear. 12.11.09.

(C.) 519/09. Bernhard Baron. Improved means for charging smoking pipes. 12.11.09.

(P.) 520/09. Thomas Henry Bradbury. Improvements in electric resistance furnaces. 12.11.09

(P.) 521/09. David Robert Colman. Improvements in means or apparatus for depositing mine residues or other material upon mine dumps or other depositing sites. 13.11.09.

(P.) 522/09. John Whitford (1), Joseph Amos Mills (2). Improvements in means for changing vehicles traversing mine shafts. 13.11.09.

(P.) 523/09. George Taylor Philip (1), Arthur Chichele Plowden (2). Improvements in or relating to couplings for trucks and other vehicles. 15.11.09.

(C.) 524/09. Frederick Godfried Carl Rincker (1), Louis Wolter (2). Improved method of and means for obtaining gas from tar oil and the like in gas producer plant. 16.11.09.

(P.) 525/09. Alfred George Newkey Burden. Improvements in guides for the stems of the stamps of stamp mills. 17.11.09.

(P.) 526/09. Edward Henry Croghan. Extraction of litharge from used cupels. 19.11.09.

(P.) 527/09. Sidney Rickman Adams. Improvements in means for regulating or shutting off the underflow from de-watering and classifying cones and the like. 19.11.09.

(P.) 528/09. Joseph Henry Dobson. Electric heat accumulator. 19.11.09.

(C.) 529/09. Allan Macpherson. Apparatus for generating and storing petrol and like gas. 19.11.09.

(C.) 530/09. Arthur Owen Jones. Improvements in or relating to coke ovens. 19.11.09.

(C.) 531/09. Arthur Owen Jones. Improvements in or relating to coke ovens. 19.11.09.

(C.) 532/09. Arthur Owen Jones. Improvements in or relating to coke ovens. 19.11.09.

(C.) 533/09. Arthur Owen Jones. Improvements in and means and appliances for discharging and conveying away the coke product from vertical coke ovens or gas retorts with base discharge. 19.11.09.

(P.) 534/09. Edmund Arthur Hamblin. An improvement in or relating to the method or manner of uncoupling railway engines, carriages, trucks or vehicles of any description. 20.11.09.

(P.) 535/09. Frederick Walter Cindel. Improvements relating to stamps. 20.11.09.

(P.) 536/09. George Watt. Tool for shaping rock drills. 22.11.09.

(P.) 538/09. John Wilkinson Kirkland. Improvements in electrical hoisting apparatus. 23.11.09.

(C.) 539/09. John Turns. An apparatus for the storage and compression of air gases and liquids under pressure. 23.11.09.

(P.) 541/09. Victor Frolich. Improvements in means for operating gravity stamps. 24.11.09.

(P.) 542/09. Hugh McMenamin. Improvements in means for feeding ore to the stamps of stamp mills. 24.11.09.

(P.) 543/09. Douglas William Stacey. An improved filter for cyanide solutions. 24.11.09.

(P.) 546/09. Charles Harrison Dixon (1), William Beaver (2). Mechanical finger for receiving and holding gravity stamps when idle. 25.11.09.

(C.) 547/09. Oscar Clarence Beach. Improvements in grinding mills. 26.11.09.

(C.) 548/09. Robert Lemmon (1), Robert Patrick Holmes (2). Automatic Filter press. 26.11.09.

(C.) 549/09. Claude Albermarle Bettington. Improvements relating to steam generators. 26.11.09.

(C.) 550/09. Paul Brenner (1), Otto Hoffmann (2). Improvements in a self acting valve for pumps and similar devices. 26.11.09.

(P.) 551/09. Kenneth Bingham Quinan. Improved means for facilitating the complete detonation of gelatinous nitro-glycerine explosives. 29.11.09.

(C.) 552/09. Edward James Mahood. Improvements in apparatus for raising and lowering the propeller in that class of agitating apparatus in which a propeller is used as the agitating medium. 30.11.09.

(C.) 553/09. Alexander Albert Holle. Improvements in steam and other fluid turbines. 30.11.09.

(P.) 554/09. Frederick Hudson Shepherd Shepherd (1), Justinian Myer Cohen (2). Improvements in apparatus for drilling or boring holes by hand or other power for mining and the like operations. 1.12.09.

(P.) 555/09. Gustavus Adolphus Sheeley. Improvements in linings for tube and similar mills. 1.12.09.

(P.) 556/09. James Wilson McNicol. Improvements in jockey or rope grips for mechanical haulage. 3.12.09.

(P.) 557/09 Alfred Sidney Colam. Improvements in side or end discharge trucks. 3.12.09.

(C.) 559/09. Loftus Gray. Explosive compound and process for making same. 3.12.09.

(P.) 562/09. Cecil Francis Webbe (1), Peter David Voigt Alexander (2). Improvements in jockeys or rope grips for mechanical haulage. 4.12.09.

(P.) 563/09. Hugh Ross (1), George Richard Mayor (2). Miss-fire proof paste. 6.12 09.

(P.) 564/09. Andrew Swan. An improved apparatus for the separation of liquids from finely divided solids. Particularly applicable to the treatment of gold slimes, ore slimes and the like. 6.12.09.

(P.) 566/09. Robert Allen. Improvements in apparatus for filtering or clarifying liquids. 8.12.09.

(P.) 567/09. Hans Charles Behr. Improvements relating to clutches suitable for winding engine drums. 8.12.09.

(C.) 569/09. Hugo Philip Crush. An improved method of and apparatus for filtering liquids. 10.12.09.

(C.) 570/09. Alfred Moffitt Sedgley. Improvements in centrifugal pump, filters and strainers. 10.12.09.

(C.) 571/09. Hugo Ruth. Improvements relating to reciprocating feed troughs. 10 12.09.

(C.) 572/09. Lewis Condict Bayles. Improvements in direct acting engines. 10.12 09.

(C.) 573/09. William Edward Bandfield. Improvements in the attachment of telegraph and similar line wires to insulators. 10.12 09.

(C.) 574/09. John Collins Clancy. Treatment of precious metalliferous ores. 10.12.09.

(C.) 575/09. Thomas Julian Barre. An improvement in valves. 10.12.09.

(P.) 576/09. David Gilmour. Improvements in fixing cam pulleys to cam shafts. 13.12.09.

(C.) 577/09. Henry Livingstone Sulman (1), Henry Howard Greenway (2), Arthur Howard Higging (3). Improvements in or relating to the concentration of ores. 15.12.09.

(P.) 578/09. Arthur George Taylor. Improved apparatus for interlacing wires. 17.12.09.

(P) 579/09. Hugh McMenamin. Tube feeder. 17.12 09.

(C.) 580/09. Reginold Vanderzee Farnham. Improvements in gas producer plant. 17.12.09.

(P.) 581/09. William Charles Stephens. Improvements in valve mechanism for rock drills and other reciprocating engines. 17.12.09.

(P.) 583/09. Alexander John Arbuckle. Improvements in apparatus for classifying crushed ore products. 18.12.09.

(P.) 584/09. John Sacks. Improvements in the process for production and concentration of acetic acid. 18.12.09.

(P.) 585/09 Henry Wilfred Frewin. An improved amalgam trap. 20.12.09.

(C.) 586/09. Frank Ondra (1), Alfred Adair (2). Improvements relating to the treatment of zinc solutions, for the recovery of metals or metal compounds therefrom. 20.12.09.

(P.) 587/09. Jean Harte. Improvements in detonators. 21.12.09.

(C.) 591/09. Henry Malcolm Caldwell (1), Thomas Smith (2). Improvements in or relating to means for governing and/or regulating pressure of gas or supply of gas; either with or without an indicator. 24.12.09.

Abstracts of Patent Applications.

(C.) 484/09. Henry James Shedlock Heather. An improved method of automatically isolating damaged electrical cables. 27.10 09.

The application relates to a device for automatically isolating faults on three-phase cables. The applicant proposes to use the Mertz-Price system of balanced current transformers for operating the trip coils, but omits the pilot wires and connects the secondaries of the current transformers in series with the trip coils directly across the lines.

(C.) 445/09. Edward Farrar. Improvements in classifying and concentrating apparatus. 30.9.09.

This application relates to a classifying and concentrating apparatus, including one or more deep and narrow channels which are of considerable length in proportion to their width, and a horizontally disposed plate or plates so adapted as to divide the stream or streams at any desired point.

(P.) 465/08. Charlton Frederic Bayly. Device for regulating the underflow of cones, spitzkasten and the like. 24.11.08.

This is a device for regulating the underflow of pulp from classifying cones, spitzkasten, etc., and consists of a flat iron side, fitting in a box attached to the apex of the cone or spitzkasten. The slide is fitted with a movable bush of the desired diameter of orifice, which is quickly replaceable when worn or a change of diameter is required, by partially withdrawing the slide, bringing the bush beyond the edge of the fixed box where change of bush may be effected.

(P.) 21/09. Frank Gowine Alfred Roberts (1), James Frederick Ferguson (2), Alwine Rupert Troye (3). Improvements in assay furnaces. 13.1.09.

This invention relates to a combined crucible and muffle assay furnace designed for the use of coal or similar fuel, in which the construction of the partition walls are so arranged that the heated gases from the separate combustion chambers serve to mutually heat both muffle and crucible chambers. This purpose is attained by passing the products of combustion through the heating chambers and returning them under the floor of the crucible chamber and along the side of the muffle chamber.

(P.) 57/09. William Arthur Caldecott. A new or improved substance for precipitating metals from their solutions. 3.2.09.

The above application relates to an improved substance for precipitating metals from their solutions, and consists of employing the product obtained by the application of sufficient heat to any carbonaceous matter capable of yielding hydro-carbon compounds and a residue of fixed carbon (such as coal, peat, tar and the like) in order to reform the hydro-carbon compounds, but insufficient to expel all the hydrogen.

The solution is brought into contact with the above substance in any approved manner, the metal being extracted by such contact, and recovered by subsequent burning and treating the resulting ash, or any other suitable method.

(P.) 105/09. Edward Macartney. Improvements in means for distributing tailings or the like on mine dumps, applicable also for other analogous purposes. 4.3.09.

This invention consists of a large and long Archimedean screw, operated by power and enclosed in a box inclined at any angle to give the desired elevation at the discharge end. This apparatus is laid into the tailings dump and projects over the edge of the dump. It may be fed by trucks dumping their contents into the lower open end of the box or may be in use for the full length of the haulage from the tank to the dump.

(P.) 298/09. John Bartle (1), John Miller Anderson (2). Block and tile liners for launders and tube mills. 19.6.09.

This application relates to specially prepared blocks for tube mill lining and tiles for lining launders. The composition of the above is stated as:—

	Per cent.
Slag (boro sodii silicate with base metal oxides and sulphides)	83·3
Iron Filings	8·3
Pot clay	4·2
Sodium borate	4·2
	100·0

In manufacture the slag is broken up and fed into crucible, the other ingredients are mixed and added together and the whole heated to 1,050° C. and run into moulds to whatever shape may be desired.

(C.) 322/09. A. A. Lockwood. Treatment of ores and carboniferous earths and to the treatment of oils to this purpose. 9.7.09.

This specification describes and claims the process of treating ores with an oily liquid and insoluble metallic salts, soap, magnetic iron oxide and sulphate of alumina.

(C.) 433/09. William Stansfield (1), John Joseph Hatt (2). Improvements in and relating to magnet windings of electro-magnetic apparatus. 24.9.09.

This invention has for its object the maintenance of either constant magnetisation or constant voltage in dynamo electric machinery. It is specially applicable to generators for lighting railway trains.

For these purposes the applicants make use of additional exciting coils in combination with resistances composed of selected materials having specially suitable temperature coefficients. They also describe and claim special arrangements of armature conductors in combination with the fields produced as above described.

(C.) 457/09. John Hugh Fortescue. Roller mill concentrate crusher. 6.10.09.

This refers to a roller mill for finer crushing of sands and concentrates consisting of three pairs of steel rolls, one pair above the other. The claims are for the "jigger" table screening apparatus separating sands from slimes, and for springs behind the rollers and set screws for adjusting same.

(P.) 487/08. Alexander John Arbuckle. Improvements in tanks or vessels for separating comminuted solid matter such as pulverised ore from liquid. 11.12.08.

This application relates to the construction of vats or vessels, the main features consisting in providing inside the vat or vessel a certain number of inverted hollow cones, so as to provide a greater settling surface for the deposition of the pulp under treatment, and also in combination with the above an arrangement of upright cones arranged between the uttermost inverted cone and the inside of the vessel. An angle of about 60° is given to the depositing or settling areas thus produced. The patent also includes the form of apparatus in which the vessel comprises an upper cylindrical portion and a lower inverted truncated conical portion containing an internal cylinder and a plurality of inverted truncated cones arranged inside the internal cylinder and cone, as well as an overflow launder to carry the clear liquids obtained during the working of the apparatus.

(C.) 472/09. Bernard Alexander Firth (1), Herbert Henry Ashdown (2). Improvements in or relating to shoes for mining stamps and the like. 16.10.09.

This application relates to the manufacture of shoes for stamp mills, and proposes the use of a grade of metal in the shank differing from that used for the body of the shoe, a comparatively soft metal being used for the former as being less liable to fracture, while a hard metal is used in the body to give greater resistance to wear.

The applicants propose to make the shanks separate pieces, and to secure them to the bodies of the shoes by casting or forging the latter around them and thus securing a permanent connection between the two portions. They claim that this method enables a selection to be made giving hard metal for the body of the shoe and soft tough metal for the shank.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Blaeser, C. A., *to* P. O. Box 3634, Johannesburg.
Cooke, R. C. H., *l/o* London; Queen's Hotel, Salisbury, Rhodesia.
Dodds, J. B. K., *l/o* Springs; New Heriot G. M. Co., Ltd., P. O. Box 10, Cleveland.
Douglass, Ross E., *l/o* California; P. O. Box 162, East Eley, Nevada, U.S.A.
Grierson, N. V., *l/o* W. Australia; Uruguay Consolidated G. M. Ltd., Cūnapiru, Departmento of Rivera, Uruguay, South America (*via* London).
Nalty, F. J., *l/o* Bulawayo; Post Office, Gatooma, Rhodesia.
Nicholls, J. C., *l/o* Benoni; Dutch Flat, Placer Co., Cal., U.S.A.
Price, W. S. V., *l/o* Boksburg; Angelo Deep G. M. Co., Ltd., East Rand.
Shallcross, V. F., *l/o* Sydney; P. O. Box 206, Kalgoorlie, Western Australia.
Strout, E. A., *to* Hotel de Genère, 8a Calle de Liverpool, No. 133, Mexico, D.F.
Thomas, G. G., *l/o* Johannesburg; Post Office, Sabie.
Thurlow, J. R., *l/o* Johannesburg; P. O. Cleveland.
Wessels, J. J., *l/o* Nigel; Robinson Deep G. M. Co, Ltd., P. O. Box 1488, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. X. JANUARY, 1910. NO. 7.

Proceedings
AT
Ordinary General Meeting, January 15, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, January 15th, Mr. Chas. B. Saner (Vice-President) in the chair. There were also present:—

49 Members: Messrs. W. R. Dowling, A. Richardson, R. Stokes, J. E. Thomas, H. A. White, R. G. Bevington, W. A. Caldecott, A. F. Crosse, E. H. Johnson, S. H. Pearce, R. Allen, G. S. S. Anderson, A. Avent, F. W. Bentley, A. J. Bowness, W. Broom, D. Coll, M. H. Coombe, M. J. Doyle, J. F. Ferguson, G. Goodwin, J. Gray, C. Hunter, A. J. Johnson, T. Johnson, G. A. Lawson, H. Lea, J. Lea, R. MacGregor, C. E. Meyer, P. T. Morrisby, W. Nicklin, F. B. Ogle, F. D. Phillips, W. S. V. Price, J. F. Pyles, A. Salkinson, H. A. Scarf, C. O. Schmitt, C. B. Simpson, S. H. Steels, W. H. Stout, W. A. C. Tayler, J. R. Thurlow, C. Toombs, J. F. Walker, F. Wells, and L. J. Wilmoth.

15 Associates and Students: Messrs. G. F. Ayers, E. Brennan, J. Cronin, D. W. Greig, W. J. R. Hunter, A. King, H. Lomberg, C. J. McCaffrey, J. H. Neethling, N. N. Newland, A. F. Packman, H. G. Roake, J. M. Robinson, J. Sharp, and H. Stadler.

9 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the December *Journal*, were confirmed.

NEW MEMBERS.

Messrs. S. H. Pearce and H. A. White were appointed scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been unanimously elected, as follows:—

HOPKINS, THOMAS, Mount Retief G. M. Ltd., Kaapsche Hoop. Mine Manager.

HOVEN, CHARLES FREDERICK VAN DER, Mount Retief G. M. Ltd., Kaapsche Hoop. Miner.

HOWE, BEN, Redjang Lebong Mine, Benkoelen, Sumatra, Dutch East Indies. Mining Engineer.

IRWIN, ALBERT EDWARD, East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Cyanider.

PIASECK, STANISLAW, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Shift Boss.

ROSS, HAROLD, West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp. Mine Surveyer.

SMITH, GAVIN HILDICK, B.Sc. (Birmingham), Stud.Inst.C.E., F.G.S., Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Chief Surveyor.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associate.—

ROAKE, HENRY GEORGE, Stand 120, Jack St., Germiston. Miner.

As Students.—

BEARDWOOD, JOSEPH PATRICK (Messrs. Littlejohn & Whitby's Laboratory), Rathmines, corner of Marshall and Keizer Streets, Belgravia, Johannesburg. Analytical Chemistry and Assaying.

BRUCE, CHRISTOPHER YULE, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mining.

KENYON, EDWARD GUNNER, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Learner.

GENERAL BUSINESS.

The Chairman: Under the head of General Business I may remark that I represented you at the inaugural meeting of the South African Institute of Electrical Engineers last Thursday night. It was said in the papers that I made a humorous speech. I think that was rather severe on me; however, I did not pat them on the back too much, but on behalf of our Society I wished them well in their work.

Mr. A. F. Crosse (*Past-President*): As I was present at that meeting, I may say that I was very glad our Chairman was there, and I believe he thoroughly expressed the views of many of our members, because there was a general feeling of amalgamation in the air, which I am certain most of us are dead against, at least so far as this Society is concerned.

THE LATE PROF. HILARY BAUERMAN.

The Chairman: I regret to have to announce that we have lost by death, one of our Honorary Members, Prof. Hilary Bauerman, who was a distinguished member of the scientific world in the old country, and whose death will be a great loss not only to ourselves, but to the whole of the metallurgical and mining societies throughout the world.

I am also sorry to have to report that our Corresponding Member in London, Mr. G. T. Holloway, has recently had a very severe accident—I understand he has broken his thigh—and although you will be glad to learn he is almost well again, still, as he takes a great personal interest in the doings of our Society, the least we can do is to ask the Secretary to write and express to him the sympathy of the Society, and to hope that he will soon be quite recovered from the effects of the accident.

TESTS FOR CUPELLATION.

Mr. H. A. White (*Member of Council*): Mixtures of various composition have been proposed by members of this Society for use in making tests for cupellation, and in some cases an account has been given of their durability under working conditions. For purposes of comparison I desire to place on record the performance of a test, made with bone-ash in the old fashioned manner but tamped down and dried out very thoroughly before use. This test was made in September, 1908, and was first used in January, 1909; it was gradually heated up for 24 hours before putting in the lead bullion. After finishing the cupellation for December, 1909, the test was no longer fit for use, as the sides were getting thin and the last hole was worn out. During the life of this test 12 cupellations were performed on it; the lead bullion weighed 273,000 oz. and the gold obtained was 24,491 fine ounces. The test was made and all the cupellation done by Mr. J. Berryman at the May Consolidated Gold Mine.

CORRESPONDING MEMBERS OF COUNCIL.

The Chairman: I have also to mention that the Council has elected as Corresponding Members of Council for Rhodesia, two well known gentlemen who were very active members of the Society and of the Council when resident in Johannesburg, namely, Mr. Clement Dixon and Mr. C. B. Kingston.

Mr. Robt. Allen (*Member*) read the following paper:—

NOTES ON THE PRECIPITATING EFFECTS OF SUBSTANCES CONTAINING VARIOUS FORMS OF CARBON AND CELLULOSE ON CYANIDE SOLUTIONS CONTAINING GOLD AND SILVER.

By A. J. Clark (Associate) and W. J. Sharwood (Corresponding Member of Council).

These notes contain the results of a series of experiments and observations made by the authors at the Homestake mine, Lead, South Dakota, U.S.A., together with other information on the subject, gathered from various sources; also a summary of the literature bearing on charcoal precipitation, as well as a list of patents granted to date protecting the various uses of charcoal as a precipitant.

Experiments on the Relative Activity of some Precipitants. — Two sets of experiments were made on the precipitating effect on gold bearing solutions of various precipitating substances, with the results tabulated below under Series A and Series B. In each experiment 300 c.c. of solution were taken. In Series A 5 gm. of the precipitant were added; the mixture was stirred vigorously for a moment and then filtered, with further agitation each time the funnel was refilled with the liquid. After filtration the filtrate was passed through the filter again, and thus exposed to the sediment retained by the filter a second time. In Series B the conditions were the same as in Series A except that a larger quantity of precipitant (which was not weighed, but was at least 20 gm.) was taken, and there was only a single passage through the filter.

Series.	Precipitant.	Original Assay of Solution.	Assay of Filtrate.	Percentage of Gold Precipitated.
A	Graphite (Dixon's flake)	$2·63	$2·65	0
	Bituminous coal (40 mesh)	,,	2·45	7·0
	Coke (40 mesh)	,,	2·11	19·6
	Charcoal (pine, 40 mesh)	,,	1·92	27·0
	Calcium carbide	,,	2·50	5·0
	Zinc dust	,,	·02	99·3
B	Graphite	2·73	2·73	0
	Coal	,,	2·13	22·0

Repeated tests have shown similar results. Pure graphite, whatever its source, seems to be entirely without effect on the gold in cyanide solutions, and anthracite coal nearly so. The semi-bituminous or sub-bituminous coals of the Western United States (practically bituminous coals but having the high water content and brownish tinge when powdered, characteristic of lignites), have a feeble precipitating action; well burned coke is fairly effective and charcoal much more so. Lubricants containing graphite with mineral oil, intended for use in rock drills, and the clean graphite extracted from them, are entirely without effect after several hours' contact with solution.

Either a sufficiently long contact with, or a slow percolation through, a relatively large amount of bituminous coal, removed as much, in some cases, as 99% of the gold in a cyanide solution, but the efficiency of the coal as a precipitant fell off with repeated use, as is the case with charcoal.

Experiments on the Effect of Wood Ashes containing a small proportion of Charcoal on the Cyanide Extraction of Gold from Tailings containing Pyrites.—These were made in percolators upon pyritous tailings, to which various proportions of ashes were added together with a suitable amount of lime (about 5 lb. per ton), the charges being thoroughly mixed. The cyanide solution was in actual contact with the charges four days, but the charges were drained dry once during their treatment, as well as at the conclusion.

The tailings assayed about $1.75 in gold per ton, and upon them, under normal conditions, without the presence of wood ashes, an extraction of at least from 70 to 75% would have been obtained.

The results of the experiments were as follows :

Percentage of Ashes added to Tailings.	Percentage of Gold extracted by Cyanide.
0·17	52·8
0·8	46·0
1·73	35·8
8·0	3·5

At the Homestake cyanide plants, when similar ashes or charcoal have been inadvertently introduced into the treatment vats, a very small percentage has been found to lower the gold extraction from between 70 and 75% to below 60%. As to the origin of these ashes, in one case it had been traced to an old stope, into which they had been dumped from the surface some years before; in another case charcoal had been produced by the charring of the timbers in a former mine fire, and had become mixed with the ore treated. Sometimes charcoal and cinders have been introduced into the plants, by the dumping into the mill launder of ashes from a heating stove, as a convenient means of disposal ; at other times accumulations of ashes and clinker, produced by the burning of chips (collected from the mortars), had been run through the batteries.

Some years ago heavy losses were reported in the cyaniding of ore at an Australian mine, at which crushing and amalgamation were facilitated by the device of heating the ore by burning it like limestone in a furnace with alternate layers of cordwood, and quenching the heated ore with water. A similar kiln treatment has been used successfully at some small California mines, rendering refractory ore containing pyrite and tellurides more amenable both to crushing and amalgamation in an arrastre, but it seems to have proved fatal to cyaniding in the former instance, probably owing to the presence of charcoal. Park (in "The Cyanide Process," 1897 ed.) states that in New Zealand it was found advisable to hand-pick charcoal from dried ore prior to crushing and cyaniding.

Several instances have been reported where "graphitic" ore (usually slate or schist) has caused reprecipitation. Most likely this is due to the fact that the carbon in such carbonaceous schists is often far from being completely converted into graphite, and may exist largely in the form of hydrocarbons, in a condition similar to that of partially burned coal or wood.

Precipitating Effect of Chips, Charcoal, etc., upon Gold and Silver bearing Solutions.—Some floating rubbish, consisting of charcoal and charred pieces of wood, as well as some clean chips and twigs (which came from a settling dam and creek bed) and particles of coke (which came from the smoke stacks of the locomotive and steam shovel) was skimmed from the surface of the cyanide solution in a slime agitator vat.

The solution contained approximately :—

Gold, 0·023 oz. per ton ($0.47)
Silver, 0·28 oz. per ton ($0.14)
Potassium cyanide, 0·025%.

The skimmed rubbish was rinsed twice with water, and dried at 100° C. It was then divided into two portions, one containing the charcoal and charred wood, and the other the clean chips and twigs. The two portions were then weighed, some of each being burned in clean roasting dishes in a muffle. The substances were assayed, before and after burning, with the following results :—

No.	Material.	Proportion of Total Weight of Original Sample.	Ounces per Ton.		Values per Ton.		
			Gold.	Silver.	Gold.	Silver.	Total.
1	Coke, charcoal and charred wood...	40%	2·25	7·50	$45.27	$3.75	$49.02
2	Ash of same ..	$\frac{1}{8}$ (or 25% of total ash)	18·00	60·0	$365.40	$30.00	$395.40
3	Clean chips and twigs	60%	0·24	2·70	$4.87	$1.35	$6.22
4	Ash of same ...	$\frac{1}{4}$ (or 75% of total ash)	0·96	10·8	$19.48	$5.40	$24.88

As neither of the two classes of material carried more than its own weight of solution, when removed from the vat, obviously a considerable precipitation had occurred in proportion to the weight of the material, the coke, charcoal, etc., precipitating the gold contents of approximately 100 times their weight of solution, and the twigs, etc., precipitating ten times their weight, the corresponding amounts for silver contents being 27 and 10 respectively.

In reply to a suggestion by Mr. Caldecott that these chips, etc., might have been partially burned or charred, it may be stated that they were to all appearance unaltered. It is possible, however, that some of them had lain in the dam or creek for months, or even for several years, so that actual decay may have commenced.

Decaying Wood.—That decaying wood is an active precipitant for gold in cyanide solution was evident from a series of samples taken from a residue dam, the sand discharged into which contained appreciable amounts of solution as well as traces of dissolved gold, so that the aeration consequent on discharging no doubt resulted in the dissolution of a little more gold. While the average assay value of the sand was lower than that of the residues as discharged, the lower strata were slightly higher in gold contents, as might be expected from the leaching that had since taken place. At two points, however, a decided concentration of values was noted; one of these was found to be in the neighbourhood of a pine post, and another around the decaying stump of a tree that had been killed by the residues. Mr. Caldecott's suggestion that the effect of the wood fibre itself is practically *nil*, and that apparent precipitation by wood is due merely to absorption of solution, is confirmed by our next series of determinations.

Absorption of Gold and Silver by Cellulose.—Several writers have published experimental results[*] showing a large decomposition of cyanide and also a considerable precipitation of precious metals, due to contact with wood, canvas, cocoanut matting and other fabrics used as filters—all forms of cellulose. Some of the writers, working on a laboratory scale, had failed to take into account the ratio of the surface of the precipitating material they used to the volume of solution experimented upon, and did not then compare this ratio with that which would have resulted by working on a large scale. Consequently some operators have been misled with believing that discrepancies in clean-ups were attributable to precipitation by the wood of tanks or filter bottoms, and therefore to hope, that, at some future date, the burning of the material would reimburse them for their supposed losses.

As the writers referred to differed considerably from one another, either as regards their opinions or in experimental results, we give some of our results obtained by examination of various materials that had been exposed to gold and silver bearing solutions for considerable periods.

(1) *Materials from a Sand Leaching Tank.* This was nearly 40 ft. in diameter, the bottom having an area of 1,200 sq. ft. It had been in use over a year when the first tests were made. The values given represent the contents of the entire material in the tank, through which many thousands of tons of solution had circulated, carrying up to \$6 per ton in gold, and up to \$2 silver.

(*a*) *Bottom Filter Cloth of Canvas.*—8 oz. duck, 1,200 sq. ft., in use for 8 months. Entire cloth: gold, 5 cents; silver, 2 cents.

(*b*) *Bottom Filter of Cocoanut Matting*—1,200 sq. ft., in use 12 months. Entire filter: gold, 46 cents; silver, 3 cents

(*c*) *Wooden Slats.*—These supported the filter and were of unpainted Oregon pine, 3,800 running feet with a cross-section of 2·5 × 1·5 inches, about quarter of the material being crozed out; altogether about 75 cub. ft. In use 12 months. Entire material: gold, \$16.60; silver, \$5.90. Wood per cubic foot: gold, 22 cents; silver, 8 cents.

(*d*) *Fine Slime.*—This had accumulated under and between the filters during eight months, and consisted of fine ore together with precipitated carbonate of lime; altogether 200 dry pounds. Entire material: gold, 20 cents; silver, 4 cents.

In the last case, and probably in all, most, if not all, of the values were contained in solution absorbed and not necessarily precipitated.

(2) *Tank Staves.*—(*a*) The staves of a similar large leaching tank, after use between three and four years with sand averaging \$3 gold and \$1 silver value per ton. They were of California redwood, originally painted inside with two coats of P. and B. paint, and outside with red oxide of iron in linseed oil. A slight coating of carbonate of lime had formed on the inside. A large sample from these staves, burned and assayed, yielded only traces of gold and silver.

(*b*) *Similar Staves* from a leaching tank in use at another plant for four years contained 40% moisture. The dried wood contained: gold, 47 cents per ton: silver, 3 cents per ton.

(3) *Precipitating Box.*—A precipitating box of $1\frac{1}{2}$ in. lumber was painted inside and out with P. and B. paint, and used three years to precipitate low-grade cyanide solutions with zinc shavings, the slimes averaging about \$4 to \$5 per lb.

---

[*] A. Simon, *Min. and Sc. Press*, Dec. 2, 1893, pp. 67-356.
F. L. Bosqui, *Eng. and Min. Jour.*, Feb. 26, 1898, pp. 65-248.
A. Forsyth and G. H. Clevenger, *S. Dakota School of Mines, Bulletin 3*; also *Min. and Sc. Press*, Sept. 9, 1899, pp. 79-285.
A. N. Mackay, Trans. *Inst. Min. and Met.*, 1905, pp. 14-41; also *Min. and Sc. Press*, June 17, 1905, pp. 90-387.
S. H. Williams, *Eng. and Min. Jour.*, Sept, 9, 1905, pp. 80-435.

Samples were taken by boring completely through the wood at various points in sides and bottom, and were burned and assayed. A little zinc slime was visible on the surface.

The average value of the material, including a trace of adhering precipitate was: Sides, 0.75 cent per sq. ft. Bottom, 2 cents per sq. ft.

(4) *Mine Timber.*—A piece of blackened but sound timber (yellow pine) which had been some time underground, and contained ferrous and ferric sulphates derived from mine water was tested. At the time of the test it contained 56% of water. Kept in contact with about three times its weight of cyanide solution, which assayed $1.20 per ton gold value, no precipitation took place in 20 hours. It seems evident therefore that wood, when well painted with any suitable protective medium, absorbs little more than traces of precious metal from cyanide solutions of ordinary value.* As dry pine and redwood absorb about their own weight of water, one would expect such wood, if dried after a thorough soaking in solution, to assay the same value per ton as the solution taken up—evaporation and precipitation excluded. Alternate soaking and drying, or evaporation on the outside of unpainted wooden launders or tanks in a dry climate, might be expected to result in a distinctly higher value in the wood. Adhesion of rich telluride slime or floured amalgam may also cause an increased value.

It is difficult, however, to reconcile with one another the results obtained by the following authors, and only those of Bosqui are at all in line with our own experiences. The other writers named worked entirely on a laboratory scale.

S. H. Williams found that an 8 in. wooden launder, 75 ft. long, used seven months at a cyanide plant in New Mexico, having passed 63,000 tons of solution, worth $3 per ton, contained: 2·766 oz. gold and 32.646 oz. silver in 15·5 cub. ft., or 0·18 oz. gold and 2·1 oz. silver, per cub. ft., or a total value of about $4.65 per cub. ft. An apparent typographical error in the original article makes the value $60.45 per cub. ft.

A. N. Mackay states that red pine, from a tank used 18 months, in which had been treated only 1,200 tons of ore containing 20 dwt. gold per ton, yielded 12.4 oz. gold per ton of wood.

F. L. Bosqui experimented with pine and redwood, dressed and rough, both with intermittent exposure and drying, and with continuous immersion, for periods up to 12 months, using solution which averaged $5 gold per ton and more or less silver. The highest absorption was with rough pine, exposed altogether about 12 months, this assaying $5.18 gold and 14 cents silver per ton of wood.

On the whole, it seems probable that the precipitative effect of charcoal may be occluded gases, but definite proof is lacking. S. Brussow, in a recent article, attributed the precipitation of gold from haloid salts by charcoal to adsorption. This seems most unlikely, as he admits that the chlorine of gold chloride remained "almost entirely" in the solution; and we have found that charcoal, which has been in contact with solutions of either chloride or double cyanide of gold, is thoroughly impregnated with metallic gold, easily distinguishable with a good lens or low-power microscope.

Literature Relating to Charcoal as a Precipitant.—

Gaze: "Handbook of Cyaniding," p. 121, *et seq.*

N. D. Power: *Australian Mining Standard*, Oct. 14, 1897, p. 2305.

H. Watson: *Australian Mining Standard*, June 22, 1899.

W. B. Gray: Trans. *Australian Institute of Mining Engineers*, 5, p. 138; *Mining Industry*, 7, p. 327.

W. D. Johnston: *Bulletin* 5, *California State Mineralogist* (Schneidel), p. 133.

G. I. Lowles: Trans. *Institute of Mining and Metallurgy*, 7, p. 190; *Mining and Scientific Press*, July 29, 1899, 79. 122; and Sept. 26, 1905; Proc. *Chemical, Metallurgical and Mining Society*, 2, p. 897.

R. B. Lamb: *Pacific Coast Miner*, July 25, 1903, 8, p. 62.

Anon.: *Mining and Scientific Press*, Nov. 4, 1899, 79, p. 517; Oct. 1, 1899, 79, p. 458; Feb. 2, 1901, 82, p. 71; Sept. 24, 1902, 89, p. 204.

S. B. Christy: Trans. *A.I.M.E.*, pp. 26-748.

S. Brussow: *Zeit. Chem. Ind.*, Kolloide, 1909, p. 5-137; see *Journal of the Chemical Industry*, Oct. 15, 1909, p. 28-1040.

W. A. Caldecott: *Journal of The Chemical, Metallurgical and Mining Society of South Africa*, Vol. IX., April, 1909, p. 327.

Patents Relating to Charcoal Precipitation Claims.—

W. M. Davis (U.S.P. 227,976, 1880). The use of granulated carbon of any kind as a precipitant for gold in aqueous solutions generally (presumably referring to halogen salts).

---

* When launders of unpainted wood are used in the Plattner chlorination process they absorb gold enough to make it worth while to burn them to recover it. The same is true of canvas used for concentration in mills where floured amalgam escapes the plates, or where the tailings contain fine gold or rich tellurides. Absorption of silver similarly takes place in hyposulphite leaching. This is preventable by soaking the launders, etc., in hot paraffin or painting with asphaltum.

Dr. W. G. Johnston of San Francisco (U.S.P. 522,260, 1894). A series of filters containing charcoal.* Aitken, of New Zealand, applied for a patent about same time, 1893.

H. B. K. Davis (U.S.P. 546,113). Diffused hydrogen through a solution and then passed it over charcoal filter.

J. C. Montgomerie (U.S.P. 580,948). Use of charcoal, to be regenerated by heat.

J. C. Montgomerie (Brit. Pat. 29,721 of 1897), and E. J. Fraser (U.S.P. 543,546). Filters of alternate layers or beds of charcoal and zinc, evidently to form a galvanic couple. (The latter's system was used for some time at one mine in California).

G. O. Pearce (U.S.P. 556,690). Charcoal saturated with ferrous sulphate, or oxalic or tartaric acid.

M. Crawford (Brit. Pat. 13,537 of 1895). Charcoal with sulphate of iron or of aluminium. A similar German patent (87,005) was issued.

C. H. Gage (U.S.P. 719,207). Bed of charcoal containing sulphate of iron, "subjected to an electric current."

S. B. Christy (U.S.P. 883,170). Pervious electrode of powdered charcoal, or carbonised wood in the form of excelsior, etc., to be used in frames as a series of intermediate electrodes—one side forming anode and the other cathode—between the principal electrodes.

Hay and Tennent (French Pat. 399,877, 1909). Heating wood charcoal in a current of steam in a muffle, claiming to produce hydrogen carbonyl in the charcoal, making it 20 times more efficient.

The Chairman: We must thank Messrs. Clark and Sharwood for submitting this most interesting paper. When a corresponding member of our Society sees fit to write, in conjunction with one of our Associates abroad, such a valuable paper and send it all the way from the United States here I think he most heartily deserves our thanks. It is a good example, which might be more readily followed by the members of our Society who are resident in South Africa, and who, I hope, will try to emulate them by giving us equally interesting and valuable papers on present day practice on the Rand, and other centres.

Mr. Jas. E. Thomas (*Member of Council*): I beg to second the vote of thanks to the authors of the paper, and I should like to mention that in 1905, when we were taking down the wooden vats on the Simmer and Jack Proprietary Mines, Ltd., and replacing them with steel vats, one of them was reduced to ashes. It was built of Oregon pine. There was about a ton of ashes, and it contained about 5 oz. to the ton. I think that confirms the statements in the paper.

THE CONTINUOUS COLLECTION OF SAND FOR CYANIDING.

(Read at August Meeting, 1909.)

By W. A. Caldecott, M.Inst.M.M. (Past-President).

REPLY TO DISCUSSION.

Mr. W. A. Caldecott (*Past-President*): In replying to the discussion upon my paper I desire in the first instance to acknowledge the appreciative remarks of our Past-Presidents, Mr. J. R. Williams and Prof. Yates, upon the sand filter tables at the Simmer Deep, after they had seen them in operation.

In regard to Mr. H. A. White's comments, I am afraid that some confusion has arisen between the Simmer Deep-Jupiter sand plant handling up to 1,600 tons of sand daily, and the ideal sand plant dealing with double that tonnage of sand. The use of 26 vats in place of 48 refers to the former plant, whilst the estimated costs given in the paper of erection on the ordinary and continuous sand collecting system refer to the latter. Only a summarised estimate was quoted in the latter case, without occupying the *Journal* space with columns of details, because any company contemplating the use of sand filter tables would obviously make its own estimates of costs, and it was unlikely that the conditions of the case cited would correspond.

As regards the East Rand plants to which Mr. White refers, it may be pointed out that whilst the use of sand filter tables in such cases affords a much improved sand product and greatly increases the treatment capacity, yet the diagrams IIa. and IIb., given in the paper, clearly show that a single treatment of sand completed in the vat wherein it is collected is longer by three days than by the transfer system, preferably from an upper to a lower vat (see Fig. V.)

The cost of operation of the sand filter tables at the Simmer Deep-Jupiter plant is about a penny per ton of ore milled, but as against this may be set in a plant designed for this system, as shown in Figs. V. and VI., (*a*), say, 8% more slime at 6d. per ton less cost of treatment, (*b*) an increased sand extraction (see p. 47), and (*c*) elimination of cost of belt or truck transfer. At a very conservative estimate the foregoing savings are more than ample to cover the 1d. a ton involved, and hence the large saving in capital expenditure is not attended by any increase in total working costs but the reverse. Since the paper was written it has been found possible at the Simmer Deep to gravitate the solution overflowing the sand collectors direct back to the launder beside the table, except when the pulp has been recently turned into a collector, and by this means capital expenditure is still further reduced, as only one solution storage is needed in place of the two shown in Fig. VI. in the paper.

---

* Johnston claims that one filter extracts 25% of the gold from a cyanide solution (see Christy, Trans. *A.I.M.E.*, pp. 26-748); W. B. Gray states that in Victoria five filters, in series, each with 560 lb. charcoal will precipitate respectively from solution flowing at 1 ton per hour, 45%, 25%, 15%, 9% and 5%.

As regards the moisture in the collected sand Mr. White is in error in calculating the percentage on the total ore instead of the sand only, but in any case it must be remembered that in the Simmer Deep installation there is at present no spare table, so that the sand beds between the ploughs and filter-cloths are not changed as often as would otherwise be the case ; hence the moisture percentages as low as 12%, which the tables are capable of yielding in practice have not hitherto been regularly attained, and the figures quoted in the paper represented the actual results under existing conditions.

Our President has quoted figures showing satisfactory dissolution of gold in the slime plant in spite of the – 200 mesh sand contained in the slime.

Mr. C. O. Schmitt's contribution, confirming various statements made in the paper, calls for little further comment. The following gradings of Simmer Deep sand collected on the filter tables and Knights Deep sand collected by the usual method of hose-filling show that only two-thirds as much – 200 mesh product is contained in the former sand as compared with the latter.

Meshes per Linear Inch.	Simmer Deep-Jupiter (Filter Tables).	Knights Deep (Hose-filled).
+ 60	14·5%	13·5%
– 60 + 90	28·1	32·1
– 90 + 125	17·0	16·4
– 125 + 200	27·1	17·5
– 200	13·3	20·5
	100·0	100·0

It may be added that finer crushing, owing to the recent addition of two tube mills to the Simmer Deep-Jupiter joint crushing plant, has resulted in the percentage of slime now collected being raised to 44%, whilst better settlement of slime in the collectors, owing to the presence of – 200 mesh sand, has caused the percentage of moisture in the collected charge to be reduced to 42·5%.

Both Mr. W. R. Dowling and Mr. C. O. Schmitt have laid stress upon the use of the diaphragm in cone classifiers, and the latter has asked for further information thereon. There is no doubt that this device has largely contributed to the success of the continuous sand collecting system for the reasons stated by the former gentleman. It is difficult to say much more in description of so simple an appliance as a disc of wood than has already appeared in the columns of our *Journal*, but the writer anticipates that the advantages it possesses in its various applications will in time become generally realised for the reason that the extent of adoption of any metallurgical device ultimately depends upon its intrinsic merit.

Diaphragms are now in regular and satisfactory use in Mexico in a dozen tube mill cones for classifying and dewatering. With Simmer and Jack current ore slime pulp it has yielded a steady underflow, containing 39% moisture in a cone 20 ft. in diameter. All the former numerous pyramidal and conical classifiers with launders and dewatering cones at the Simmer and Jack and Simmer Deep-Jupiter joint plants have lately been replaced by a single 9 ft. × 6 ft. cone for each tube mill with an 8 in. diaphragm supported by a cross with 13 in. arms, whilst the classification at the Simmer East plant is likewise now carried out by cones fitted with diaphragms. It is to be hoped that Mr. G. O. Smart will shortly give us details of the results of the Simmer and Jack installation. The use of diaphragms in conical bottomed dissolving vats (10 ft. diameter by 18 ft. deep) for agitating battery or ordinary sand by centrifugal pump or air-lift has enabled charges up to 36 tons to be readily treated, or rather more than double the usual capacity of the ordinary Brown vat of such dimensions. Beyond its use for the above purposes, it is not unlikely that its application may be extended in various countries to classifying and dewatering the pulp from various base metal ores, such as copper, tin, zinc and lead, prior to concentration or regrinding or elevating. In addition, its use is possible in classifiers generally where water is scarce, to permit a thick tailing underflow to run to waste, whilst returning a clear overflow for re-use.

The Chairman : On your behalf I will thank Mr. Caldecott for his reply. The only fault about it is that it is just as good as the original paper, and it should be open for further discussion.

A METHOD FOR THE RECOVERY OF ZINC FROM SOLUTIONS OF SULPHATE.

(*Read at September Meeting, 1909.*)

By Wm. Cullen (Past-President) and G. F. Ayers (Associate).

REPLY TO DISCUSSION.

Mr. G. F. Ayers (*Associate*): We are glad that this short paper has been criticised, though we should have liked to have had more of it, for there are certain points upon which we ourselves were anxious to have more information.

To Prof. Stanley and Mr. Meyer we are indebted for their analysis of the economic aspect, and our reply will practically be directed to these remarks.

There is no doubt a large amount of recoverable zinc goes down the gutter at every clean-up. That is the one incontrovertible fact, and we see no reason, after having again gone into our figures, for amending our original estimates of this amount; indeed the closer we look into them the more convinced are we that our own estimate is fairly correct. True, it is only an estimate, and we fully expected that definite figures on this essential point would have been forthcoming. Might we suggest to cyanide managers that this is a point which is well worth looking into, and one which could easily be settled notwithstanding the "harry scurry" of the clean-up.

Both Prof. Stanley and Mr. Meyer think that our estimate of recoverable zinc is too high, but Prof. Stanley's argument is based on hypothetical percentage of gold present in the slime, which we are not prepared to accept, while Mr. Meyer offers no figures nor other basis of calculation, to substantiate his reason.

Prof. Stanley, in the course of his remarks, says:—"It is questionable whether any but the largest producers would consider it worth while to precipitate their zinc-bearing waste solutions, since, over 70 producing mines, the recoverable zinc will only average about 2 tons per mine." Now although there is a certain amount of truth in this contention it would also apply "pari passu" to his own proposal to precipitate electrolytically. However, it is perfectly obvious that, quite apart from any particular process, the absolute saving on a large mine would be greater than on a small one, but there is nothing to prevent a number of the small mines (or for that matter the whole of the mines) combining and carrying out the recovery at one centre, in the same way as is now done with bye-products of other classes. The carriage in tank wagons is wonderfully cheap and we could easily devise a plant, for the handling of the solutions, which would cost next to nothing. The objection that the solutions are acid could be obviated by neutralising the acid with magnesite before it is transferred to the wagons. We do not think any central precipitating plant is at all necessary as most of the mines have old tanks of one kind or another and the precipitate could have a whole month (or more, if solutions were accumulated) for treatment. The process is so simple that any unskilled person could carry it out after the necessary assays were done. We are not certain that all the precipitated zinc could be absorbed locally for paint, but a large proportion of it undoubtedly could be, as the importations are very heavy and the price is very high.

However, accepting Prof. Stanley's figures for the moment, we arrive at the following:—"Saving *per mine*, if the precipitate be smelted locally, on the basis of 2 tons recoverable zinc per mine":

1·54 tons of magnesite at £7 per ton required to precipitate 2 tons zinc	£10	15	0
Precipitating, handling, etc., of 3·4 tons precipitate (58% Zn) at £1 per ton	3	8	0
Smelting locally of 3·4 tons precipitate at £3 per ton	10	4	0
	£24	7	0
Reckoning on an 85% recovery on 58% Zn in 3·4 tons precipitate 1·67 metallic zinc would be produced local value £31 per ton ...	£51	14	0
Saving per mine per month ...	27	7	0
Saving per mine per annum ...	328	4	0
Saving per 70 mines per annum	23,000	0	0

This saving is hardly a negligible quantity. If it were found feasible to recover the zinc locally, the powers that be might go one step further and treat the very excellent zinc blende found in this country. The Dynamite Co. might even go as far as utilising the sulphur contained therein, a perfectly feasible scheme, which would go a long way towards the cheapening of the process of zinc recovery and zinc supply in this country.

We have already referred to Prof. Stanley's suggestion to recover the zinc electrolytically. Some years ago we went into the matter very fully but dismissed the idea. No doubt cheaper power now obtainable has altered the conditions somewhat, and we are again taking up the matter —we are, however, sceptical about a satisfactory solution being found in this direction.

Mr. Meyer has sorted up the various sources of loss correctly, and applied headings of "chemical" and "mechanical" incorrectly. His mechanical loss (caused by dissolving the zinc out by acid) is evidently a chemical one, while his chemical loss (caused by imperfect washing of the filter press) is purely a mechanical one. His conclusion that we can only recover one-third of the original quantity of zinc used, is practically the starting point of our experiments and calculations.

The Chairman: Mr. Ayers is to be congratulated on his further interesting remarks.

ANALYSES OF GASES FROM BURNING NITRO-GLYCERINE EXPLOSIVES.

(Read at September Meeting, 1909.)

By W. CULLEN (Past-President) and D. W. GREIG (Associate).

REPLY TO DISCUSSION.

Mr. D. W. Greig (*Associate*): The discussion on this paper has been practically nil. We desire to thank Dr. Moir for his kindly remarks. Dr. Moir is such a severe critic of analytical methods that for our method to be uncommented upon by him is cause for congratulation to us.

When commencing these experiments, we had the nitron precipitation method, of which Dr. Moir speaks, in mind; but having no nitron in stock were obliged to abandon it. We will bear in mind the work which Dr. Moir has suggested, but we can only repeat what one of us has said before; there is a great deal more work to be done on this subject, before anything approaching completeness can be claimed.

We would like to correct a misconception which seems to have arisen, owing to some remarks by Dr. Macaulay.

Our experiments were exclusively burning tests. By figures which we have previously placed before this Society, we have proved that the detonation, or normal use of antifume gelatine, produces gases which are at least ten times safer than those from ordinary blasting gelatine.

We felt that it was necessary to clear up this point, as the local press at any rate, has succeeded in confounding the two issues.

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

(Reprinted in this Journal, September, 1909, from the Bulletin of the Institution of Mining and Metallurgy, August 12, 1909.)

By W. A. CALDECOTT (Member).

DISCUSSION

Mr. S. H. Pearce (*Past-President*): I shall commence with an apology to Mr. Caldecott for not having had sufficient time at my disposal to adequately do justice to the classic paper he has given us. I should like to say a great deal about this paper, but I am afraid I shall have to content myself this evening by merely stating that our appreciation of the pioneer work he has done with regard to heavy gravitation stamps has been shown on these fields by their general adoption.

As one of the signatories to the report on standardisation of battery screens, issued by this Society in June, 1906, I have been asked to reply to Mr. H. F. Marriott's remarks in discussing this paper before the Institution of Mining and Metallurgy[*] wherein he deprecates the want of recognition of the value of the standards brought out by that Institution by members of this Society. In this I am assuming that Mr. Marriott is voicing the opinion held by that Institution as a whole, and I hoped to take advantage of Mr. McDermott's presence here this evening to correct the erroneous impression that appears to be held in regard to the matter. A reference to the report issued by this Society will show that "the Committee feel it undesirable to specify certain screens as Rand standards, but prefers to recommend a system of screen description, etc."[†] This report had for its object the enlightenment of mill men as to the difference that existed in the relationship of diameter of wire to number of apertures and its effect on discharge area. It gave tables enabling them to select screens suitable for the work required under various circumstances. For this purpose it was found convenient to classify them under the headings of aperture measurement, rather than the number of holes per inch as adopted in the makers' catalogues. I certainly consider this a rational method, the advantages of which have been demonstrated in our later researches into grading analysis of battery pulps. On the other hand, the I.M.M. standard was compiled for the purpose of enabling comparison of grading analyses to be made all over the world—irrespective of the class of machine that may have been employed to crush the ore, or the kind or size of screen that was used to regulate the output of that machine—in other words, to enable a comparative value of the crushing effect obtained to be measured. Incidentally I might mention that this is also based upon aperture measurements expressed in different terms. To further illustrate the difference between these standards, I would not for one moment suppose that Mr. Marriott ever intended to advise us to crush rock through a battery screen of four holes per square inch with a wire $\frac{1}{4}$ in. thick, simply because that happened to be their standard unit. Nor on the other hand would we desire to depart to any extent from the standards they have laid down, and which we have gladly accepted, for the grading of crushed pulp. It will, therefore, probably be of interest to state that not only has this Society adopted the I.M.M. standard, but also that the Mines Trials Committee in their investigations into tube mill efficiency have used all the available I.M.M. standard screens for their grading analyses, and have gone to no little trouble to obtain the corresponding finer meshes down to 200 by careful micrometer measurements of that already on the market. This was done with the idea of obtaining as accurately as possible the average grading of the finer portions, so that the energy used in grinding beyond the requirements of extraction might be determined. I hope some day that permission will be obtained to submit the results of these investigations to this Society, because it has apparently been demonstrated that a comparison of ground material can be made by averaging the grading analyses and expressing its fineness as a single number, instead of, as at present, comparing a confusing array of percentages through the different meshes usually given.

---

[*] See this *Journal*, Vol. x., Dec., 1909, pp. 218-9.

[†] See this *Journal*, vol. vi., June, 1906, pp. 394.

Mr. C. O. Schmitt (*Member*): With the permission of the Council of this Society the writer would like to make a few remarks on Mr. Caldecott's excellent paper, dealing with the important subject of stamp mills.

The author provides a wealth of information not only in his remarks but also by frequent references to work previously done, enabling the reader to follow him into details, and thus does justice to those workers in the same field who published the result of their labour. It is indeed pleasing to see a writer pay so much attention to the literature on the subject dealt with, and to freely acknowledge work done by others. To do so involves a great deal of work, but as it is a great aid to the reader, and since it breathes the true spirit of unbiassed technical writing it must be welcomed.

Apparently, the author did not wish to enter into details of construction, this being more within the province of the mechanical engineer, and the writer will, therefore, endeavour to supply a few details which, it is hoped, will help to a better understanding of the changes involved by the introduction of the heavy type of stamp battery.

The question of obtaining the maximum capacity from stamp batteries already erected and that of reducing the cost of plants contemplated is naturally of the greatest importance to mining communities such as exist on the Witwatersrand and in other parts of the world. The money invested in stamp mills on the Rand alone, not considering what is commonly called the cyanide plant, probably exceeds £4,000,000, and whilst some of the money may have been recklessly spent, the greater portion has been used to the best advantage, considering the knowledge of the engineers and metallurgists at the time. Approximately 9,000 stamps are now in operation on the Rand, and it is estimated that the erection of a further 6,000 stamps is contemplated. The amount involved for these stamps or their equivalent in other crushing and grinding machines is probably £1,800,000, so that a saving of 30%, as hinted at by the author, although he does not give this figure, represents the respectable sum of £540,000. It is obviously necessary for all those considering the erection of new plants to decide how best to obtain this saving, and at the same time increase the efficiency from other points of view.

The saving that can be effected in new plants is, however, not the only one inasmuch as in many cases where the scale of operation of existing plants is to be increased, it must be determined whether the object in view can be more advantageously obtained by increasing the weight of the existing stamps than by adding more stamps of the same or greater weight. It has been found on the Rand that batteries, after having been in use for from 6 to 12 years, require a considerable amount of overhauling. In the writer's opinion this is the time when consideration should also be given to the possible future increase of the tonnage to be handled, so that provision can be made for meeting such future demand in the most efficient way. In a case of this nature it will probably be found that a very small amount of extra expenditure beyond that contemplated will permit of using heavier stamps when necessary. For the purpose of illustrating this point, the writer would like to refer to the work recently done on the stamp battery of the Simmer and Jack Proprietary Mines, Germiston, where it was found necessary some time ago to seriously consider the re-conditioning of the mill foundations. This work could have been done more or less successfully by patching up the existing foundations, but in order to meet a possible future increase of the tonnage to be milled, if considered advisable, it was decided to do the work so as permit of an increase in the weight of the stamps. Figs. I. and II. show the battery foundations before and after the alterations.

The stamps on this mine have a new weight of 1,250 lb., and an average running weight of 1,150 lb., while after the alterations as shown it is considered that an average running weight of 1,400 lb. will be permissible, taking into account the size of camshaft and stem. This represents an increase of 21·74% on the old weight, and will permit of a similar increase in the stamp duty. In addition to the mere increase in stamp

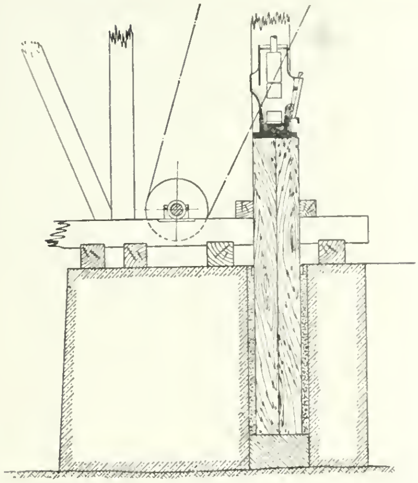


Fig. I.—Foundation of battery before alterations, Simmer and Jack Proprietary Mines.

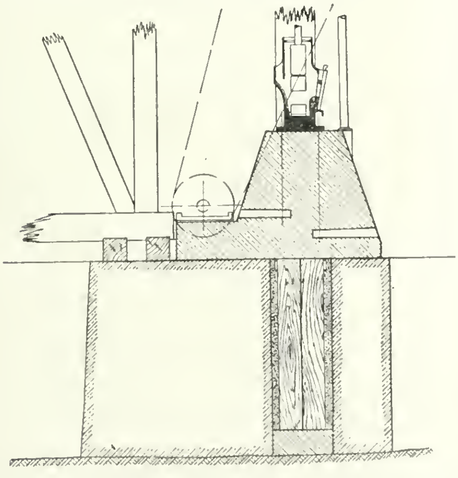


Fig. II.—Foundation of battery after alterations, Simmer and Jack Proprietary Mines.

duty a greater efficiency is to be expected since, as the author points out, the heavier stamp is more likely to reduce the maximum size of ore fed to the mortar box, say, 2 in. cubes, in one blow to the fineness required for passing through the screen.

It should be pointed out that the sketch (Fig. II.) is merely schematic and does not show all the alterations made. Besides providing the mortar box with a substantial concrete foundation, the kingposts were cut off and are now mounted on heavy cast-iron shoes which permit of holding the posts firmly in place. The bearings of the mill line shafting are also provided with new foundations, thus making them independent of the ore bin and assuring a true line for this shafting quite independent of whether the bin is empty or full. The camshaft platform was also rebuilt, and it is now carried on columns standing on the foundations, thus avoiding the vibrations always present if the said platform is supported by the kingposts. This portion of the work, although not affecting the crushing capacity of the mill is, nevertheless, very important as it enables the operators to perform their work in comparative ease and comfort.

The cost of doing this work was £1,190 for 20 stamps, or £59 10s. per stamp. At least two-thirds of this amount is chargeable to the old plant as being necessary repair work, and one-third to the weighting up of the stamps. Since the increase in weight is about 20%, one new stamp of 1,400 lb. running weight would have been necessary for every five stamps thus weighted up. The cost of one new stamp would have been £240, but the cost of improving five old stamps is only $\frac{5 \times 59 \cdot 5}{3}$, say, £100, to which should be added £20 for obtaining the extra 250 lb. on five stamps, or a total of £120, or a saving of £120, equal to 50% on the cost of a new plant. In addition to the saving on capital account there will also be a reduction in the cost of labour and maintenance, thus favourably influencing working costs.

The author points out that the only way to crush more ore to the same degree of fineness is to strike a heavier blow, and so far the increased weight of the stamp in every case resulted in a higher duty, of course, with the same screen; or in other words, if the duty is to remain the same a finer screen can be used and a finer product is the result. It is well known that with increased fineness of the screen product the efficiency of the stamp rapidly drops.* This does not really affect the heavy stamp as such except that when it is possible to use a coarser screen resulting in a greater duty the stamp more oftens falls on large pieces which results in greater efficiency as the heavy stamp is more likely to reduce the ore in one blow to a size suitable for fine grinding in a secondary machine such as the tube mill. This point has been referred to by the author, who also states that the limit in the size of the ore fed to the mortar box is probably 1¾ in., even for the heavy stamp. The writer is inclined to think that a size up to 2½ in. might be admissable for stamps of 1,700 lb. and above, but this point can only be settled by experiments on a large scale.

In order to obtain the maximum efficiency from a stamp it is necessary to make arrangements for the immediate removal from the mortar box of all particles suitable for regrinding in a secondary machine, as soon as the stamp has done its work, and this is accomplished with the aid of the coarse screen and an efficient water feed. Tests recently carried out on the Rand have proved this conclusively, and the writer, who has referred to this in another place,* trusts that the author will find it convenient to enlarge upon this point in his reply.

---

* See H. Stadler, "The Computation of Crushing Efficiency of Fine Grinding Machines," *Journal of the South African Association of Engineers*, vol. xiv., p. 107.

Referring once more to the maximum size of the ore suitable for feeding to the mortar boxes, the writer wishes to point out that this not only concerns the stamp mill, but it is also of vital importance to that part of the reduction plant doing the preliminary breaking, viz., the crusher station. It is admittedly difficult with the large crushers now in general use to regularly reduce the ore coming from underground to a size not exceeding 1¾ in., and this difficulty must be expected to increase as larger crushers are used as a result of the greatly increased tonnage now handled by modern plants.

Another point which, as far as the writer knows, has not been previously referred to is the manner of dealing with the fine ore particles washed off the larger pieces on sorting table or belt. These are usually collected in special settling vats provided at the sorting stations and are, as a rule, periodically sent to the stamp battery. However, it must be obvious from the foregoing that the stamp mill is the last place to which they should be sent, and a more suitable method of dealing with this product is to send it direct to the secondary crushing or fine grinding machine. This was provided for when laying out the new plant now jointly used by the Simmer Deep, Ltd., and the Jupiter Gold Mining Co. If this product is sent to the stamp mill not only is the latter unable to efficiently deal with it but extra water is required to again wash it out of the mortar box.

The maximum weight attainable by the ordinary gravitation stamp is obviously determined by the limit in size of camshaft and stem, and also by the increased wear on the faces of cam and tappet. Up to a certain weight, which the writer would tentatively place at 1,750 lb., there should be an increase in the efficiency of the stamp battery, while above that point the benefits to be derived from the extra weight will be greatly minimised by the extra expense in first cost and particularly in maintenance. Another point not to be ignored is the fact that the individual parts of the stamp become too heavy for convenient handling by the operators and mechanical devices are necessary, which also tend to increase the capital outlay. None of the plants employing stamps of over 1,750 lb. weight have been running a sufficient length of time to permit of final conclusions, but observations made on batteries employing stamps up to 1,700 lb. in weight lead to the conclusion that it becomes increasingly difficult to obtain the high class material needed in order to avoid excessive size. This particularly applies to camshafts but in a minor degree also to the stem, cam and tappet.

In the following remarks, and with the aid of a few sketches, the writer will endeavour to show the developments that have taken place in the various parts of the stamp mill to meet the severe conditions that have arisen from the increased weight of the stamp. In order to facilitate comparisons the principal dimensions of the most important parts of the stamp are given in the table (see opposite page), while in the sketches only the design is indicated.

Beginning with the shoe and die, practically the only change brought about by the introduction of the heavy stamp is in the diameter, and there the increase in size is by no means very marked. There is undoubtedly a lack of information as to the relation between the falling weight of the stamp and the diameter of the shoe and die, and it appears to the writer that engineers have in the past made the shoe of the same diameter as the head without considering what would happen if the shoe were made smaller. It is evident that experiments should be made to determine the proper diameter of the shoe for a given weight of stamp. The length of the shoe and the height of the die are, of course, determined by the consideration of their respective lives taking into account that on the Rand, for every ton crushed with a comparatively fine screen, the wear of the shoe amounts to about 0·4 lb., and that of the die to about 0·3 lb., or a total of 0·7 lb. This amount of wear per ton of ore crushed is considerably reduced when using a coarse screen, say, from 9 to 36 mesh per sq. in. It must also be considered that the weight of the shoe is a wasting factor, which must be compensated for in a manner similar to that indicated by the author.

The stem has always had a length of approximately 17 ft., and the heavier stamp only required a larger diameter, as indicated in the table. It is perhaps not out of place to refer to the habit of millmen in general to increase the length of the stem by welding on pieces of varying length, thus adding to the weight of the stamp. This practice is not to be recommended for imitation as the weight so added is not very effective from the crushing point of view owing to the elasticity of the material.

The head is a vital part of the stamp, and its importance has been more fully realised since the

---

* See *Journal of the South African Association of Engineers*, vol. xv., p. 113.

TABLE SHOWING WEIGHT AND SIZE OF MILL PARTS FOR VARIOUS WEIGHTS OF STAMPS.

Name of Mine.		Simmer and Jack Proprietary Mines, Ltd.	Knights Deep, Ltd.	Simmer East, Ltd.	Simmer Deep, Ltd.	City Deep, Ltd.
Weight of stamp	lb.	1,250	1,375	1,550	1,670	2,000
Weight of shoe	lb.	235 = 18·8%	286 = 20·8%	286 = 18·5%	286 = 17·0%	290 = 14·5%
,, head	,,	315 = 25·2%	297 = 21·6%	376 = 24·3%	408 = 24·5%	872 = 43·6%
,, stem	,,	575 = 46 0%	630 = 45·8%	726 = 46·9%	726 = 43·5%	556 = 27·8%
,, tappet	,,	125 = 10·0%	162 = 11·8%	162 = 10·3%	250 = 15·0%	282 = 15·1%
Size of shoe ...	...	9 in. dia. × 12 in. long	9¼ in dia. × 14 in. long	9¼ in. dia. × 14 in. long	9¼ in. dia. × 14 in. long	9¼ in. dia. × 14 in. long
,, head ...	...	9 ,, × 21 ,,	9¼ ,, × 20 ,,	9¼ ,, × 24 ,,	9½ ,, × 24 ,,	9½ ,, × 50 ,,
,, stem ...	...	3½ ,, × 18 ft. long	3¾ ,, × 17 ft. long	4 ,, × 17 ft. long	4 ,, × 17 ft. long	4 ,, × 13 ft. long
,, tappet ...	...	9 ,, × 13 in. long	9⅝ ,, × 13 in. long	9⅝ ,, × 13 in. long	10⅛ ,, × 18 in. long	9 15/16 ,, × 20 in. long
Spacing of stamps	in.	10 in.	10¼ in.	10¼ in.	10¾ in.	10¾ in.
Diameter of camshaft	in.	6½ in.	5¾ in.	6½ in.	7 in.	7 in. hollow (1½ in. bore)
Spacing of camshaft bearings		6 ft. 0 in.	6 ft. 5 in.	6 ft. 5 in.	6 ft. 6 in.	6 ft. 5 in.
Thickness of mortar box base		9 in.	9 in.	9 in.	15 in.	11 in.

introduction of the heavy stamp. It is evidently that part of the stamp which must mainly serve to increase the weight of the latter for the reason that its weight does not decrease as does that of the shoe, and also because its weight acts immediately upon the rock to be crushed. With an increased size of stem the head also had to be increased so as to give the required strength, but as there is a limit in this direction imposed by the distance from centre to centre of stamp the additional weight required is generally obtained by increasing the length until, at the present time the length has in some cases reached 4 ft. The breakage of heads is now-a-days very small, owing to the hydraulic press superseding dynamite for removing the broken ends of stems.

Considerable attention has lately been paid to the design of the tappet, and from Figs. III., IV., V. and VI., may be seen the successive stages in the design to meet the conditions imposed by the increased weight of stamp.

Fig. III. shows the single-gib two-key type tappet, suitable for 1,250 lb. stamp.

Fig. IV. shows the single-gib three-key type tappet, suitable for 1,550 lb. stamp.

Fig. V. shows the double-gib four-key type tappet, suitable for 1,700 lb. stamp.

Fig. VI. shows the treble-gib three-key type tappet, suitable for stamps above 1,700 lb. stamp.

Fig. VII. illustrates the tappet designed by Mr. G. O. Smart, and it is suitable for stamps up to 1,300 lb. weight, having the advantage above that shown in Fig. III., that only one key need be driven instead of two keys in the latter type.

The cam has always been a source of anxiety to millmen even in the very simple form in which it is principally known to-day, quite apart from the many designs of built-up cams that have been put forward and tried at various times, and which in every case resulted in failure. The changes necessitated by the increased weight of stamp generally resulted in a cam of more substantial design without change to the involute described by the face of the wings. An improvement has, however, been effected by Mr. H. C. Behr, and was tried in the joint mill of the Simmer Deep, Ltd., and the Jupiter Gold Mining Co. Fig. IX. is an outline sketch of the new cam, and Fig. X. gives an illustration of its performance as compared with the ordinary cam in use in this mill and which is illustrated in Fig. VIII. It will be seen that with the improved cam, having the same overall diameter as the ordinary cam, the height of the drop is increased from 8⅝ in. to 9⅜ in. or by ¾ in. equivalent to 8%, and that the curve for the improved cam is, particularly in its rising part, very much more even that that for the ordinary cam.

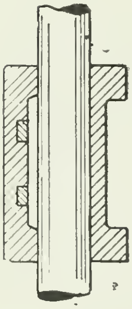


FIG. III.—2-key single gib tappet.

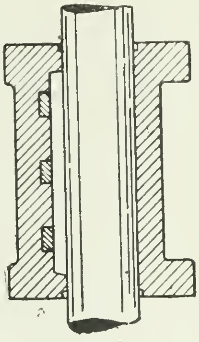


FIG. IV.—3-key single gib tappet.

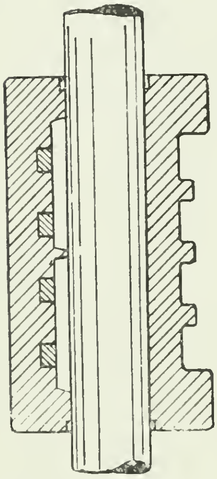


FIG. V.—4-key double gib tappet.

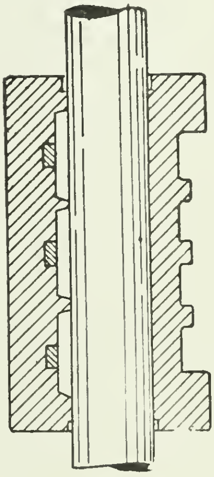


FIG. VI.—3-key treble gib tappet.

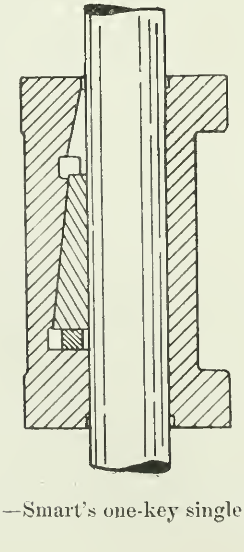


FIG. VII.—Smart's one-key single gib tappet.

The apparatus used for obtaining these curves was designed by Mr. Behr, and it has greatly helped in showing the behaviour of different cams under varying conditions. The curves show that, while the curve for the new cam must necessarily be steeper than that for the ordinary cam owing to the greater height attained in the same time, this curve is at its beginning much flatter than that for the ordinary cam which is due to the smaller generating circle employed. The shock due to striking the tappet is obviously less with the new cam, and this is clearly shown by the repeated jumps or jerks in the curve of the ordinary cam. In other words, the curve for the new cam more nearly approaches the ideal curve obtainable when lifting a stamp without a mass. It is also clear that when using the improved cam the wear on cam and tappet must be reduced. This, however, cannot as yet be proved from actual work, it being too early to give a final opinion.

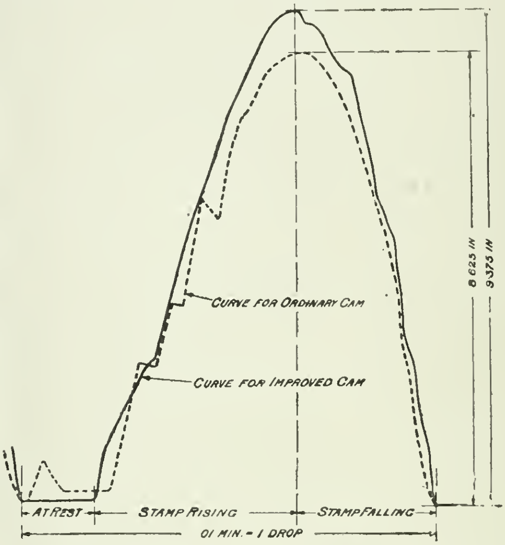


FIG. X.—Velocity curve for ordinary and Behr's improved cam.

The introduction of the heavy stamp was followed by considerable changes in the design of the camshaft. The writer does not wish to discuss in detail the various methods of fastening the cams to the camshaft, but it may be stated, in passing, that four methods are now in vogue, viz.:

(*a*) By the ordinary key.
(*b*) By the "Blanton key.
(*c*) By the "Blanton" rifling, and
(*d*) By the "M. P. Boss" method.

The last mentioned has lately been adopted by Mr H. C. Behr with considerable success, and with regard to the two "Blanton" methods opinions on the Rand are very much divided,

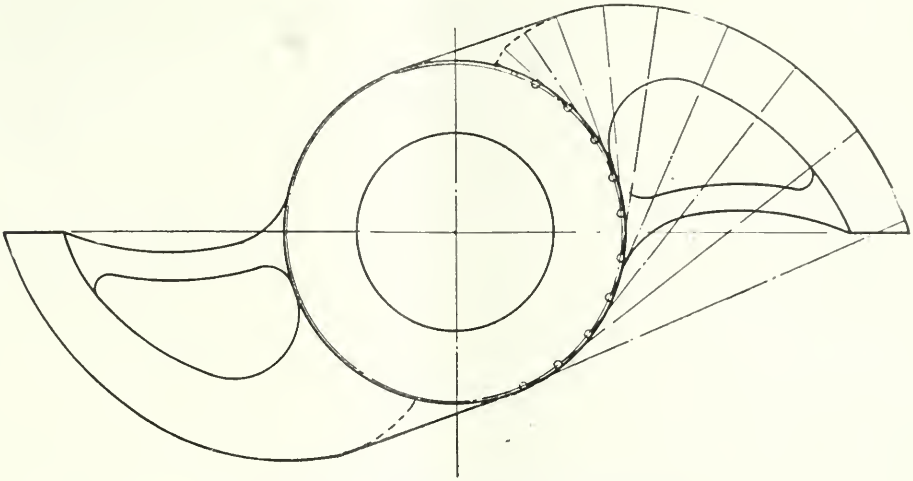


Fig. VIII.—Ordinary cam, Simmer Deep, Ltd.

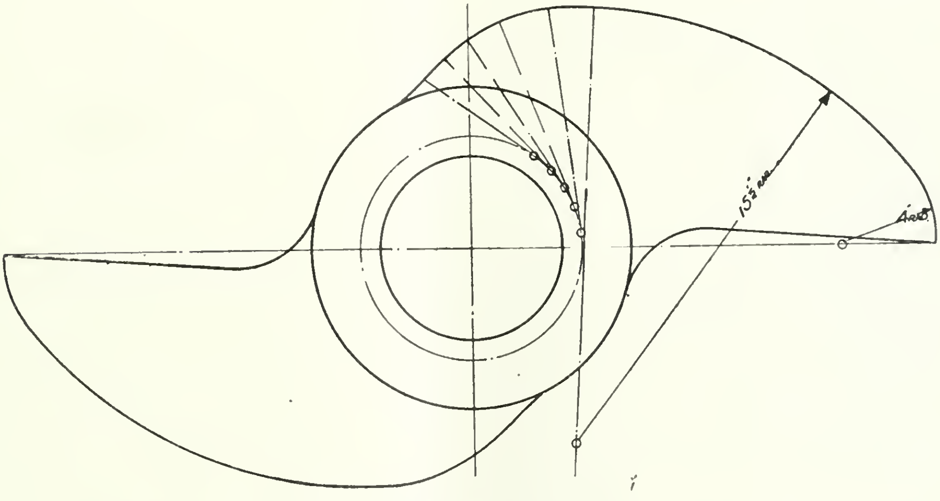


Fig. IX.—Behr's improved cam.

The ordinary key may be said to be a thing of the past, except for very light stamps, requiring as it does too much labour and time in removing cams. There are three types of camshafts in use on the Rand, the classification being in accordance with the number of cams carried by one shaft. They carry 5, 10, and 15 cams respectively. It is argued in favour of the five-stamp camshaft that it is carried in two bearings only, and that particular attention for keeping these in line is not required. Against this it is pointed out that the end of the shaft removed from the pulley vibrates considerably in its bearing. The extent of this can be seen from the curve shown in Fig. XII., which was obtained in the mill of the Wolhuter Gold Mining Co., and is an average sample of a great number taken by the writer.

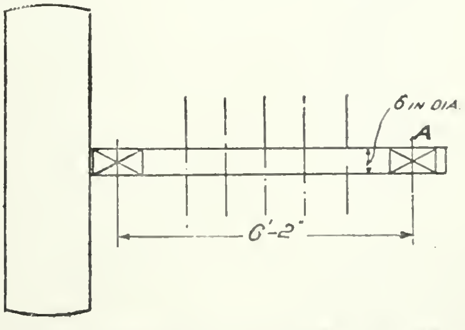


Fig. XI.—Five-stamp camshaft, Wolhuter Mines, Ltd.

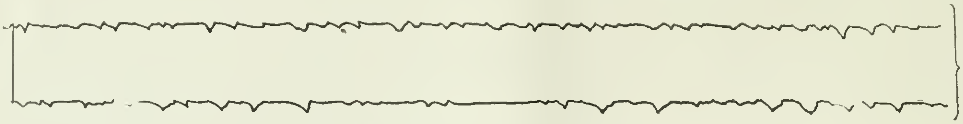


FIG. XII.—Vibration curves of five-stamp camshaft.

The weight of the stamps in this case is 1,380 lb., and the battery has cast-iron anvil blocks on concrete foundations. The camshaft is 6 in. in diameter.

The ten-stamp camshaft is carried in three bearings requiring more attention by way of keeping these bearings in line. The vibrations of this type of shaft in the bearings A and B are illustrated by the curves Fig. XIV., and these are average samples of those obtained in the joint mill of the Simmer Deep, Ltd., and Jupiter Gold Mining Co.

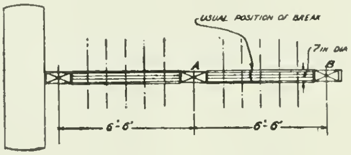


FIG. XIII.—Ten-stamp camshaft, Simmer Deep, Ltd.

The weight of the stamps is 1,670 lb. when new, and the diameter of the camshaft is 7 in. over the rifling.

The fifteen-stamp camshaft is, as far as the writer knows, only in use in the mill of the Witwatersrand Deep, Ltd., Fig. XVI. shows two average vibration curves for each of the bearings A, B and C.

The weight of the stamps is 1,380 lb., and the camshaft is 6½ in. in diameter. In this mill the mortar-boxes are carried on timber mortar-blocks and the kingposts on timber sills.

With regard to breakage of camshafts in batteries using heavy stamps, some further information would be acceptable. How far the increase in the weight of the stamp accounts for the breakage of camshafts the writer is not prepared to say, but it is a fact that microscopic examination of broken camshafts in many cases reveals faulty material such as in the cases illustrated by Mr. A. McArthur Johnston, President of this Society, in his recent Inaugural Address.*

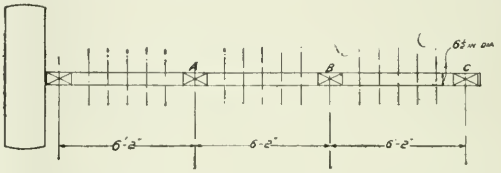


FIG. XV.—Fifteen-stamp camshaft, Witwatersrand Deep, Ltd.

It is a well-known fact that ten-stamp camshafts mostly break at the point indicated in Fig. XIII., and this was taken advantage of in the mill of the Luipaards Vlei Estate and Gold

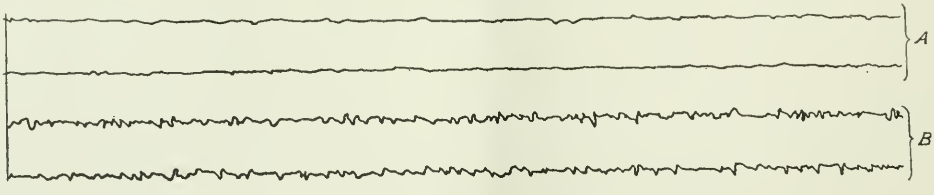


FIG. XIV.—Vibration curves of ten-stamp camshaft.

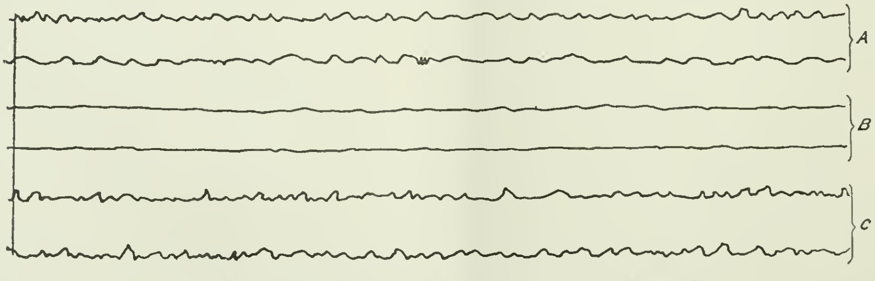


FIG. XVI.—Vibration curves of fifteen-stamp camshaft.

---

* See this *Journal*, vol. x., p. 5, *et seq.*

Mining Co. by making provision for a number of five-stamp camshafts so that all those ten-stamp shafts breaking as indicated can be used again in five stamp batteries by cutting off the waste end. A special type of ten-stamp camshaft, having four bearings, is shown in Fig. XVII.

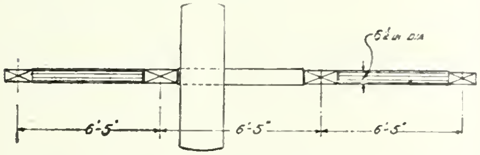


Fig. XVII.–Ten-stamp camshaft, Knights Deep, Ltd.

This is a design not to be recommended, and it has apparently been adopted to take better care of the torsional moment which, however, is a negligeable quantity.

It cannot strictly be said that the design of the camshaft pulley has been altered in order to meet the demand made by the heavy stamp, but the design due to Mr. Rodwell, of the Luipaards Vlei Estate and Gold Mining Co., is worthy of consideration, and is shown in Fig. XVIII.

The special feature of this pulley is the placing of the boards tangentially to the hub, each section being obtained by cutting a board of the required length diagonally, thus having no waste. The hub of this pulley is that designed by Mr. M. P. Boss of San Francisco.

The design of the kingpost has also been considerably modified but only partly as the result of the greater weight of the stamp. It is not always possible to procure timber sufficiently seasoned, and as a result of the shrinkage following in the wake of using partly seasoned timber the bearings carrying the camshaft are soon out of line and require a considerable amount of attention. Figs. XIX. to XXII. show the development in the design of the kingpost that has recently taken place.

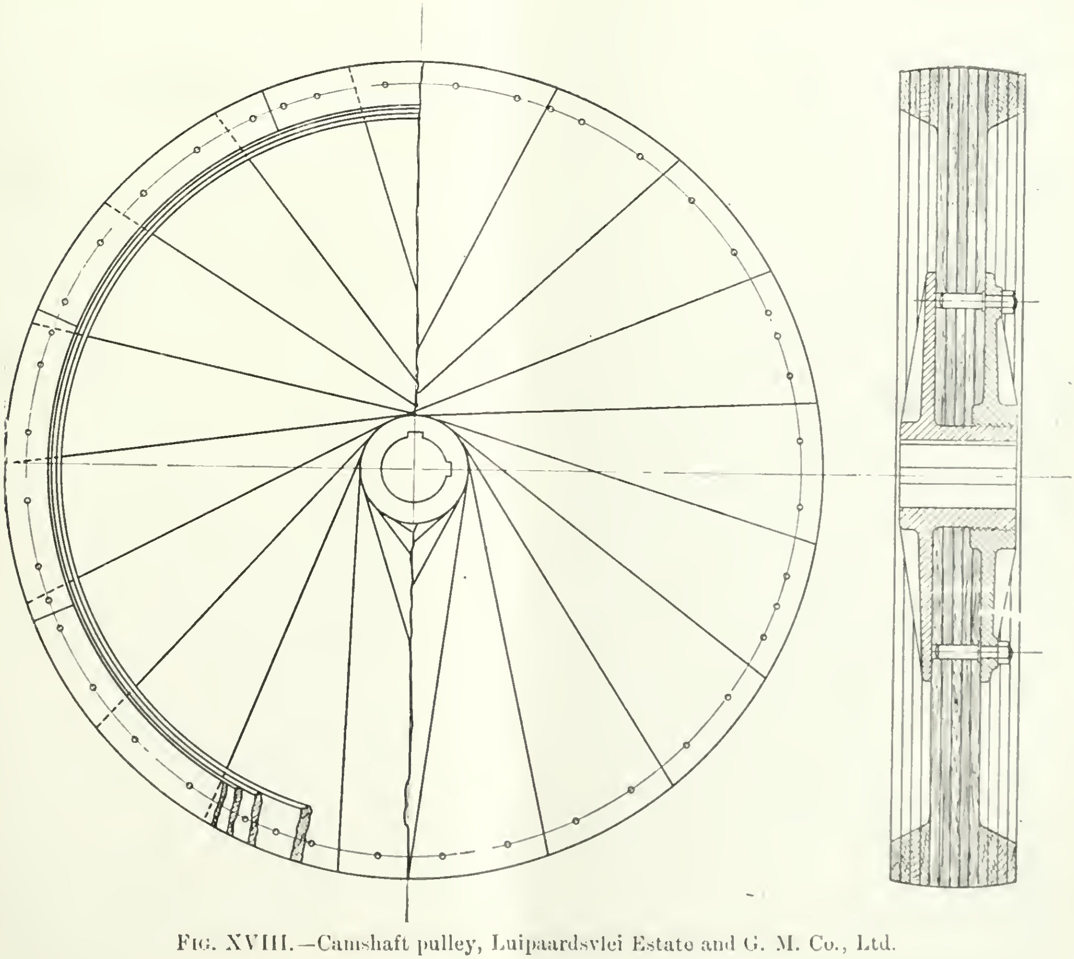
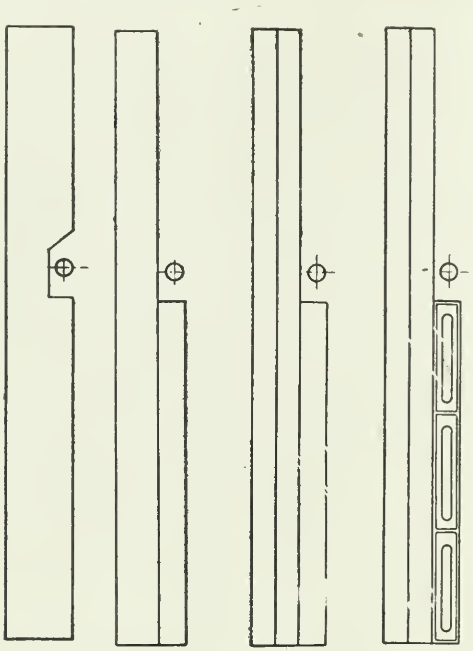


Fig. XVIII.—Camshaft pulley, Luipaardsvlei Estate and G. M. Co., Ltd.



Figs. XIX. to XXII.—Various types of kingposts.

Fig. XIX. shows the early design in which a single piece of timber was used and a recess was cut to take the camshaft box. This involves a considerable waste of timber as it is evident from Fig. XX., which shows a type of kingpost in which the piece of wood carrying the camshaft box is bolted on. The latter design will dispense with a lot of trouble, but it is still subject to the warping of the post, and in order to prevent this the design shown in Fig. XXI., and due to Mr. J. E. Thomas, of the Simmer Deep, Ltd., was adopted. Here the post is cut in two, and one half reversed so that should the timber tend to warping the two halves will do so in opposite directions and thus neutralise the action. However, all types of kingposts made entirely of wood possess the objectionable feature of shrinking, and as long as a number of bearings belonging to the same shaft are carried on timber it will remain difficult to keep them permanently in line. To overcome this difficulty the writer has advocated the design shown in Fig. XXII., in which the camshaft boxes are carried on cast-iron columns bolted to the timber posts, and it is anticipated that the result will be satisfactory. In the new mill of the City Deep, Ltd., the kingposts will be made of re-inforced concrete, and it will be interesting to see how these will behave under the severe conditions present in a battery using stamps of 2,000 lb. weight.

The author dealt with the foundation of the battery at some length, and he gave in his Fig. I. a sketch of the foundation used in the mill of the Simmer Deep and Jupiter joint plant. The block serving for 10 stamps contains about 50 cub. yd. of concrete, and for the benefit of those interested it may be stated that the bulk of it was made up as follows:—

22½ cub. ft. of 2 in. stone
7 ,, ½ in. stone
13 ,, washed sand
1⅛ barrel of cement
} make 1 cub. yd. of concrete.

While for the last 2 ft. from the top 1¾ barrel of cement was used for the same quantities of stone and sand.

In his Fig. III. the author shows the type of foundation used in a number of mills having stamps of 1,550 lb. weight, the main feature being the use of the cast-iron anvil. This type has now been discarded in favour of that using concrete only, resulting in a considerable saving. The cost of the foundation for 10 stamps in the Simmer Deep mill (the author's Fig. I.) was £175, while the cost of the foundation with anvil block and timber cushion (the author's Fig. III.) was £450, a difference in favour of the former of £275 for 10 stamps, quite a considerable item.

In the writer's Fig. I. is shown the old type of foundation using the timber mortar block, but it is impossible to give reliable data regarding the cost, so that a comparison from this point of view can not be made. There can, however, be no doubt that the latest type of concrete foundation is the more durable of the two seeing the short life of the old type as evidenced by the statements made at the commencement of these remarks.

The increase in the weight of the stamp has been followed by considerable changes in the design of the mortar box. It would lead too far to go into all the details, and a reference to the table will show the alterations made in the thickness of the base. In the days when the anvil block was considered a necessity it seemed to the writer that there was too much cast-iron involved in the design, and in order to reduce this it was proposed to combine the anvil block and the mortar box by making the body of the box separate from the base as shown in Fig. XXIII.

It was thought by making the body of mild steel or cast steel it could be made very light, thus minimising the difficulty of holding it to the base. Of course, the base could then be made of any thickness, and the difficulties always present when casting a body very heavy at one end and very light at the other would be overcome. There should be no difficulty in making a tight joint between body and base. However, since it was found that the anvil block was not needed and

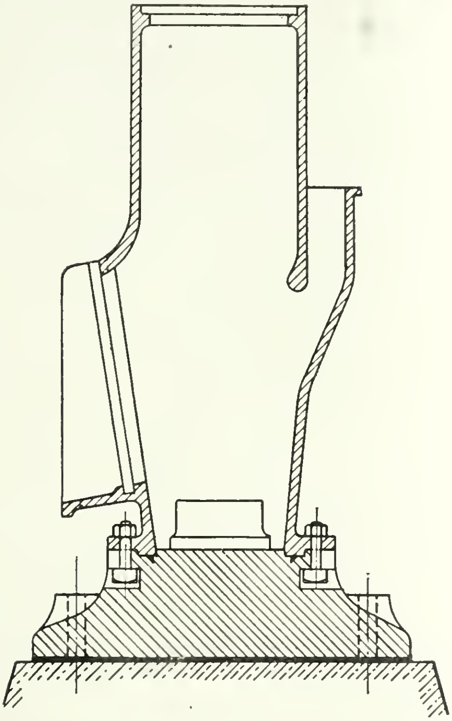


FIG. XXIII.—Schmitt's composite mortar box.

that a moderate thickness of base in the mortar box was sufficient to meet the demands of the heavy stamp, there may be doubts about the necessity of making a composite box such as the one described. Heavy stamps with good foundations have now been in use for several years, and the fact stands out that the breakage of mortar boxes has decreased if it has changed at all.

In the old type of battery the kingpost was rigidly connected to the mill bin through the knee beam. This was one of the weakest points in the design as the knee beam tends to lift the kingpost whenever, under the increased load in the bin, the knee beam is deflected. In the Simmer Deep mill this difficulty has been overcome by making a flexible connection between the member holding the kingpost laterally and the bin framing while at the same time rigidly holding the kingpost against any longitudinal motion. This also prevents any vibration there may be in the kingpost from being transmitted to the bin framing. Arrangements were also made (see the author's Fig. I.) to carry the camshaft platform independent of the kingpost, and this is a considerable improvement to which reference has already been made when describing the alterations to the mill of the Simmer and Jack Proprietary Mines. It was first suggested by Mr. G. O. Smart, Reduction Works Manager of the mine just named.

Ever since the increase in stamp duty began it was felt that arrangements must be made for increased storage capacity in the mill bin. With a stamp duty of 5 tons per day a storage capacity of 12 tons per stamp was sufficient to allow of milling over Sunday, which is permitted by the laws of the country which, however, do not permit of hauling ore either in the shafts or on the surface. The Simmer Deep mill bin was designed to give a storage of 26 tons per stamp, and this provided amply for a proposed duty of 8·2 tons per stamp per day. Conditions, however, are rapidly changing and stamp duties from 14 to 20 tons per day will in the future be quite common. With the usual allowance of three day's storage this means a bin capacity up to 60 tons per stamp, and it is a problem of considerable magnitude if the cost of the plant is to be kept within reasonable limits. Mr. A. M. Robeson* anticipated this to some extent by proposing the storing of the ore outside the mill building and feeding the inside bin by a system of belt conveyors. This, of course, means that additional machinery is to be maintained and that extra power will be consumed, leaving alone the main question as to whether the law permits of running such conveyors on Sunday. The obvious solution is to stop the mills on Sunday but failing this, the writer would prefer to make the mill bin of the required size rather than introduce additional machinery.

It cannot be denied, in view of the foregoing remarks, that the coming of the heavy gravitation stamp is responsible for considerable changes in the mill design, and that a number of new problems have arisen, some of which still await a satisfactory solution. Meanwhile efforts have been made to find a solution for the main problem, that is, crushing more ore for the same or less capital outlay and at reduced running costs, without increasing the weight of the gravity stamp. It is impossible to mention all the experiments made in the course of this discussion, but a few which have come under the writer's observation shall be briefly referred to. They are :—

(*a*) The Morrison high speed stamp.†

(*b*) The Holman pneumatic stamp.

(*c*) Mr. H. C. Behr's accelerated stamp.

(*d*) Coarse crushing with re-grinding.

The Morrison high speed stamp has already been mentioned by Messrs. H. S. Denny and R. E. Commans in the discussion on Mr. Caldecott's paper, but as it has been previously described, and was a failure as far as the Rand is concerned, further remarks are unnecessary.

---

* See *Journal of the Mechanical Engineers Association of the Witwatersrand*, vol. ii., p. 45.

† See *Journal of the Transvaal Institute of Mechanical Engineers*, vol. vi., p. 392.

The Holman pneumatic stamp is at present undergoing extensive trials at the New Kleinfontein Co's mill and, as far as the writer is aware, the results obtained are quite satisfactory.

The result of Mr. Behr's experiments with stamps accelerated with the aid of compressed air have not yet been published, but the writer knows that Mr. Behr succeeded in running the ordinary 1,670 lb. stamps at the Simmer Deep at 140 drops per minute. Figs. XXIV. and XXV. show velocity diagrams of the Simmer Deep stamps running at 100 and 140 drops per minute.

With regard to the coarse crushing in the stamp mill and re-grinding in secondary machines there is, of course, no reason why coarse crushing and heavy stamps should not go together although if there should be any serious objection to the heavy gravitation stamp, coarse crushing will be the chief means for obtaining the desired result without having to raise the weight of the stamp. Experiments recently carried out by the Mines Trials Committee at the Simmer East mill with stamps weighing 1,400 lb. and crushing through a 9-mesh screen (·27 in. aperture) gave a duty of 15·1 tons per stamp per day, and the product from 17 stamps was easily reduced to the required fineness by one 22 ft. × 5 ft. 6 in. tube mill. Further, experiments made at the Simmer Deep, at the suggestion of the author, with stamps of 1,640 lb. and crushing through a 30-mesh screen (wire ·0482 in., aperture ·1348 in.) gave a stamp duty of 16·7 tons when running 75 stamps in conjunction with four 22 ft. × 5 ft. 6 in. tube mill. The writer has analysed the result of the last mentioned experiment,* in which a crushing and grinding plant of

(*a*) 75 stamps and 4 tube mills, and
(*b*) 75 stamps and 1 tube mill

were compared. The result showed that the saving due to running the plant with 4 tube mills was as follows :—

	Per Cent.
Saving in capital expenditure	30·0
Saving in ordinary working costs† ...	17·5
Saving in ordinary working costs plus capital charges‡	23·7
Saving in power costs alone	15 5

Considering the excellent results obtained by a judicious combination of stamps and tube mills it appears to the writer that the time has arrived for re-considering the whole question. Of course, any improvement made in the stamp mill will also benefit a combination of stamp and tube mill but, as there is a limit to the weight of the gravitation stamp, the combination referred to will enable engineers and metallurgists to regard this limit with more equanimity than would otherwise have been the case. The introduction of the heavy stamp has done good beyond estimating, by calling attention to the weak points

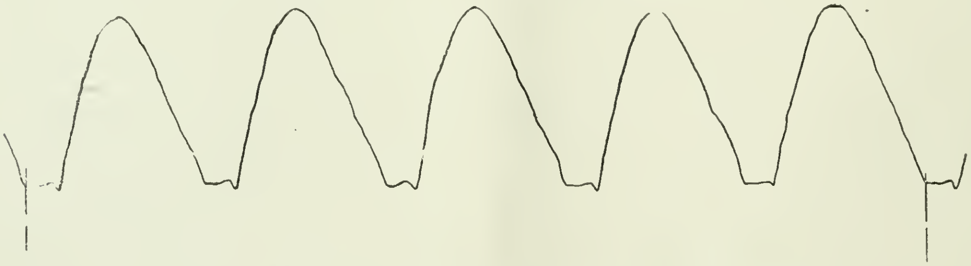


Fig. XXIV.—Velocity curves of 1,670 lb. stamp, at 100 drops per minute.

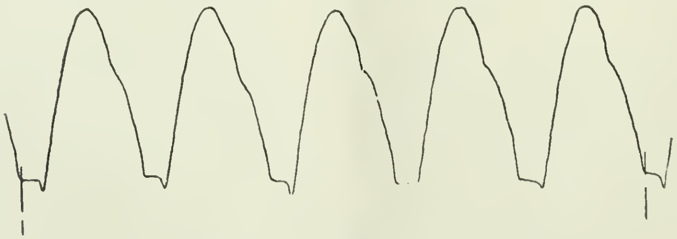


Fig. XXV.—Velocity curves of 1,670 lb. stamp, at 140 drops per minute.

---

* See *Journal of the South African Association of Engineers*, vol. xv., p. 113.

† These include labour (white and coloured), power, mercury, shoes, dies, screens, pebbles, liners, lighting, water, maintenance and general charges.

‡ Capital charges on the basis of 7% interest, plus 3% for redemption of capital.

in the stamp mill design, and for this alone, if for nothing else, Mr. Caldecott, as its chief exponent on the Rand, deserves the greatest credit.

The Chairman: Gentlemen, our hearty thanks are due both to Mr. Pearce and Mr. Schmitt for their able contributions to this already classic paper.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

Part I.—Crucible Experiments.

(*Read at November Meeting, 1909.*)

By Prof. G. H. Stanley (Member of Council).

DISCUSSION.

Mr. D. W. Greig (*Associate*): The contribution to the question of titaniferous iron ore smelting by Prof. Stanley, in which his crucible experiments are detailed, is in my humble opinion the best that has been written on this subject, inasmuch as he is the first to attempt to prove experimentally that titanium oxide can act, and in fact usually does act as a base. At a works with which I was connected this oxide was always considered as an acid and treated as such, hence, no doubt, as Prof. Stanley has pointed out, accentuating the very evil we wished to overcome.

With the authors views on the electric smelting of titaniferous ores I am in absolute agreement. Our object would not be to make special steels, but for the ordinary article the cost of the electric process would be prohibitive. Those who advocate the electric smelting of ores of the Onderstepoort class are very fond of quoting from the Canadian Government Report, published two or three years ago; but anyone who takes the trouble to read the report carefully cannot but come to the conclusion that it is not very sanguine. It must be borne in mind too that Canadian conditions are vastly superior to those appertaining here, especially as regards price of current.

To come now to the author's experiments. He performed a number of reductions of the Onderstepoort ore, and protests that he obtained fusible slags. His slags contain a tremendous amount of iron, but this was to be expected, seeing the reductions were carried out in crucibles. A considerable amount of iron in a slag, however, generally lowers its melting point and always increases its fluidity. It will be pleasing therefore if, in the experiments the author is now carrying out in the more perfectly reducing atmosphere of a small blast furnace, he is able to obtain fusible titanium—carrying slags which contain little iron.

I think Prof. Stanley, at the beginning of his paper, passes over the question of fuel rather hurriedly. The matter of fuel is perhaps more important than the ore itself. He quotes one sample of coke which came under his notice containing 9·7% ash, but we can hardly settle questions of this kind on individual samples. A number of coal analyses were given by Mr. A. J. Andrews in his paper on "South African Coals," read before the April meeting.* The lowest ash content of these samples was the Natal coal, which showed 11·5% while the Middelburg variety showed 13·91%, and our local stuff he put down at 18·3%. The coke produced from even the best of these coals would contain 15 to 16 or even a higher percentage of ash, and one of the rules of thumb in vogue around Middlesborough is that a coke containing over 8% ash is unsatisfactory in blast furnaces.

There is a more important point still in connection with the coke, and that is the quantity of sulphur it contains. Sulphur is the bane of an iron and steel maker's existence; if reduced it causes the iron from the blast furnace to run "white," and to prevent as far as possible its being reduced, and entering the iron the furnace must be run "hot" and the slag "limey." Lime would be necessary, as I am afraid Prof. Stanley's weak base, Ti_2O_3, would have no affinity for sulphur.

Near the end of the paper the author comes back to the fuel question, and suggests that in places where no coking coal exists, coal itself might be used. It is a well known fact however that coal driven furnaces never run quite so "hot" as those using coke, and ores containing titanium require all the heat possible. Coal also tends to encourage "hanging" in the blast furnace; an evil which the Onderstepoort ore would introduce without encouragement. The dolomite Prof. Stanley quotes is poor stuff, the 4% of silica it contains will demand about 8% of the lime to form a flux, reducing considerably the utility of the dolomite and adding to the quantity of slag.

The author endeavours to make out a case for "white" iron. Iron makers at home find very little sale for this class of pig, and I would like to ask the author for what kind of open hearth steel furnaces it is well adapted, as a "white" iron low in silicon is generally high in sulphur? Prof. Stanley's "white" irons are certainly low in silicon, the sulphur is not given.

Whatever may be our individual opinions regarding some of Prof. Stanley's conclusions, there can be no doubt that one phase of this interesting subject has been filled in by a "man on the spot," and, I for one, anxiously await the publication of the larger experiments, upon which I understand he is now engaged.

---

* See this *Journal*, Vol. ix., April, 1909, p. 330.

The Chairman: The Society is to be congratulated upon a keen associate. I wish some of the members would emulate him.

TREATMENT OF ORE SLIME.

(*Read at November Meeting, 1909.*)

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

DISCUSSION.

Mr. C. O. Schmitt (*Member*): I would like to congratulate Mr. Crosse on his excellent paper which, I am sure, will be appreciated by all having the advancement of the metallurgy of gold at heart. I do not wish to deal in detail with the plant described, but I should like to point out that the value of the paper would be greatly enhanced by an estimate of the capital outlay required for a plant dealing with an assumed tonnage, as this would enable us to see how the new plant compares with the decantation plant. A detailed estimate of the probable cost of treatment would also be useful, and would certainly assist us in arriving at a final conclusion.

ENDLESS ROPE HAULAGE.

(*Read at November Meeting, 1909.*)

By H. G. Kay (Associate).

DISCUSSION.

Mr. Tom Johnson (*Member*): The author is to be congratulated on his choice of subject, seeing that so many of our members are, or should be, interested in endless rope haulage. He confines himself to surface work, but there is a wide field for exploitation underground, both in haulage uphill and in self-acting inclines, but not in drives. I hope our mine managers and cyanide managers will direct their attention to the amount of rock hauled per day with each rope at the Premier Diamond Mine and compare it with what is done on our dumps on the Rand. I fancy there is plenty of room for improvement.

The author seems to be of the opinion that a divergence at a curve of 15° to 30° is as great as can be safely negotiated. I beg to differ. By using flat sheets or channels instead of ordinary rails at the curves, a deflection of 90° or more can be accomplished with little difficulty. I also differ with him as to the probability of cars "upending" on grades steeper than 1 in 6, we have steeper dumps than that on which the rope travels at a speed of four miles per hour. His explanation of how the hooking on place should be laid out is quite correct, although on the Rand little attention is given to the point. If hooking on places are laid out as described, and if the wheels of the cars have a wide coupling or base the liability to "upend" is greatly reduced. One advantage of the side rope on the frame is that it reduces the liability of "upending," the rope does not have the same leverage on the car. As to keeping one rail of the track higher than the other to get the rope over the centre line of the track, this is not necessary, the jockeys should be so fixed to the car, that the rope lies over the centre, when the car is standing on a horizontal base.

I heartily endorse his remarks about disengaging frames, tipping and righting arrangements, and commend them to our dump men and cyanide managers; the tipping and righting arrangements should also appeal to mine captains, there is scope for such things on our shaft boxes in most mines. As to the side rope and jockey method of haulage, I am glad he considers it economical as I happen to be the inventor and patentee of this method. I have not described it to the Society as it would have looked too much like advertisement, considering that I have drawn royalty for twenty installations. I put this system in at the Rose Deep 4½ years ago.

I think there are a few of our members who can speak as to its merits or otherwise. I see that at the Premier the jockey is placed at the centre of the length of the frame, between the wheels with the idea that the trucks would get round curves easier. It was not necessary, the jockeys would have suited if placed just ahead of the front wheels as is usual here on the Rand. The use of channels would have solved the difficulty and enabled the cars to get around sharper curves. In any case, I think it would have been better if they had used a proper birdcage wheel at the curves instead of the arrangement they did. A birdcage wheel is a series of small rollers held between two spiders or checks, which turn on a central shaft, the small rollers in contact with the rope revolving in their own bearings. When a jockey comes against the first roller the whole cage revolves on the central shaft. This arrangement reduces the shock and jar to the rope when jockeys are passing which is so noticeable in the ordinary arrangement.

With regard to endless rope haulage underground, I would refer members to some remarks in my paper, entitled "Notes on Rand Mining,"* the discussion and the answer to same.

---

* See this *Journal*, vol. viii., March, 1908, p. 255, *et seq.*

Self-acting inclines, or what we term brake-roads, offer a good field for endless rope work, chains being used to attach the cars to the rope. The braking wheel could be placed at the bottom of the incline for stope work, and the return wheel placed a little higher up the stope than the highest side track. Side tracks would be 50 ft. or 60 ft. apart, and on both sides of the incline tracks, the inclination of the incline track would be decreased at the junction with the side tracks, bridge rails, tipping platform, and slotted plates or other means used for turning the car to go in the required direction. In working, a system of signals for indicating to the brakesman and others at the point cars are being attached must be devised, and these may be either coloured lights or bells. Supposing there were, say, four sets of side tracks, and at No. 1 it was required to send a loaded car down, the proper signal would be given and returned, the car would be pushed on to the slotted plates the chain attached taut to the car and rope, the car turned into the right direction for going down the incline, the stop block would be opened and the signal to lower given. Of course an empty car would have already been attached to the rope at the bottom of the incline. On lowering the full car this empty car would be drawn up to No. 1 track and there taken off, and the rope would be ready for letting down the next car from whichever side track required.

As to the steepness of inclines on which ordinary cars can be handled, I was roughly drawn over the coals by one of the contributors to the discussion on my paper, "Notes on Mining," for stating that cars could be handled on an incline of 45°. As I stated in my reply, I had myself done this, and since then I find that others have also done it or seen it done. G. L. Kerr, in his book on mining, mentions working on inclines, from 10° to 45°. He mentions one case at Morton Collieries, Manchester, of an incline 2,900 ft. long, inclination 26° to 39°, with several landings to hook on at, swing rails being used at the landings. Then A. R. Sawyer says* it was formerly the general custom to jig down inclines of as much as 50° with ordinary pit wagons. In the district Mr. Sawyer speaks of, I helped to put in an endless haulage in an incline with seven pairs of levels on it, at which they hooked on, the rope travelling continuously. I forget the inclination, but remember one part passing through a fault, where steps were required. I know it was steep enough to require the cars to be handled in pairs, coupled together short so that one would stop the other from "up-ending." The incline was 1,100 yards in length. Of course all the loads were hooked-on on the one side, the cars passing from one side of the incline to the other, through a passage cut in the roof of the incline. One point deserving notice, was that the endless chain was first installed, but the engine that eventually was used for the endless rope could not move the endless chain when empty.

The Chairman: Mr. Johnson, as usual, has given us something to think about, something we may not all agree with, but that cannot be expected in a Society like ours.

A ROTARY EXTRACTOR FOR PRECIOUS METALS FROM SOLUTIONS.

(*Read at December Meeting, 1909.*)

By W. D'Arcy Lloyd and Ernest T. Rand
(Members).

DISCUSSION.

Mr. A. King (*Associate*): The authors of this interesting paper certainly deserve every praise for the rotary extractor which they have so cleverly devised. I have not seen this extractor at work so I am scarcely in a position to be able to fairly criticise the mechanical arrangements; but according to the sketch the baffle plates in the settling tanks underneath the rotor seem to extend so far down that the force of solution passing round the baffle might tend to prevent the settlement of any gold slime at that point, and if this gold slime be carried round past the baffle then it will not readily settle down through the solution which is now passing upwards to the stand pipe. One of the advantages claimed for this extractor is that there is a saving of zinc by using clippings instead of ordinary filiform zinc. This does not seem quite clear because the actual weight of zinc used in packing the rotor will be much greater if clippings be used. If I have taken the correct meaning of that part of the paper where the authors deal with this, then if the rotor be packed with clippings this zinc ought to last eleven times longer than if filiform zinc be used. The extraction results which are given show that the precipitation is not so good with this rotary extractor as with the ordinary stationary extractor. For the purpose of comparison I will quote the average results obtained for last month at Knights Deep where the extractors are of the usual kind (see table, next page):—

Incidentally I might mention that a large number of the assays (which are made daily) of the solution leaving our extractors only showed traces of gold, but in making up the above average values these traces were taken at 0·010 dwt. per ton.

---

* Miscellaneous Accidents in Mines, p. 132.

	Stationary Extractor.		Rotary Extractor.	
	Tons of Solution per cub. ft. of zinc per 24 hrs.	Value of Solution leaving, dwt. per ton.	Tons of Solution per cub. ft. of zinc per 24 hrs.	Value of Solution leaving, dwt. per ton.
Strong	1·40	0·010	1·01	0·053
Weak	1·70	0·012	1·84	0·029
Slime	2·00	0·013	1·53	0·031

By comparing these results with those obtained with the rotary extractor it will be seen that although more solution was precipitated in 24 hours per cubic foot of zinc, with the exception of the weak solution which is slightly lower, yet the amount of gold in the solutions leaving the extractors is much lower than any of those given by the authors; and every cyanide worker knows the bad effect produced on sand and slime residues and on second wash solutions if the gold contents of solutions leaving the boxes be high. Before one could arrive at a definite conclusion as to whether this disadvantage, which the rotary extractor seems to have, would be counterbalanced by the advantages which it undoubtedly has, a few more details would require to be known. Perhaps in their reply the authors may be able to increase the value of this already very interesting paper by giving us some idea of what the initial and working costs of this extractor might be, and of its dimensions and capacity, so that a closer comparison might be made between it and the ordinary stationary extractor.

The Chairman: I think we may congratulate ourselves upon having had a very interesting first meeting of the year.

The meeting then closed.

Obituary.

HILARY BAUERMAN.

The following obituary notice by one of our Corresponding Members of Council is reprinted from *The Mining Journal* (London), Dec. 18, 1909.

Mining and metallurgical circles both at home and abroad, and many branches of science, have suffered a heavy loss through the death of Prof. Bauerman, who passed away on Sunday, the 5th, from heart failure after a prolonged illness. Associated so intimately with mining and metallurgy, both from the purely scientific aspect and from the practical standpoint, and connected as he was, almost as intimately, with the study of mineralogy and geology, it is difficult to say which will feel the loss most.

His astonishing versatility and the depth of his knowledge; his ability to impart it and his willingness to give the benefit of his experience to those who sought it in the right way, are well known; but only those who had the opportunity of meeting him frequently, and under conditions other than those where business alone was under consideration, could fully appreciate the extent of his quiet influence. His marvellous memory, and his ability to use it at the right moment, and to express his opinion in the most opposite though sometimes unexpectedly emphatic way, rendered his conversation invariably interesting and his advice valuable.

His extensive experience at home and abroad, extending over more than thirty years and embracing a great variety of survey, geological, and metallurgical work through most of the important or growing mining districts of the world, excepting Australia and New Zealand, gave him opportunities for studying countries and people, and their habits and languages, of which he took the fullest advantage.

Bauerman's professional career dates from 1851, when, at 18 years of age, he became a student of "The Government School of Mines and of Science applied to the Arts," where he studied under Playfair, Ramsay, Forbes, Warington Smyth, Hunt, and Percy. The conditions for working were very different from those now existing. A day's work often commenced at 7 in the morning and lasted until midnight, either in the laboratories, which were little more than cellars, or at the best makeshifts, or at the house of Percy, where a few privileged students were allowed to study his minerals and assist him in the preparation of the notes from which "Percy's metallurgy" was ultimately evolved. It is interesting to note that the manuscript of this work passed, on the death of Percy, into the possession of Bauerman, and was presented by him to the Institution of Civil Engineers.

From the Government School, whose title was ultimately changed to "The Royal School of Mines," Bauerman, who may thus be regarded as our first Royal School of Mines-man with a record of fifty-eight years, proceeded to the Freiberg Mining Academy, whence, after three years' further study, he returned in 1858 to accept a position as one of the assistant geologists to the Geological Survey of Great Britain. He was soon transferred to the North American Boundary Commission, where during six years, he did much important official work, including the delimitation of the Hudson Bay territory. For the next twenty-five years he was engaged in Government and professional work—geological, mining, and metallurgical—in Sweden and Norway, in various parts of the United States, and Mexico, South America, Egypt and Arabia, and in many parts of the European continent. His experience in England, both in mining and metallurgy, his actual knowledge of the principal metallurgical works and processes, and his personal friendship with those in control, were unequalled. He had the entrée to many of the most exclusive works, and took advantage of it to show to parties of his Woolwich students and others, processes and plant which few were privileged to see.

As a teacher, and particularly when chatting or demonstrating, apart from his official "lectures," he was particularly successful both with the cadets at Woolwich, who admired and revered him, not withstanding their occasional practical jokes, from which they usually emerged second best; with the practical miner and metallurgist, including his workmen students at the Firth College, Sheffield, and elsewhere, and with his personal friends and acquaintances.

A lengthy list might be given of his published works; of his contributions to the transactions of our learned and technical societies; of the medals and other awards which he received from the Institution of Civil Engineers, the Institution of Mining and Metallurgy, the Iron and Steel Institute, etc.; of the societies of which he was honorary member or member of council or committees; of the innumerable international exhibitions, on whose committees and juries he sat, often as president or chairman; and of the congresses and Parliamentary committees with which he was connected: but the writer is content to say that no one could have compressed even in so lengthy and active a professional career more solid or lasting work. The results are perpetuated, either in his published books or papers, often contributed without signature to the technical press, or in the memories of his friends.

As an examiner, he was peculiarly unorthodox and successful. He had a power of ascertaining what a student really *knew*, or whether he had been "crammed," which was almost uncanny, and his power frequently caused some uneasiness to such friends and acquaintances as ventured to join issue with him in the discussions which he delighted to initiate or continue. His ability in this respect was so well recognised that he was for a great number of years examiner to the Civil Service Commissioners for the appointment of mine inspectors; to the Science and Art Department and Board of Education in mining and metallury, and, for a time, in mineralogy; and an examiner of mining students to the Royal School of Mines and the Camborne School of Mines.

Of Bauerman's personal qualities and habits which might be written did space permit. Trained under Percy, whose mantle and rugged personality and manner seemed to have been bequeathed to him, Bauerman's didactic manner, coupled with the fact that, even to the very end, he was practically always right on matters of fact or deduction and seldom wrong on questions of policy, gave the greatest weight to his opinion and rendered him a trusted colleague and, to the younger generation, a most valued mentor.

His deafness, which existed before he reached middle age, although scarcely noticeable in the earlier years, and which increased as time progressed, caused him to be often misunderstood, and limited his progress in many directions; but, although many public honours which should have been his passed him by, few men occupied so many official positions or were members of so many important councils or committees. Few of our congresses passed without his presence—he was a member of the Metallurgical Committee of the Seventh International Congress of Applied Chemistry, held in London this year, and president of a section of the Sixth Congress held in Rome three years ago—and most of our International Exhibitions had the benefit of his experience as counsellor or juryman.

From the Great Exhibition of 1851 to the Franco-British in 1908, on whose metallurgical section he wrote two excellent monographs, which were published by the Iron and Steel Institute, he was associated with most of the important ones at home and abroad, and his contribution to the published jury reports of that of 1851 was probably his earliest serious literary effort.

Although Prof. Bauerman retired from official service two years ago when he resigned his position as Professor of Metallurgy at the Ordnance College at Woolwich, he continued his professional work until the time of his illness.

To so active a man, the enforced idleness proved most irksome, but he bore it with the greatest fortitude, and constantly expressed his pleasure at seeing and hearing from his friends, and his hope of some day joining their circle. Even his increasing weakness and the sleeplessness which was his greatest trial, left his brain clear, and almost his last words referred to the future of those he was leaving.

On the morning of the day preceding his death he expressed a desire to make some notes, saying that if he could do some "real work" perhaps rest and sleep might come to him. From his dictation, a review of a book which had been sent to him was taken down by the devoted companion to whom he owed the comforts of his later years, and who had nursed him as no other could have done throughout his illness.

On the following day, the morning of Sunday, the 5th inst., he passed away peacefully in his sleep, seated in his favourite chair, which had been bequeathed to him by Percy, the old friend under whom he had assayed his earliest steps in metallurgy, whom he had assisted, with others of his fellow students, in the preparation of "Percy's Metallurgy," and whom he had succeeded as Prof. of Metallurgy at Woolwich.

The death of Bauerman will be deeply felt by many, but particularly by those of the younger generation to whom he formed a link between the past and the present, and through whom he looked into a future which he himself could not expect to share. His personal friends were many, his intimates few, but all—even his acquaintances—will miss the familiar personality whose very eccentricities were those of genius. Although many of his contemporaries are still with us, his death robs us of another veteran of the Royal School of Mines, whose dignity he did so much to maintain, and whose individuality was so admirably exemplified in him.

G. T. Holloway.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Chromite Ore from the Transvaal.—The ore, said to have been obtained near Jachtlust, consisted chiefly of chromite with some impurity in the form of pyroxene and felspar. Three lumps had only small quantities of impurity and specific gravity was 4·35, the fourth lump contained much impurity, nearly all pyroxene had a specific gravity of 4·10. The first sample on analysis yielded Cr_2O_3 47·0%, FeO 23·99%, valued at 65s. per ton c.i.f., and the second Cr_2O_3 38·4%, FeO 21·5%, valued at 50s. per ton c.i.f.—*Bulletin of the Imperial Institute*, vol. vii., No. 3, 1909, p. 277. (J. A. W.)

Recent Progress in Cyanidation of Gold.—"In the treatment of auriferous sand by potassium cyanide, consumption of this product constitutes the chief expense. Hence the numerous researches to reduce the amount of reagent employed to a minimum. Mr. Nugel made experiments with a view to determine the mode of dissolution of the gold in double cyanide of potassium and zinc. For a long time it had been noticed that the solutions of double cyanide of potassium and zinc only containing the amount of potassium indicated by composition of the double cyanide dissolved a certain percentage of gold. From these experiments he was able to conclude that in these solutions of double cyanide only half the cyanogen reacts, and that it suffices to add caustic soda to make the other half useful. The following formula represents the reaction which intervenes to liberate the potassium cyanide—$ZnK_2(CN)_4+2KOH=Zn(OH)_2+4KCN$. The alkalinised solutions fixing oxygen less readily, their employment was very limited until the day when this faculty of absorption was increased by bubbling air through these solutions. This bubbling is possible with double cyanides less decomposable by the carbonic acid in the air than potassium cyanide. According to the *Chemiker Zeitung*, Mr. Nugel applied this process at the works of Abbontiakoon Block I., Ltd. Primitively solutions containing 0·3% cyanogen were employed in form of potassium cyanide and 1·25% double cyanide. He reduced the total cyanogen to 0·15%, and by adding the indicated amount of caustic potash indicated by the previously mentioned reaction obtained an excellent result. With half less consumption of reagent the yield of gold is better than with the primitive processes. Applied in Mexico for silver, cost of treating the ore was greatly decreased."—*London Mining Journal*, Sept. 11, 1909, p. 356. (A. R.)

Assay of Dressed Tin Ores.—"There are certain points in connection with the assay of dressed tin ores which cannot be too strongly insisted upon. They are: (1) That the reporting of dry assay buttons of metal as tin is worse than bad assaying; it is dishonest. (2) That the SnO_2 gravimetric assay is probably the worst analytical method in common use in inorganic analysis. (3) That the omission to effect complete solution of the ore as an essential part of any method makes that method unsound, not only for dressed ores, but for poor material such as slimes and tailings. (4) That the use of direct fusion assays as against the gas reduction method involves the possibility of low results, and with errors of commercial importance. (5) That the ferric chloride titration of stannous chloride as described in my book is the best and soundest method of determining the tin.

The use of nickel instead of iron in effecting reduction of $SnCl_4$ to $SnCl_2$ for iodine titration is, in my opinion, a source of possible error, as nickel reduces $CuCl_2$ to Cu_2Cl_2 without precipitating the copper as iron does, in which case, unless enough potassium iodide is added, reversal of the iodide copper reaction may take place to an appreciable extent, the iodine solution being reduced and Cu_2I_2 and $CuCl_2$ formed. In any case, on dressed tin ores, the use of the iodine assay is an unsound procedure."—L. Parry.—*London Mining Journal*, Oct. 16, 1909, p. 98. (A. R.)

The Determination of Nickel in Presence of Cobalt.—"J. A. Sanchez (*Bulletin Société Chimique*, pp. 641 to 647) finds that when a neutral solution of cobalt and nickel salts is treated with sufficient potassium cyanide to re-dissolve the precipitate first formed, the liquid being then evaporated to dryness, the potassium cobaltocyanide is oxidised to the cobalticyanide, the potassium nickelocyanide remaining unchanged. Since the cobalti-cyanide forms a sparingly soluble silver salt, the method may be employed to separate cobalt and nickel. The sulphides of these metals being insoluble in warm dilute hydrochloric acid, can be separated by this means from the other metals of the same group, and after being dissolved in aqua regia and evaporated to dryness, are taken up with water, a 10% solution of potassium cyanide being added until the turbid solution becomes limpid. After evaporation to dryness, the residue is taken up with water and treated with an excess of silver nitrate, which throws down silver cyanide and cobalti-cyanide. The filtrate contains all the nickel which is precipitated as sesquioxide after the excess of silver has been removed by potassium bromide. The washed precipitate is warmed to 60° to 70° C. with a little water, 20% solution of potassium iodide, and 20% sulphuric acid, the deposited iodine being titrated with 1/10-N thiosulphate solution. To estimate the cobalt, the two metals are thrown down as sesquioxides, re-dissolved in hydrochloric acid, and the cobalt is found by difference after the determination of the nickel."—*London Mining Journal*, Oct. 2, 1909, p. 13. (A. R.)

METALLURGY.

Zinc Smelting.—"Pure zinc sulphide can be roasted completely to zinc oxide, provided the conditions are such as enable the resulting zinc sulphate to undergo decomposition and allow free escape to the sulphur dioxide or trioxide formed. The imperfect elimination of sulphur in roasting blende is due—apart from incomplete decomposition of the zinc sulphate—to the presence of iron, either as oxide (which forms ferrous sulphide, and also prevents the roasting of a portion of the zinc sulphide, causing a loss of 6% of sulphur), or as pyrites, which, in spite of the presence of a large proportion of calcium carbonate, checks the formation of sulphate and gives a loss of 9½% of sulphur. A still more injurious effect is due to calcium carbonate, which is transformed into sulphate, and causes a loss of 12% of sulphur. In reduction distillation, a portion of the calcium sulphide formed by the reduction of calcium sulphate, reacts on the latter to form lime. The metallic sulphides are attacked by the carbon, a portion of the zinc sulphide being volatilised, and the ferrous sulphide still more strongly affected. In the muffle, with exclusion of air, 2 molecules of ferric oxide react on 4 molecules of zinc sulphide, the simultaneous presence of carbon leading to the formation of metallic iron and zinc, together with carbon monoxide and carbon disulphide, this substance being also formed during the reaction of carbon on zinc and iron sulphides. In the presence of zinc oxide, ferrous oxide and manganous oxide, no calcium sulphide results from the reaction of carbon on calcium sulphate, lime being alone produced. The practical conclusions drawn are to the effect that, whilst the iron almost invariably present in zinc blende has a slightly injurious effect by preventing the roasting of a small portion of the zinc sulphide, it is very useful in zinc smelting when present in the form of ferric oxide. The reasons for this are, firstly, that it reduces to the condition of oxide any of the zinc sulphide that has escaped roasting, and thus renders the same accessible to the processes of reduction and distillation; and, secondly, because it fixes the sulphur resulting from the reduction of calcium sulphate, and prevents the same from combining with zinc. Consequently, in order to increase the yield of zinc, it is advisable to add ferric oxide when charging the muffle in the case of roasted calcareous blende. On the other hand, the necessity for such addition could be obviated by eliminating the greater portion of the lime from the ore previous to roasting—if possible, concurrently with the washing of the ore. In such case the roasted blende would be as simple to treat as roasted calamine, the amount of ferric oxide in the ore being sufficient, in the absence of calcium sulphate, to convert any unroasted zinc sulphide into zinc oxide, and render it available for reduction to metal. No appreciable increase in the yield of zinc can be expected to result from the diminution of the percentage of sulphide sulphur, so long as no measures have been taken beforehand to prevent the formation of sulphate sulphur, the latter being transformed into the former in larger quantities as soon as the reduction process begins, and combining with the metal and not with the calcium. Hence, more extensive roasting would merely increase the cost and lengthen the process, without any appreciable increase in the production of zinc.—V. Lepiarczk.—*Metallurgie*, July 8, 1909, p. 326. (G. H. S.)

Ore Dressing in the United States and Mexico.—"Occasionally gold quartz ores of workable grade are found, whose tailings, after amalgamation, do not carry sufficient value to warrant either cyaniding or concentration. Such conditions are infrequent, however, and usually only temporary.

In general, nowadays, the tailings from amalgamation are given subsequent treatment over stratifying tables, vanners or canvas or all these, for the recovery of sulphides, the sulphides being either shipped to the smelter, roasted and chlorinated, or finely ground and cyanided.

Where the tailings, even after careful concentration, carry sufficient gold or silver to warrant further treatment, cyaniding is usually found applicable as a supplementary process.

Frequently where cyaniding has been employed as an adjunct in this way to an amalgamation concentrating process, it is in time found profitable to discontinue amalgamation and run as a straight concentrating cyaniding process, with crushing in KCy solution. The decision as to this practice will depend on the amount of amalgamable gold and to what extent the advantage of catching part of the gold early in the treatment, possibly with a slightly higher total recovery, offsets the added expense of amalgamation, coupled with the necessity of subsequent pulp thickening before adding KCy solution and of wasting larger volumes of weak KCy solution than when crushing in solution.

The Dos Estrallos mill at El Oro, Mexico, is an illustration of this transition in treatment methods, in that it started off as a pan amalgamation mill, then as a cyaniding plant with amalgamation discontinued, and later crushing in KCy solution was adopted.

At the Homestake of South Dakota on the other hand, where, some years ago, cyaniding was added as a supplement to the simple amalgamation treatment, it still serves only as a supplement crushing in water along with amalgamation being practised the same as previously.

One of the best examples of the combined amalgamation—concentration—cyaniding system of treatment is probably found in the recently constructed mill of the Goldfield Consolidated Company, amalgamation being carried out at this plant both on the battery discharge and on the reground sands after passing through tube mills, with subsequent concentration of all slime over No. 3 Deister tables before de-watering and addition of cyanide solution.

Equipment.—Neglecting for the present the cyaniding feature which will be considered more in detail under the 'Cyaniding Group,' the equipment of mills coming under the 'Amalgamation Group' presents much of a sameness.

The initial crushing machinery may vary a little and be either gyratory or some of the various jaw crushers of the Blake type, but almost invariably the subsequent crushing is done by stamps. The tendency is in general toward heavier stamps, but outside of this and such lesser features as individual stamp guides, concrete mortar blocks, etc., there have been no important changes in the design of stamp batteries in recent years.

An exception to this statement is found in the Nissen stamp, with its circular individual mortar, but this has so far found its best application in crushing finely disseminated concentrating ores and will be considered under the 'Concentration Group.'

For concentration after amalgamation, hydraulic or surface classifiers are usually used, one or more sizes made from the sands for concentration over stratifying tables, and the slime thickened and sent to vanners, No. 3 Deister, or some of the various other slimers now manufactured. Sometimes, particularly in California, mechanical concentrators are not used on the tailings after amalgamation, but the total tailings are sent over canvas tables of varying slope with intermittent washing off of the concentrates.

Cyaniding Group.—The cyaniding ores of the United States, as so far treated, with a few notable exceptions, have been largely gold, the silver being of lesser value; as a consequence a large number of the milling plants come under the amalgamation-cyaniding class of treatment, instead of the straight cyaniding here considered.

In Mexico, on the other hand, where the ores in general contain relatively large silver values, amalgamation is rarely practiced in conjunction with cyaniding, and at most plants in Mexico crushing is now done in cyanide solution.

Until some five years ago, slime in connection with cyaniding was as undesirable as is still the case in strictly concentrating ores. About this time, however, the El Oro Mining and Railway Company began extensive experiments * looking to an improvement over the low extractions from leaching battery sands, and demonstrated conclusively that a large portion of the gold value of these sands was silica-encased, and this was not released for dissolution by cyaniding until the ore was ground to approximately 150 mesh. These experiments also showed that by fine grinding not only was the extraction increased, but the period of treatment shortened.

El Oro was the pioneer in the use of tube mills on an extensive scale for fine grinding in connection with the cyaniding process, and from the success at this camp with gold-silver ores the cyaniding of more distinctively silver ores was developed at Guanajuato and at Pachuca.

Higher percentage extractions can be made on a given ore in the condition of slime than as sand suitable for leaching, and for this reason wherever the gold-silver values of the ore justify the extra expense, cost of power, etc., duly considered, not only finer grinding, but all sliming is usually adopted as the process giving maximum profit.

This is the process at most of the plants of El Oro and Pachuca; it is also the process contemplated for a large plant now being figured on at Guanajuato. In the United States, the Montana-Tonapah and the recently constructed Goldfield Consolidated are on this all sliming basis of treatment, and at the Homestake Mining Company's large plants, treating both sand and slime, regrinding of the coarser of the sands now being leached is being adopted, necessitating additions to the slime treatment plant.

In the matter of concentration before cyaniding, much variation in practice is found. Some plants cyanide direct without removal of sulphides, experience with their ore leading them to decide that though a slightly higher percentage recovery of the gold-silver values in the concentrates might be obtained by their removal and separate treatment by smelting or cyaniding with stronger solutions, finer grinding and longer treatment, yet in commercial results it was more profitable to adopt the simpler treatment system with the slightly lower recovery of the value in the sulphide.

Some plants, on the other hand, concentrate the battery discharge, either sand only or sand and slime, and still others, with higher grade more refractory sulphide, find it profitable to go into the matter still further and concentrate also the slime resulting from tube mill grinding of the concentrated battery sand.

Equipment.—The modern cyanide plant on an all sliming basis usually starts off with gyratory or jaw crushers, followed by stamps crushing in solution. The battery discharge is classified into slime for further treatment and sand for regrinding either by cones, or in the more recent plants by mechanical classifiers. The sand thus delivered is thickened for regrinding in tube mills.

The tube mills themselves are usually trunnion type or trunnion bearing at feed end, with tire on roller supports at discharge end, thus permitting a large discharge throat into which can be shoveled pebbles or mine ore for similar purpose, using a reverse worm in feeding, coupled with a baffle plate of the Neal type in discharge. The lining is usually of the El Oro ribbed type. The usual trunnion feed has been a fixed pipe with stuffing box, but of late at many plants a smaller curved pipe has been used, which fits loosely into a round hole of larger diameter in the vertical plate covering of the feed end. Through this the feed is projected as a jet into the mill.

The final slime is then agitated, either by revolving arms in round tanks of moderate depth with or without the aid of centrifugal pumps, or more recently by tall Pachuca tanks. These latter employ a central tube for agitating by the air lift principle. They were originally designed in New Zealand and have within the last year or two been adopted by many of the newer plants.

For filtering the treated slime, either with or without previous washing by decantation, vacuum filters, pressure filters, or filter presses are used. Of the vacuum filters the type most largely used is the Butters or Moore; of the pressure filters, the Burt, and of filter presses the only one used is the Merrill sluicing discharge type.

The initial and the largest Merrill filter press installation is at Deadwood, S.D., where 26-96 frame presses handle 1,500 tons per day of Homestake slime. This is a rather crystalline slime and enables a porous cake to be made with frames of 4 in. thickness.

It was not thought that slime of flocculent character could be satisfactorily filtered and washed in such a press as a commercial proposition, but within the past year experiments have shown it entirely feasible to fill the frame only about 80 per cent. full, leaving a thin centre parting through which washing can proceed both ways through the cake. This is the system employed at Silver Peak, using 3 in. frames, and will also be employed for a certain class of slime at the new Esperanza plant at El Oro. The standard 4 in. Merrill frame will be used, however, for the reground sand slime.

For precipitation, zinc shavings is still the usual method, returning short zinc to the boxes and omitting acid treatment. Precipitation by zinc dust is, however, gaining in favour, and whenever acid treatment of the zinc dust precipitate can be dispensed with its use will show an economy, both in labour and in supplies over the shaving method, coupled with equal metallurgical results.

---

* "The Cyaniding of Ore at El Oro," *Trans. Inst. of Min. & Met.*, Oct., 1904. "Fine Grinding of Ore by Tube Mills and Cyaniding at El Oro, Mexico," *Trans. A.I.M.E.*, *Vol. XXXVII*, 3-55.

For the average cyaniding ores, say of Nevada, and particularly Mexico, the amount of gold and silver present in the ore and therefore in the pregnant solution is sufficient to insure a low ratio of zinc consumption to bullion, say not over 4 : 1, and in such case the cakes of precipitate from the press can be melted direct without the above-mentioned acid treatment.

Chlorination.—A considerable tonnage of telluride ores is handled in Colorado by chlorination, although cyaniding is competing with it to a considerable extent. Outside of these, chlorination is little used, except for the local treatment here and there of concentrate from amalgamating plants.

Lixiviation treatment of silver ores by hyposulphite solutions has been in recent years almost entirely displaced by cyaniding.

Leaching of oxidized copper ores by acids or solutions of metallic salts has been used to some extent, for example, at the Arizona Copper Co., Clifton, Arizona, and the Snowstorm, Coeur d'Alene, Idaho, but the application of such treatment is rather limited.

Flotation processes of the Broken Hill type, using acid or acid salts solutions have as yet not passed the experimental stage in this country and Mexico. The same may also be said of the Elmore oil and oil vacuum processes.

Some interest has been aroused by the flotation of sulphides on water by the use of Macquisten tubes with internal helical grooves at the Adelaide plant in Nevada, and it is possible this process has a field in the separation of sulphides from gangues of heavy specific gravity—say garnet gangues, where water concentration does not make a satisfactory separation.

Magnetic processes, in addition to iron ores, have found extended application in the separation of zinc blende from iron or iron and copper sulphides, either by removing the blende from the raw sulphides or the sulphides after a slight roast from the blende. The Wetherill belt type and rotary, the International rotary, the Ding and the Cleveland-Knowles, are at present the more usual machines for these separations.

Static separators of the Blake-Morscher or S.S. & S. type are sometimes used for similar separations.

The Sutton, Steele & Steele Dry Concentrator does very fair work where for any reason it is not possible to use water concentration. Where the feed is very closely sized by screening this table yields rather better results in the separation of minerals of close specific gravities—say blende from iron and copper sulphides. The separated products are then treated by the usual water concentration by tables of the Wilfley type. Close sizing by fine screens, where any considerable tonnage is handled presents, however, many difficulties."—H. A. GUESS. —*Proc. of the Colorado Scientific Society*, Sept., 1909, p. 236. (H. A. W.)

CONSTRUCTION AND OPERATION OF THE STAMP MILL.—*Stamp Duty.*—Power consumption, which is most economical when the weight of stamp is as great and the drop as low as practice will allow, is not a factor guiding the operation of a mill, but rather the duty per stamp is looked to, and this is controlled by the capacity of the mill to perform two distinct functions: firstly, to reduce the ore to the required fineness by crushing, and secondly, to expel the pulp through the screen; and it is a fact that the first operation is performed more effectively than the second. An examination by grading the pulp which has passed the screen will show that a large portion of the sands, varying from 40 per cent. to 60 per cent., is ground much finer than the corresponding screen openings, proving that unnecessary work is taking place due to excessive crushing (*a*). Therefore, in endeavouring to attain the highest stamp duty, the speed and height of drop of stamps must be arranged to give the largest capacity of screen discharge. Further reference to these points will be made subsequently.

Vibration.—In a stamp mill two sets of vibrations are set up; one, caused by the impact of the stamp dropping on the ore overlying the die, and the other when the tappet, in the act of raising the stamp, strikes the cam. The first set, which is unavoidable, is confined in its effects mainly to the mortar box and pile block, and any attempt to reduce it, additional to existing provisions, would result in loss of efficiency. The second set of vibrations is more widely distributed over the structure, and more harmful in its effects; it is transmitted by the king-post direct to the sills causing them to spring, and it is the chief cause of all trouble in the mill with nuts and bolts. Though demanding full attention, little has been done in the way of minimising this trouble. Where king-posts rest in cast-iron shoes a rubber cushion is placed under the foot of the post, and where a similar cushion inserted between the cam-shaft box and its seat vibration would be absorbed at a point where the most good could be effected.

Framing.—With the usual type of mill at work to-day, the weakest point in the design of the framing is the method of carrying the king-post on the strake-sill. The strake-sill has to bridge the distance between the mud-sills lying on either side of the pile block, and there is wanting, as a consequence, the necessary stiffness and rigidity at this point. The vibration transmitted by the king-post, already referred to, causes the strake-sill to spring, with an increasing bad effect on the whole structure. Various attempts have been made to overcome this difficulty: (1) By placing an additional thickness of strake-sill under the king-post; (2) by extending the pile block lengthwise, so that the section of strake-sill carrying the king-post may receive direct support therefrom; (3) by extending the king-post to the bottom of the pile pit. In some recent mills the king-post is carried on a rubber cushion in a cast-iron shoe. The shoe is secured to a concrete foundation built up between the pile blocks of adjoining sets of stamps, or, in place of an iron shoe, a short hard-wood sill may be substituted. Where, instead of wood piles, concrete foundations (*b*) are used, the block is extended out to carry the foot of the king-post. By the adoption of either of these last-mentioned methods a great improvement in the running of the mill is obtained, and the cost of many heavy wood sills, always in a state of deterioration, is at the same time saved. The counter-shaft bearings then can with advantage be carried on separate concrete blocks, and the ore bin foundations built quite independently of the mill framing—as, in fact, they always should be.

Bolts.—The most important bolts in the mill are those holding down the mortar box to the pile-block and the king-post to the strake-sill. The longevity of the mill depends on the king-post bolts being maintained in good order, with the nuts tight. In many cases these bolts are so placed as to be difficult of access; they are consequently liable to neglect, with the result that the mill may be thrown out of centre and the true alignment of the stems affected. The mortar box bolts are often smashed by mill water-mains and other avoidable obstructions; and the position of the king-post bolts is too confined to allow a spanner to be used on the nuts with proper effect. The latter bolts are best made in two lengths and joined in the middle with hooked ends. This method enables a defective screwed end to be replaced by unhooking the upper half of the bolt, instead of having to dig down into the foundations as would be the case with a solid bolt. The king-post is best held down to the strake-sill by bolts attached to a heavy yoke, which should be fixed at a height clear above the feeder platform (instead of underneath, as is usually done); in this way the nuts of the bolts are always in sight and accessible with a heavy spanner (*c*).

Centre Marks.—With the narrow mortar boxes now in use and the consequent small clearance, back and front, for shoes and dies, it is advisable to have permanent marks, such as brass arrows, let into the king-posts and guide-beams, in order to denote the exact stem centres both lengthwise and crosswise of the mill. By this means the original centres are always maintained, and work is facilitated when 'taking up' the wear in the guide blocks.

Screens and Discharge Area.—Seeing that the duty per stamp is controlled by the capacity of the screen to discharge the crushed product, it may be wondered why no attempts are made to increase the screen area. Years ago, boxes with double discharge were tried with not very encouraging results; mill men were not in favour of this arrangement on account of the inaccessibility of the back screen. From its position it was more liable to damage than the front screen, and the consequent loss of time in renewal counterbalanced the extra discharge gained while in use. Also, the table had to be lowered into a more inconvenient position in relation to the lip of the mortar, in order to receive the flow from the back of the box.

With, in some recent arrangements, the abolition of the amalgamating table, or even the removal of the table from actual proximity to the mortar, this latter difficulty disappears, and the damage formerly caused to the back screen due, chiefly, to large lumps of rock finding their way into the box, would be minimised, now that so much attention is devoted to the sizing and regularity of the feed. As to inaccessibility, this could be materially remedied in the original designing of the mill. For the back screen, a high discharge, say 6 in., would be necessary to avoid banking against bottom edge of screen under the feed shoot. Under these altered conditions it is quite probable that a trial would show an increased tonnage in a box provided with screens back and front.

Mortar Boxes.—Boxes are now made as narrow as possible, with corners rounded in order to avoid waste spaces. The inside depth, measuring down from the screen opening, varies from 4 in. to 9 in.; the shallow box has the advantage that the height of discharge may be regulated by chuck-blocks placed under the screen frame; whereas, in a deep box, wear has to be compensated by packing up the dies.

Liners.—Mortar box liners are made of hard cast iron, mild or cast steel, and lately of manganese steel. Many liners still supplied are of poor design and awkward to fix and renew. A proper set of liners should be interlocking and self-supporting; they require but one hardwood wedge to be driven down at either end of box to keep them fast and secure. A complete set of such liners can be renewed within 10 minutes from the time the dies are removed from the box. Liners should not extend below the level of the false bottoms.

Height of Discharge.—The height of discharge should be just so short as to avoid banking of sands against lower edge of screen; say, from 1½ in. to 2½ in. above top of dies, according to depth of feed employed. On many of the small mines in Rhodesia where the gold is coarse and free, inside amalgamation is in use, and rightly so; in such cases the height of discharge can be slightly increased with advantage to allow more time for intimate contact of the gold with mercury.

Shoes and Dies.—Dies should be turned about once in two weeks in order to equalise the wear; the highest edge should then be placed at the back of the box. Dies and shoes should be so proportioned that they wear out simultaneously; a 7 in. die will usually last out a 12 in. shoe.

Speed and Drop.—The best results are obtained with a speed of 95 to 105 drops per minute with a corresponding fall of 8½ in. to 6 in., varying according to the type of mill and quality of ore to be crushed. It should be borne in mind that with a set drop of 8½ in. the effective drop is only from 7½ in. to 8 in.

Guides.—Hardwood guides are good and reliable, and with proper care will last for years. The 'Ralok,' an improved guide supplied by the Sandycroft Foundry Company, Limited, is coming into favour; it is of cast-iron, made in halves. The bore is provided with annular grooves to form receptacles for the lubricant, consisting of four parts soft soap to one of graphite. The lubricant only requires replenishing at long intervals, and is so efficient that the wear in a 1,250 lb. mill is stated to be only about 1-32 in. in twelve months; chipping faces provide means for taking up the wear.

Plates and Amalgamation.—Plain copper plates are most in favour in this country, they are more sensitive and absorbent, and once set with gold amalgam will hold a thicker film of mercury than those silver-plated. The difficulty in the initial stages met with in the use of plain copper is due to the fact that a small quantity of copper goes into solution in the mercury (*d*); this is easily attacked by oxidising agents and by soluble salts of the ore, forming a green and tarnished surface to the plate, which prevents proper amalgamation. The alternative is to use silver-plated copper, by which means the above-mentioned trouble is avoided, and good amalgamation can be obtained from the start. The effect of plating is, however, to render the copper less absorbent than before. When steaming great care must be used in scraping or the silver-plating will be removed with the amalgam, leaving the copper bare: hard scraping is at any time liable to injure the plated surface. Should the mill water be acid, the silvered plate will scour unless lime is added to the feed to neutralise the effect.

A modification of these two methods is sometimes adopted by using silver amalgam as a first dressing on the bare copper plate, enabling a good gold recovery to be made from the start, and producing a surface more absorbent and sensitive as an amalgam catcher than silver-plated copper. The habitual use of cyanide solution in dressing plates is to be condemned; its use, except where special precautions are taken, usually results in some loss of gold, and, although it enables the plates to be easily kept bright, the amalgam never acquires that solid and sticky condition induced by proper scouring and rubbing of the plate. Gold amalgam being a solution of gold in mercury, increase of temperature of feed water, when it can be maintained without variation up to, say, 80 deg. F., is conducive to good amalgamation by causing increased absorption of mercury by the freed gold and by the plates.

Feed Water.—The water supplied to the mill is governed not so much by the conditions in the mortar box as by the flow over the plates. If the supply is in excess, 'black sands' and fine gold will be carried away; if inadequate, slimes and fine sands will adhere to the plates. The finer the rock is crushed the more water will be required to keep the plates free from settlement. Of the various methods of introducing water into the mortar, the best is to direct the stream so that it first strikes low down the back of the box; the water then spreads and washes back over the dies in not too strong a stream to remove the uncrushed ore from the dies, but is sufficient to carry with it towards the screen the fine particles ready for discharge. The mill should be supplied with water under a constant head. The branch leading to each box should be fitted with two cocks (*e*) or valves: the one to be used as a regulator and to remain set to pass the required amount of water and not otherwise touched, the other to be used whenever the mill is stopped or started for opening and closing the total supply to the box. In many mills the branch pipe is fitted with but one valve or cock, rendering adjustment necessary every time the mill is started.

Mercury Losses and Traps.—Losses of mercury in amalgamation can only be properly considered in conjunction with the tonnage crushed and plate recovery, and the quality of the ore must also be taken into account. A comparison, therefore, is not very easy. Under ordinary conditions with a 7 dwt. plate recovery the loss per ton crushed, including the process of retorting, should not exceed 1 dwt. of mercury. Where inside amalgamation is practised in the small mills of Rhodesia, the losses in many cases are known to be high; wherever this is the case, not one, but two or more mercury traps of good design should be inserted in the tailings launder, the benefit derived will be apparent on examination of the contents of the traps. These traps add nothing to the cost of working, and advantage of a suitable stop can be taken to clear out the collected sands.

Blankets.—Where used as concentrators it is a good scheme, instead of using fixed strakes, to have the blanket (*f*) mounted on rollers as an endless belt, and to revolve it slowly against the stream of pulp; the under side of the blanket passing through a trough of water is washed automatically, thereby saving the dirty and disagreeable work of constant hand washing.

The treatment of the pulp after leaving the mill will be dealt with in a subsequent article.

Notes by the Editor.—(*a*) This is not conclusive: each blow a stamp makes, produces a product of varying fineness, not of the same fineness.

(*b*) We consider all modern mills wherever possible should have good concrete foundations, if not too costly.

(*c*) We should be glad to learn more of the use of self-locking bolts in mills.

(*d*) Is it correct to state that copper goes into solution in mercury? We believe it would be more correct to say that mercury wets copper and gold.

(*e*) We commend Mr. Fison's remarks on mill water supply using two cocks instead of one, and we would further supplement his suggestions by introducing the use of a dial on the first cock, so that the exact opening can be seen and maintained.

(*f*) The use of blankets, as Mr. Fison suggests, in special cases is of undoubted advantage and economy."—G. H. FISON.—*London Mining Journal*, Sept. 4, 1909, p. 297. (A. R.)

MINING.

MODERN PROGRESS IN MINING AND METALLURGY IN THE WESTERN UNITED STATES.—*Mine Mapping.*—"Not many years ago most mining companies thought it amply sufficient to have a surface map of their properties and a composite map showing the different underground workings in their mines. To-day, almost every important concern maintains, in addition to the above, both stope and assay-maps, while many of the larger companies add individual-level horizontal and vertical cross-section maps showing the underground geology in full. Upon these maps conventional designs in black ink are used to designate the various rocks, while the different veins or vein-systems are shown in colors. These sections are frequently drawn also upon glass sheets, which are then inserted in wooden frames provided with vertical or horizontal slots or grooves cut at the proper relative distance apart to correspond with any desired planes of cross-section or with the working-levels of the mines in question. The great advantages of such plans and sections cannot be overestimated. They not only show at a glance the tonnage and value of the ore in sight, but also afford a guide for development-work, whereas the old-fashioned maps were nothing but a record of the work performed, and were practically useless for any other purpose. The improvement named has brought about another, the importance of which is just beginning to be recognized. I refer to the employment by large mining companies of economic geologists, who are not burdened with the duties of surveying, directing workmen, etc., but give their whole attention to the geological problems encountered in the work. The advice of such experts in the purchase of property, the running of exploration-drifts, the location of shafts, etc., and the interpretation of local fault-systems, and other structural features, has already proved of inestimable value to their employers.

Surface Mining.—Large ore-bodies occurring near the surface can, in many cases, be most cheaply and satisfactorily mined by stripping off the overburden and loading the mineral into cars by either the 'milling' or the steam-shovel system. In the West, steam-shovel mining is confined almost entirely to the low-grade copper-deposits at Ely, Nev., and Bingham, Utah. The system employed follows closely that of many iron-mines of Minnesota; and 95-ton shovels, with 3.5-cub. yd. (7 ton) dippers, are in common use. At Bingham, the Boston Consolidated Co. stripped the overburden from its deposit at the rate of 200,000 tons per month. The maximum amount handled in a calendar month was 282,903 tons, in August, 1907, and the maximum tonnage for a single day, with four shovels, is 15,000 tons.

Mine-Hoisting.—The favorite plant in Colorado in the early days was called the Gilpin county hoist, and consisted of a rope-drum securely fastened at one end to a large wooden pulley connected by a slack belt to a stationary engine running continuously at a slow speed. To-day, these primitive machines, large and small, have all been succeeded by direct or gear-connected steam-engines, equipped, whenever the tonnage is sufficient to justify the expense, with variable cut-off valve-gear, post-brakes, and every modern improvement.

Steam-hoists, capable of handling from 10 to 20 tons of total load, from depths of from 4,000 to 6,000 feet, at speeds varying from 4,000 to 5,000 feet per minute, are now not at all uncommon. These immense plants are fitted with every imaginable device for increasing the efficiency, rapidity, and safety of operation, and the skill and artistic ability displayed in the design of some of the later Nordberg creations bring them to a point where they can almost be considered works of art.

In many places where water fit for use in boilers is scarce and electric current cheap, as at Cripple Creek, the electric hoist has almost completely replaced the steam-hoist.

When large electric hoisting-plants were first installed, it was found that the great amount of current necessary to start and accelerate the load brought a very objectionable " peak " on the transmission-line. This difficulty has now been overcome by the Illgner and other similar systems, in which the energy stored in a large rapidly-revolving fly-wheel cuts down, if it does not entirely prevent, the objectionable peak.

Underground Tramming.—When mines were shallow, shafts numerous and hoisting facilities inadequate, hand-tramming was almost universally employed, but with increase in depth came the necessity for better hoisting-machinery and a reduction in the number of shafts, thereby increasing the distance over which ore had to be trammed. Then it was found that a high-priced man constituted a very expensive motive-power for pushing cars; horses and mules were put into commission; and, still later, air- and electric locomotives have come into very general use. There has been much discussion concerning the relative merits of these two latter systems of underground haulage; but there is no doubt that each has its own field. Where the openings are dry and the roof sufficiently high and firm to carry the trolley-wire insulators, there is no question as to the desirability of using electricity, but where these conditions unfortunately do not obtain, the compressed-air locomotive is an excellent substitute.

Timbering.—Where the ore-bodies do not exceed 10 or 12 feet in thickness, and have a firm hanging-wall, nothing can exceed the cheapness and simplicity of stulls; but when large ore-bodies are encountered and timbering is necessary, the system commonly employed is that of 'square-setting,' invented by Deidesheimer and first used on the Opir mine on the Comstock lode in 1861. Timber is yearly becoming more expensive, does not usually last well underground, and when the ore-bodies are large, especially if there is a tendency to movement in the walls, 'square sets' made from square timbers are apt to 'swing' and afford very little vertical support.

When large quantities of timber are required for square setting, in situations where the distance from the forest to the mine is not too great, round timbers are very much cheaper and more durable than square. A round log has about double the strength of a square timber cut from the inscribed square on its small end; and since, in the round log, the concentric rings of wood-growth are unbroken and each protects the ring immediately underneath it from decay, the comparison, both in cost and durability, is very unfavourable to square timbers. Automatic framing-machines can now be had which utilize the full strength of round timbers by making a bevel-joint outside of the square tennon necessary in all square-set timbering. This additional segmental contact-area in the joints braces the round timbers so that they are much less liable to 'swing' in large stopes than the square.

In some cases, 'cut and slice' and 'caving' methods are employed, in which the hanging-wall is allowed to come down and rest on each successive floor as the ore is stoped out, and is prevented from mixing with the ore by a mass of crushed timbers and plank which follows down on the top of the receding ore. In the Utah Copper and Boston Consolidated mines, at Bingham, Utah, about 4,000 tons copper-ore are mined daily by the 'caving' system. In some large mines the stopes are filled with waste as fast as they are freed from ore, and the ground above thereby prevented from caving, in the same way as if stulls or square setting were used. Steel and concrete are coming slowly into use in shafts, stations and tunnels, and with the natural decrease in the price of iron and cement on the one hand and the rising cost of timber on the other, it is easy to see that the more durable forms of construction will eventually supersede wood on all permanent work.

The direct-replacement system employed in the Rio Tinto copper-mines in Spain succeeded perfectly in holding both walls and surface in place on a vein from 200 to 260 feet wide, and, by taking advantage of the wonderful skill of the Spanish miners in building dry stone walls, gives a new method of safely and economically mining the lower portions of the lodes which cannot be reached by the open-cast systems extensively in vogue there.

Pumping.—In the United States, the old-fashioned Cornish pump, with its costly foundation, massive walking-beam, huge plunger-rods and ponderous balance-bob, was supplanted many years ago by the direct-connected steam-pump, which soon developed into a most efficient pumping-machine with duplex, triple expansion engines and every refinement possible in modern steam-engine practice. These have in many places been superseded by the electric-driven plunger-pump, in which the high speed of the electric motor has been reduced by suitable train-gear. Latterly, quite a large number of electric pumps have been built in which the gear is entirely eliminated. The speed of the motor has been reduced, and that of the plunger raised, forming a combination known as the express pump. Pumps of this class, with capacities of 1,600 gallons per minute, raising water 1,550 feet, are now being very successfully employed in unwatering the Comstock Lode at Virginia City.

Within the last few years great improvements have been made in the electric-driven turbine, which, with its entire absence of valves and reciprocating parts, threaten to dominate the field completely.

Already we have single-stage turbine-pumps raising 35,000 gallons of water per minute 150 feet high; five-stage pumps raising 10,000 gallons per minute 600 feet; six-stage pumps raising against 800 feet head; and eight-stage pumps raising 400 gallons 1,400 feet; and responsible firms are ready to contract to raise water by this system to any elevation up to 2,000 feet, and guarantee a pump-efficiency of from 60 to 75 per cent., according to conditions of service.

Mine-Lighting and Signalling.—Incandescent electric lighting has long since driven oil-lamps and candles out of underground stations and permanent levels in which any large amount of work is carried on. The recently invented tungsten lamp with its high efficiency, giving 20 c.-p. with an expenditure of only 25 watts per hour, makes it economically possible to extend electric lighting very greatly throughout underground workings. In some of the largest and most progressive mines, candles and oil-lamps have already been replaced in the stopes by acetylene-lamps, which are not only cleaner and safer, but give a much greater illumination for a given cost.

For mine-signalling, the flash-light system operated by interrupting, by means of well-protected switches, the current passing through the station-lamps is rapidly replacing the old-fashioned cumbersome bell-cord; and the latest moisture-proof mine-telephones give instant communication throughout the underground workings, and to and from the surface, with little danger of interruption,

Explosives.—The use of modern high explosives in mining and tunnelling is now universal; but there is a crying demand for an explosive which can be more safely handled, and which on explosion or detonation will produce a smaller amount of noxious gasses, which not only injure the health of the miners, but delay the resumption of work after each round of shots has been fired.

Mine Ventilation.—The ventilation and cooling of metal-mines have not yet received the attention which their importance demands. In this respect Western engineers could take profitable object-lessons from their brethren in the coal-fields. Very few of our Western mine-operators go to the trouble of recording temperatures and making ventilation-maps, showing the direction of the air-currents, etc., all of which data are necessary before a satisfactory system of either natural or artificial ventilation can be planned.

The latest high-speed electric direct-driven centrifugal compressors give pressures up to 45 oz., and have been built in sizes up to 40,000 cub. ft. of free air per minute; but there is apparently no limit to the size of the machines which can be built under this system.

Tunnelling.—All over the United States, water-supply, hydro-electric power and the reduction of grades on railways are acquiring new and expensive tunnels, to which, in the West, are added the great irrigation-tunnels called for in both government and private enterprises. The result of all this activity in tunneling has been a vast improvement in both machinery and methods, and a greatly increased number of thoroughly trained and skilled workmen, so that records formerly unattainable in the United States are now being made in widely-separated localities.

Gold-Dredging.—Chain-bucket dredging for gold was first attempted in 1867 in Otago, New Zealand, and the first steam-actuated dredge operating on this principle was built on the Molyneux in 1881.

From a few small dredges copied after those in use in New Zealand, gold-dredging in this country has grown into a great industry, which is carried on successfully from the frozen gravels of the Arctic to the sun-scorched river-bars of the tropics, and at all altitudes from 10,000 ft. down to sea-level. Under stress of competition and the necessity of meeting new conditions, dredges have grown both in capacity and efficiency to a point not even dreamt of a few years ago. Dredges are now built with close-connected buckets, of capacities up to 13.5 cub. ft., and capable of handling 10,000 cub. yd. of gravel in 24 hr.; some have been built with bucket-ladders capable of digging 67 ft below the water-line and 20 ft. above it. A new stacker now under construction will deliver tailings 160 ft. away from and 60 ft. above the deck of the boat. Improved construction and better management have rendered dredging-operations less dependent upon weather; and last winter in Colorado a dredge was operated continuously at an altitude of 9,990 ft, where the temperature on several occasions fell to 20° below zero.

Electric Transmission.—Already the electric current is carried to all elevations from sea-level to timber-line and there is scarcely a desert mining-camp of sufficient size to justify the erection of a pole-line that is not equipped with electric power. The ease with which this overcomes the old and apparently insurmountable problems of scarcity of water and fuel constitutes one of the delights of modern mining.

The use of electricity has also completely solved the old vexed problem of underground sinking and hoisting, so that these operations are now as readily carried on from deep tunnel-levels as from the surface.

Transmission-lines of all lengths up to 220 miles are in daily use, with carrying-capacities ranging up to 40,000 kw. Long-distance transmission-systems are in many cases operated at 100,000 volts, and new lines are building to utilize even higher pressures. Recent improvements in insulation promise to make still higher voltages possible, which would mean a corresponding increase in the distance to which current could be profitably carried.

Sampling.—In milling and concentrating-plants, where fine crushing is a necessary preliminary, sampling is a comparatively easy and reasonably accurate operation, but where the ore is to be treated by blast-furnace smelting, crushing of any kind is objectionable and fine subdivision is prohibited. Forty years ago, for the valuation of coarse ore, 'grab' sampling was in common use, and this method was replaced in slow succession by Cornish quartering, fractional division, and split-shovel sampling. Then came automatic sampling in many forms, but all taking a portion of the ore-stream continuously. In 1884 a new system of sampling was invented which automatically deflects the entire ore-stream for a varying portion (usually one-fifth) of the time into the sample-division. Numerous different machines working on this principle are now in use; and these types of sampling-plants have been perfected to such a degree that where the hopper ore-cars which are now coming into general use are employed, ore may be unloaded, crushed, sampled, and reloaded into the outgoing cars, and the ground sample delivered in a locked steel box, without ever having been handled—the entire chain of operations being performed automatically.

Concentration.—The separation of valuable minerals from worthless gangue must have been one of the earliest operations in the history of metallurgy. Up to less than 100 years ago the pan, tub, and inclined plane, which are all so graphically illustrated by Agricola, continued to be the only devices in use. Hand-jigs were first introduced for the separation of coarser particles than could otherwise be handled, and the principles involved are in use to-day, although improved mechanical appliances have changed and enlarged operations to such an extent that the primitive origin would scarcely be recognised. About 35 years ago the use of air as a concentrating medium was successfully introduced, and, despite its many disadvantages, this system, assisted by numerous mechanical improvements, still exists and manages to hold its own where water is unobtainable or, for some reason, cannot be used. In skilful hands, some of the pneumatic separators give wonderful results, but the delicacy of the adjustments and the attendant dust will undoubtedly prevent any extensive employment of this method.

The pulsating water-current recently invented by a distinguished investigator in the concentrating field has already won a place for itself in both sizing and jigging operations, and promises to become a most important factor in concentrating-work.

Specific gravity is, however, no longer the only principle taken advantage of in mechanical ore-sorting. To these have been added magnetic and static electric separation, and many different methods based on the surface-tension of water (with or without the assistance of oil or acid), resiliency, and affinity for grease. The latter method, used very sparingly in this country, finds its chief application in South Africa, where it is used in the separation of diamonds from other stones.

The recent discovery by Kunz and Baskerville that the use of ultra-violet light would enable an observer to determine by inspection, with reasonable accuracy, the percentage of willemite in concentration-tailings, has already found commercial application on a very large scale, and opens up a wide field for speculation as to what the future may hold in store for us in this field.

The enormous size of some of the new concentrating-plants erected in the West exemplifies in a marked degree the magnitude of the operations now being carried on. At Anaconda, Mont., the Amalgamated Copper Co. has an eight-unit concentrating-plant, each section of which handles 1,000 tons in 24 hr.; and some of the new plants at Ely, Nev., and Garfield, Utah, are but little smaller in size.

Roasting for Blast-Furnace Smelting.—The earliest roasting-furnaces to prepare sulphide ores for blast-furnace smelting were small hand-operated reverberatories, with or without fusion-hearths. These were followed by revolving cylinders and various types of mechanically operated reverberatory furnaces, all of which were not only expensive to operate and keep in repair, but yielded a product very badly adapted to blast-furnace work. To-day these old-fashioned furnaces, both hand and mechanical, have been almost entirely superseded by blast-systems like the Huntingdon-Heberlein, Carmichael-Bradford, and Savelsberg, which make it possible to utilize the many advantages of blast-furnace smelting for the treatment of concentrates, fine ore, and flue-dust at very low preparatory costs. These systems marked a wonderful advance over the old reverberatory roast with its pulverulent product; but they were still open to the serious objection of requiring a large amount of manual labour in charging and discharging the pots and breaking up the sintered product. In spite of this drawback, the results obtained were so desirable that study and invention along these lines have been stimulated to such an extent that there are already in use mechanical roasting and sintering-plants (system of Dwight and Lloyd), to which ore can be fed in a steady stream, and which will automatically deliver a desulphurized, sintered, and broken-up product in the best possible condition for blast-furnace work.

Lead-Smelting.—The largest lead-furnaces in the United States have hearths 180 by 44 in., and treat daily from 150 to 255 tons of ore, according to its character. Mechanical charging is used in some cases; but while it slightly reduces operating-costs, it is no improvement metallurgically.

In Australia, where different conditions prevail, the improvements in lead-furnaces have kept pace with those in iron and copper-smelting.

Reverberatory Copper-Smelting.—In 1867, when Richard Pearce (afterwards President of the Institute) built his first reverberatory furnaces at Black Hawk, Colo., they were considered the acme of metallurgical perfection, and their successful operation did wonders for the mining industry of the State. The hearths of these furnaces were only 8 by 12 ft. in size, and their daily capacity was 12 tons. As the quantity of ore produced increased and the necessity for handling larger tonnages became apparent, the reverberatory furnaces have been steadily enlarged and improved, until to-day they have attained almost incredible dimensions, having a hearth-area of 19 by 116 ft. and a daily working capacity of more than 300 tons, which, in the case of easily-smelted ores, has risen to over 400 tons. These large furnaces secure great saving of heat, and uniform, continuous operation, for reasons into which I need not enter here.

Blast-Furnace Copper-Smelting.—The early water-jacketed blast-furnaces for smelting copper-ore were small, round, wrought-iron affairs, about 30 in. in diameter and rarely smelted more than 12 tons of ore in 24 hr. From this puny beginning, keeping pace with the rapidly growing copper industry of this country, furnaces have increased steadily in size and improved in mechanical construction, until they have reached the enormous dimensions of 17 ft. in length by 4 ft. 8 in. in width, with a daily smelting-capacity of 3,000 tons of charge. These furnaces are mechanically fed, work under an air-pressure of 40 oz., and give infinitely less trouble than their smaller progenitors.

The water-jackets are completely sectionized, and it is possible to renew most of the sections without stopping the furnace. At first only one tier of jackets was used, then two tiers came into use, and now some of the most recent furnaces have air or water-jackets replacing the brick superstructure, thus doing away with much of the roof-accretion nuisance. On furnaces built with a crucible, baby water-jackets have replaced the old cast-iron plates, and water-jacketed nose-pieces have greatly lengthened the life of the furnace-discharge spouts.

The recently invented acetylene blow-pipe promises to be of great service to blast-furnace engineers, as through its use it will soon be possible to obtain welded water-jackets entirely free from the objectionable lap-seams and rivets.

Electrical Smelting.—There is no doubt that in the near future many minerals now smelted with fuel will be reduced to metals by electrical processes. The electric current possesses the great advantage of allowing a most efficient utilisation of the heat; and also complete control of the exposure of the molten metals to air or gases.

Briquetting.—At many blast furnace plants 'fines' are made into briquettes with the ordinary die-and-plunger machines; but if free acid or copper sulphate be present, the surfaces of both dies and plungers are rapidly corroded, which obviously increases the diameter of the dies and diminishes that of the plungers. As this solvent action continues, a point is soon reached when the plunger no longer fills the die-opening, and pressure forces the material through the space between them, instead of consolidating the mass.

When a sufficient amount of plastic material, such as slimes, can be obtained to mix with fine ore and flue-dust, so as to give the mass the property of 'flowage' under pressure, it may be briquetted in machines similar to those used in making building-brick by the 'stiff-tempered' process. Solvents do not interfere with the operation of these machines, and by constructing the working-parts of steel and phosphor-bronze, it is possible to make briquettes dry enough to pass directly into a blast-furnace at the rate or 600 to 800 tons per day for each machine employed, at a cost of less than half of the die-and-plunger system.

Chlorination.—Chlorination is confined almost entirely to sulphide gold-ores carrying so little silver that its loss may be disregarded, and which require a preliminary roasting before treatment. The early methods of tank-leaching have all been superseded by barrel-chlorination, and in some of the latest plants chlorine is produced by electrolysis instead of by the decomposition of bleaching powder with sulphuric acid.

The most desirable features of this process of gold extraction are the high percentage recoverable, and the rapidity with which clean-ups can be made, rendering it easy at all times to know exactly what results are being obtained.

The largest plants operating under this system are situated in Colorado City, Colo., where two mills owned by one concern have an aggregate capacity of 800 tons per day.

Cyanidation.—While the first patent for extracting gold from its ores by cyanide solutions was issued in 1867, it was not until McArthur and Forrest took it up in 1889 that practical results of any value were obtained. Since that time the use of the process has increased by leaps and bounds in all of the principal gold producing countries, and to-day it is the principal factor in the world's steadily increasing gold production. Cyaniding seems to work with equal facility on raw ore, roasted ore and tailings, and with the steady improvement in the mechanics as well as the chemistry of the process, it bids fair to do even greater things in the future than it has done in the past. Nor is its use confined entirely to the extraction of gold. In many districts it is operating with great success on mixed gold and silver ores. At Millers, Nev., two plants with an aggregate capacity of 700 tons per day are operating on Tonopah ores, in which the average ratio of silver to gold is 80 to 1.

The largest and perhaps the most complex cyanide plant in the United States is the Golden Cycle mill, at Colorado City, Colo., which has a daily capacity of 1,000 tons, and treats exclusively Cripple Creek ore, all of which requires careful roasting and very fine crushing. The Homestake cyanide mill handles a larger tonnage, but treats only tailings. Next to this plant in point of size and importance, and handling ore of much higher grade, is the new mill of the Goldfield Consolidated Co., Nevada, which has a capacity of 600 tons per day, and contains the very latest improvements, culled from American, African, Australian, and Mexican practice.

Fume Recovery.—No department of metallurgy has made slower progress than this most important division, but now, through an unholy alliance between the unscrupulous contingent-fee attorney and the greedy land-owner, the success of 'smoke-farming' is compelling the smelting companies to do for self-preservation something which they should long ago have undertaken for profit. Out of the almost numberless devices which have been tried for fume-recovery, the bag-house affords the best solution of the problem yet devised. Bags were used for the recovery of both zinc oxide and lampblack more than fifty years ago, but were not used for fume-recovery until 1878, when Bartlett set up a small plant at Portland, Me. The first large successful installation of this kind was erected by the Globe Smelting Co., in 1885, and has been in continuous operation ever since. While the bag-house has been eminently successful in the recovery of blast-furnace fume, it cannot be used on fumes from reverbatory roasting or smelting-furnaces, owing to the fact that a portion of the sulphur dioxide formed by the oxidation of the sulphur is raised to sulphur trioxide by contact with incandescent ferric oxide. At the United States Smelting Works, in Utah, provision for protecting the bag-house from sulphur trioxide is made by blowing into the flues zinc oxide, which immediately absorbs the sulphur trioxide present to such an extent that it has been found quite possible to use cotton bags. The bag-house has proved very efficient in recovering fume from lead-refineries and all lead-smelting operations excepting roasting. In large copper-smelting plants, where large volumes of gas are produced carrying a sufficient amount of sulphur trioxide to destroy woven fabrics, three systems are in use—namely: radiation, decreased velocity, and friction—and in many cases two or more of these methods are combined. At the Washoe plant, in Anaconda, long steel-covered flues of enormous cross-section have been installed for a number of years, and, while this system does not effect a complete recovery of the fumes, it is as nearly perfect as it is possible to make a plant to-day. At Great Falls, Mont., the Boston & Montana Co. is installing the friction system at a cost exceeding $1,000,000, which includes the construction of a stack 506 ft. high and 56 ft. in diameter, and a dust chamber in the flue system, of such width that the furnace gases will pass through it at a velocity considerably less than 500 ft. per minute. From the roof of this flue chamber more than a million steel wires will be suspended, an arrangement which experiment has shown to increase greatly the settling efficiency of the dust-chamber. This installation is practically completed, but it will be some time before the results obtained can be accurately determined."—D. W. Brunton. *Transactions of the American Institute of Mining Engineers.*—Presidential Address. —Sept. 1909, pp. 837-855. (J. A. W.)

MISCELLANEOUS.

Simple Mineral Tests and How to Make Them.—"Judging by the number of worthless samples submitted to assayers to be tested for various metals, and by the fact that valueless minerals are often supposed to be valuable ores, it is evident that many do not realize how easily some elements may be detected. Naturally a prospector can not be expected to burden himself with the equipment of a chemical laboratory, but an outfit, consisting mainly of a few chemicals, test tubes, blowpipe and spirit lamp, adds little bulk or weight and may be of inestimable service. The writer knows whereof he speaks, for he once carried such an outfit over 2,000 miles, during 18 months in Alaska. Few tests can be described in a brief article, and the following have been selected on account of their simplicity, and also because some of the minerals mentioned are often confused with others.

The purer the substance to be tested the better, and, should the mineral be scattered through the rock in fine particles, it is usually well to first concentrate it by pulverizing the sample and then panning in water, the concentrates being subsequently dried, and, if necessary, ground finer. A clean 6 in. frying pan is very convenient both for washing and drying what is left.

Gold.—Gold ores, when crushed and panned, generally show either free metallic gold or sulphides, or other concentrates containing gold, which, however, is not visible. If these sulphides be roasted over a fire at a dull red heat, on thin sheet iron, say a piece of a tin can, and then reground in a mortar to polish the gold, the metal in many cases will be plainly seen on panning again. Should the material be oxidized, and when powdered and panned, show nothing but the magnetic oxide of iron, commonly called 'black sand,' the sample is apt to be worthless or very low grade, for the writer's experience leads him to believe that, as a general thing, the black sand itself obtained from rock in place carries little or no gold. The dried black sand can be readily picked up with a magnet.

Copper.—Greenish rocks are sometimes mistaken for ores of copper, which metal may, however, be easily detected in the following manner: Pour into a test tube, say half an inch in diameter, enough of the finely ground material to cover the bottom to a depth of about half an inch, add two or three times as much strong nitric acid, and boil gently over an alcohol lamp until the acid seems to have no more effect and red fumes have ceased to come off. Dilute the solution with one or two volumes of water and hold a clean bright piece of iron or steel, such as the blade of a pocket knife, in the liquid for a few moments, when, if present, a thin film of metallic copper will be plainly seen on the blade upon withdrawing and dipping it in clean water.

Instead of testing this solution with iron or steel, strong ammonia may be added in excess to the liquid when cool, that is, the test tube, after being well shaken, should smell of ammonia. The addition of the ammonia generally produces a reddish-brown precipitate due to iron, and the tube should be set aside for a few minutes until this settles somewhat. If a copper mineral, the liquid will have a deep blue colour; the more copper the deeper the colour. Nickel solutions similarly treated become blue, which however, is fainter than the colour produced by copper, and in any event the ores of nickel are apparently not widely distributed. There are many other tests for copper, but the two described require little time, knowledge or skill and are as good as any for practical purposes.

Tin.—Various minerals are often taken for tin ore, but as the ordinary ore, cassiterite or tin oxide, is alone infusible and proof against all acids, it follows that if a substance can be melted without a flux, or can be dissolved in acids, it is not cassiterite. After a few trials, tests for fusibility can easily be made with a blowpipe and spirit lamp, beginners, however, being apt to increase their difficulties by experimenting with pieces far too large. A thin splinter of a mineral, no thicker than the black lead in a pencil, is sufficient, and if such be held in a small pair of forceps, and the flame be directed with a blowpipe upon the end or edge, it will soon be seen whether melting takes place. Much grief, expense and useless prospecting for tin could have been saved before now had this simple test been at first made. If tin oxide be finely pulverized, mixed with about the same amount of powdered charcoal, and twice as much carbonate of soda, the whole being made into a thick paste with water and then heated on a piece of charcoal with a blowpipe, beads of metallic tin, can with a little practice, be readily obtained. Four or five parts of pulverized cyanide of potassium may be substituted for the powdered charcoal and soda. In so-called 'reduction' tests of this kind, the inner part of a yellow flame should be used, that of a candle answering most purposes. The tin globules may be tested by warming them in a test tube with nitric acid, diluted a little with water, when unlike lead or silver, they do not dissolve, but are converted into a white powder. Tin ore being heavy is easily concentrated by panning the crushed rock, and when pulverized may be of various colours, such as brownish, gray or purplish. In parts of the Carolinas and South Dakota the purplish colour of the tin concentrates is characteristic, and serves to distinguish the powdered cassiterite from other minerals.

Quicksilver.—Red oxide of iron and other minerals are sometimes thought to be the quicksilver ore, cinnabar, but this latter if heated with a blowpipe, vaporizes and disappears. Realgar, the red sulphide of arsenic, acts in the same way, differing, however, from cinnabar in giving off a smell of garlic while evaporating, and in burning on charcoal with a blue flame. Lead chromate and vanadinite, a lead vanadate, on account of their weight and colour may resemble cinnabar, but both these minerals blacken and fuse on a piece of charcoal before the blowpipe, and in the inner part of a yellow flame, such as that of a candle, yield globules of metallic lead. If cinnabar be scratched with the point of a knife the streak is red, which is also the case with hematite, while the lead minerals mentioned give yellow or whitish streaks. When red iron oxide is heated with a blowpipe on charcoal, in the inner flame, it soon becomes magnetic and thus can be picked up with a magnet. The writer was once employed for a time in sorting quicksilver ore, the grade of which was constantly estimated by crushing and panning samples, and noting the amount of cinnabar, which, being very heavy, is easily concentrated.

Tungsten.—Tungsten is readily detected, providing that the suspected mineral, especially when dark in colour, be first ground to a very fine powder. If scattered through the rock in small particles, the sample should be crushed and panned, and the concentrates tested, after being dried and ground as fine as flour. Should no heavy concentrates appear, the rock certainly does not carry tungsten in commercial quantity, and further testing for the metal would be a waste of time. Pour into a test tube, say half an inch in diameter, enough of the fine powder to cover the bottom to a depth of about half an inch, add three or four times as much strong hydrochloric acid, and boil gently over the spirit lamp for a few minutes. Two or three minutes is usually enough for scheelite, but hubnerite may require boiling for 10 minutes before sufficiently decomposed. Decomposition has gone far enough when the tube becomes more or less coated with a yellowish film, or when a yellow or olive green powder separates out. Now drop in a piece of pure tin-foil, as big as a pea when tightly folded, heat again, and if a tungsten mineral, the solution will become a beautiful deep blue colour. Continue boiling and the blue will change to dark brown, should enough tin and acid be present. If the brown fails to appear, more tin or acid may be added as an excess of either does no harm. The tin-foil in packages of tobacco will answer the purpose, but the pure article is much better, as the former is nothing but lead thinly coated with tin. The tungsten compounds are apt to stick to the test tube, which after being well washed out, may easily be cleaned by warming a little dilute ammonia in it. Barium sulphate, or 'heavy spar,' is often mistaken for scheelite, but the barium mineral, of course, does not react for tungsten, and unlike scheelite, which is very difficult to melt, a thin splinter of heavy spar may readily be fused, by holding it in a pair of forceps, and directing the flame of the spirit lamp upon it with a blowpipe. If a thin platinum wire be dipped into hydrochloric acid, then into powdered barium sulphate, and be held against the lower part of the alcohol flame until red hot, the flame will turn green, the colour persisting for some time. Powdered lead carbonate effervesces and dissolves when heated with dilute nitric acid, neither of which is the case with scheelite or heavy spar, if similarly treated.

Molybdenum.—Micaceous hematite, graphite and molybdenite are occasionally confused, but the former, unlike the other two, shows a red streak if scratched with a knife, and, when crushed, yields a red powder. Heat and acids have practically no effect upon graphite, but if pulverized molybdenite be roasted at a dull red heat, on a piece of a tin can, or in an iron spoon, with free access of air, a smell of sulphur is given off, and the flakes on cooling become yellowish or whitish. Molybdenum may be detected by evaporating the powdered mineral to dryness in a porcelain cup, with three or four times as much strong nitric acid, then heating to boiling with two or three volumes of hydrochloric acid, when the addition of a piece of pure tin foil and further warming causes the solution to become green, and finally brown, if molybdenum be present. If pulverized molybdenite be boiled in a porcelain cup, with three or four volumes of strong nitric acid until dry, and the heating be continued for a minute or two until copious, pungent white fumes come off the surface of the powder on cooling, and portions of the cup assume a deep ultramarine blue colour which may often be improved by repeatedly breathing into the cup while cooling. Thin porcelain cups, or 'crucibles' as they are frequently styled, are better than test tubes for evaporating solutions, and they may be obtained from dealers in assayers' supplies. Two inches wide at the top is a handy size, and the cup can be conveniently held over the spirit lamp by means of an iron wire, one end of which is formed into a loop. A quarter of a teaspoon, level full of the clean powdered mineral, is ample for the above tests.

Vanadium.—Most vanadium minerals may be sufficiently decomposed for testing by warming the fine powder for a few minutes with nitric acid over an alcohol lamp, preferably in a porcelain cup. About six volumes of dilute nitric acid, half acid and half water, will generally answer the purpose, unless the substance be difficult to decompose, when a longer application of a gentle heat and more of the dilute acid may be required. After the acid seems to have no more effect, and red fumes cease to come off, the solution is diluted with one or two volumes of water, brought to a boil, and set aside to clear. Settling may often be assisted by pouring some of the liquid into a clean test tube. When fairly clear, add a little hydrogen peroxide, and if vanadium be present, the solution will immediately become reddish-brown. The ordinary 3% aqueous solution of hydrogen peroxide, which can be had at most drug stores, works well, and about a quarter of a teaspoonful of the powdered mineral may be used. If desired the presence of vanadium may be confirmed by evaporating another portion of the powder to dryness in a porcelain cup, with dilute nitric acid at a gentle heat, and warming the residue with dilute sulphuric acid, when the addition of a piece of commercial sheet zinc causes the solution to become bluish-green, then green, then bluish-violet, and finally lavender. The same succession of colours is produced by heating the residue with hydrochloric acid and tin foil. The clearer the solution the better, and after heating for a minute or two with either of the acids, it is a good plan to let the liquid settle, and then to pour some of it into a test tube before adding the zinc or tin. The dilute sulphuric acid solution usually attacks the zinc in the cold, but the tin and hydrochloric acid should be warmed. If the zinc be in excess, and be allowed to remain in the sulphuric acid solution until chemical action ceases, and the liquid be then poured off, the zinc into another test tube, the lavender soon changes to a deep brown. On evaporating the powder with the dilute nitric acid, the presence of vanadium may be indicated by the formation of dark red rings on the sides of the cup, but the colour due to iron may deceive the unwary.

When a drop of strong nitric acid is placed upon a piece of the mineral vanadinite, which is a lead vanadate, a yellow coating soon appears. If the vanadium sulphide, patronite, be pulverized and heated, say on a piece of thin sheet iron, a strong smell of sulphur is perceived and the sulphur, some of which is free, on further heating, takes fire. A green solution is obtained when powdered patronite is boiled with three or four times as much strong nitric acid until no more red fumes come off, and on diluting the liquid with several volumes of water, allowing to settle, and then adding hydrogen peroxide, the characteristic reddish-brown colour immediately appears. Other minerals have been mistaken for patronite, which latter, as far as the writer knows, has hitherto been found in Peru only. This patronite ore not only contains a good deal of sulphur, but also carbon, and the writer once attempted to fuse some of it with nitre before burning off the sulphur, and soon discovered the mixture to be a very fair grade of gunpowder, little of it remaining after the application of heat.

Tellurium.—Tellurium is chiefly of interest on account of its occurrence in combination with gold, silver and other metals. Select as pure specimens as possible, or if in small particles, concentrate the supposed telluride ore by panning, dry and grind the concentrates to a fine powder. Heat with strong sulphuric acid in a porcelain cup, when if the solution becomes carmine, it is a sign of the presence of tellurium. The colour can be improved by adding a small piece of pure tin foil, heating and allowing to cool, in fact if the ore is oxidized, the addition of the tin is necessary. The colour fades in time but reappears on heating. Instead of at first using sulphuric acid, it is better to boil the powder with strong nitric acid until red fumes cease to be given off, then to add strong sulphuric acid, heat until white fumes appear and drop in a little piece of tin foil, the same carmine colour being produced by tellurium. These methods are apt to be unsatisfactory at times, but the more conclusive tests are in general more complicated.

Borates.—Borates, of commercial value as sources of borax and boric acid, may be overlooked, and yet to test these minerals for boric acid is extremely simple. Put about a teaspoonful of the powdered material in a saucer, moisten the powder with strong sulphuric acid and pour some alcohol over the mixture. Light the alcohol, and if boric acid be present in some quantity, the flame will have a deep green colour, especially on stirring. Silicates, like tourmaline and some other minerals which contain boron, do not respond to this test, but such are of no value for their boric acid.

Nitrates.—Nitrates burn vividly when heated on a piece of charcoal by directing the flame of a spirit lamp, or candle, upon the mineral with a blowpipe. If another portion of the suspected mineral be powdered, mixed with copper filings, and then heated in a test tube with strong sulphuric acid, the red fumes, characteristic of nitric acid, are given off, providing it be a nitrate.

Phosphates.—Warm the powdered mineral for a few minutes in a test tube with three or four times as much nitric acid, and when cool, pour a few drops of the solution into another test tube containing about two teaspoonfuls of cold ammonium molybdate solution. If phosphoric acid be present a yellow precipitate will soon appear. The ammonium molybdate solution is made by dissolving some of the substance in water and adding a little nitric acid."—Gordon Surr.—*The Mining World*, Aug. 14, 1909, p. 369-70. (C. B. S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

The Natural History of the Igneous Rocks. By Alfred Harker, M.A., F.R.S., Lecturer in Petrology in the University of Cambridge. 12s. 6d. (London: Methuen & Co.)

"Prof. Harker's reputation as a petrologist stands unrivalled in this country, and many geologists have regretted that his activities as author have lately been so restricted.

Until some sort of agreement is arrived at as to the condition of the earth's interior, petrology must always be to some extent a science without adequate foundation. Prof. Harker deals with this aspect of the subject with rare ability and philosophical grasp. Discussions as to the liquid or other condition of the earth's interior, he infers, cease to be pressing, since even if the former hypothesis be established, the evidences of volcanoes now active—some of which exhibit a difference of level of nearly 10,000 ft. although only twenty miles apart—forbid us, owing to the fundamental hydrostatic law, to suppose that the lavas emitted are drawn directly from a common source. The evidence forces us to the conclusion that the local reservoirs of individual volcanoes are supplied by drafts from some much larger body of rock magma with which it communicates, while in certain cases where evidences over a large area point to community of origin, 'we must conceive a very extensive intercrustal magma-basin, or a train of such basins, underlying the tract in question.'

Prof. Harker once more disposes of the error that volcanic action either exhausts or even dominates igneous action generally; its comparatively subsidiary importance as a geological agent ('intruded rock-bodies bulking far more largely in the composition of the outer crust than extruded material') being again noted. Summarily, igneous action may be divided into three phases: the volcanic phase, the plutonic phase, and the phase of minor intrusions such as dykes and sills. There is sufficient regularity about this sequence to give it a fundamental significance.

Adopting and enlarging Judd's conception of distinct petrographical provinces, Harker, in approaching the work of rock classifications proper, distinguishes between Atlantic and Pacific branches, the types of the former being richer in alkalies and the latter in lime and magnesia. The former is, moreover distinguished by a greater diversity of rocks. Concerning the serial relationship of the rocks in chronological order, there is great difference of opinion, and we are on more difficult ground. For the plutonic phase, at least, it is the case that decreasing basicity is sufficiently regular to constitute a law. With regard to minor intrusions the succession is more complicated, but a reverse order of decreasing acidity holds good.

The last section of the book deals with petrogenesis ('the study of igneous rocks as the products of the consolidation of molten magmas'), the chapters on 'the crystallisation of rock magmas' and 'the structures of igneous rocks' being sterling examples of the scientific treatment of difficult questions."—*London Mining Journal*, Oct. 23, 1909, p. 149 (A. R.)

Beneficio de Metales de Plata y Oro for Cianuracion. By Ferdinand McCann. Pp. 354, illustrated. $6\frac{1}{2} \times 9\frac{1}{2}$ in., cloth. $5. (Mexico City, 1909: Published by the Author.)

"Mr. McCann has produced in Spanish an excellent practical and scientific treatise on cyanidation that is worthy to rank with the standard books in English on the subject, both in matter and presentation. It is necessarily in part a compilation, but back of it is the author's extended experience, particularly in the Mexican field, and a wide and intelligent study of the actual methods and results in the leading camps. The main text of the book, devoted to the theory and practice of cyanidation, is clear, concise and complete, and also sufficiently scientific to put the book above the class of manuals for beginners of which there are already too many. The chapters given to the practice and results in the leading Mexican mines—El Oro, Dos Estrellas, Mines of Mexico, Esperanza, Guanajuato Consolidated, Real del Monte y Pachuca, Coscotitlan, Hacienda San Francisco at Pachuca and San Rafael—are of timely interest and great value, although from the progressive nature of the subject are somewhat transitory for a book. Other chapters deal with special problems of cyanidation and present useful tables. The book is well illustrated. The wide application of cyanidation in Spanish-American countries makes the publication of this book in Spanish particularly appropriate. It will serve the increasing number of Spanish-speaking engineers who are rapidly realising the urgent need of knowledge of the subject and also will be of interest to the English-speaking metallurgists who can read it and who will find it a practical lexicon for the technical phrases of the business."—*The Engineering and Mining Journal*, Oct. 9, 1909, p. 746. (W. A. C.)

Ashmead, E. Twenty-five Years of Mining. Cr. 8vo. *Mining Journal.* Net 3s. 6d.

Bickford, J. S. V. Faults and How to Find Them. Motor Car Failures and their Remedies fully Explained. Cr. 8vo. *Iliffe.* Net 2s. 6d.

Blitz, H. and W. Laboratory Methods of Inorganic Chemistry. 8vo. *Chapman & Hall.* Net 12s. 6d.

Cape of Good Hope. Department of Agriculture. 13th Annual Report of the Geological Commission, 1908. Engravings. 4to. sd. *W. Wesley.* Net 3s.

Chatley, Herbert. Stresses in Masonry. Cr. 8vo. pp. 154. *C. Griffin.* Net 3s. 6d.

Chamberlain, T. C., and Salisbury, R. D. Geology Shorter Course. Cr. 8vo., pp. 996. *J. Murray.* Net 21s.

Clark, D. Gold Refining. 8vo. *I. Pitman.* Net 12s. 6d.

Del Mar, Wm. A. Electric Power Conductors. Cr. 8vo., pp. 346. *Lockwood.* Net 9s.

Fischer, Emil. Introduction to the Preparation of Organic Compounds. Translated with the author's sanction from the new (eighth) German edition by R. V. Stanford. With 19 figures in the text. Cr. 8vo., pp. xix-175. *Williams and Norgate.* Net 4s.

Franklin, W. S., and Esty, W. Dynamo Laboratory Manual, Vol. I. Direct Current Studies and Texts. 8vo. *Macmillan.* Net 7s. 6d.

Gemmel, G. H. Chemical Notes and Equations: Inorganic and Organic. 2nd edition. Cr. 8vo., pp. 280. *Bailliere.* Net 5s.

Geology in the Field. Part 1. Edit. by H. W. Monckton and R. S. Herries. 8vo. sd. *Stanford.* Net 5s.

Goldschmidt, Rudolf. The Alternating Current Commutator Motor and the Leakage of Induction Motors. 8vo., pp. 220. *Electrician.* Net 6s. 6d.

Hawkins, C. C. Hawkins, and Wallis, F. The Dynamo: Its Theory, Design, and Manufacture. 2 Vols., 5th edition. Rewritten, revised and enlarged. 8vo., pp. 1,152. *Whittaker.* 21s.

Henderson, John. Electrotechnics, with Diagrams. Cr. 8vo., pp. 180. *Longmans.* 3s. 6d.

Hill, C. S. Concrete Inspection. 18 mo. *Spon.* Net 4s. 6d.

Hole, Walter. The Distribution of Gas. 2nd edition. 8vo., pp. 868. *J. Allen.* Net 15s.

Jago, William. A manual of Forensic Chemistry, dealing especially with Chemical Evidence, its Preparation and Abduction. Cr. 8vo., pp. 264. *Stevens and Hall.* Net 5s.

Jamieson, Andrew. A Text Book of Applied Mechanics and Mechanical Engineering. Vol. 4, 7th edition, thoroughly revised. Cr. 8vo., pp. 290. *C. Griffin.* 5s.

Johnson, J. P. The Ore Deposits of South Africa. Part 2. The Witwatersrand and Pilgrims' Rest Goldfields, and Similar Occurrances. 8vo. pp. 60. *C. Lockwood.* Net 5s.

Kahlenberg, L. Outlines of Chemistry. 8vo. *Macmillan.* Net 11s.

Low, David Allan. Applied Mechanics, embracing Strength and Elasticity of Materials, etc. 8vo., pp. 560. *Longmans.* Net 7s. 6d.

Mees, C. E. Kenneth. An Atlas of Absorption Spectra. 8vo. *Longmans.* Net 6s.

Oulton, L, and Wilson, N. J. Practical Testing of Electrical Machines. 12mo., pp. 218. *Whittaker.* Net 4s. 6d.

Peckham, S. F. Solid Bitumens. Roy. 8vo. *Spon.* Net 21s.

Peel, Robert. An Elementary Text Book of Coal Mining. 14th edition. 12mo., pp. 372. *Blackie.* 2s. 6d.

Penkin, W. H., and Lean, Bevan. An Introduction to Chemistry and Physics. 12mo. pp. 236. *Macmillan.* 3s. 6d.

Redmayne, R. A. S. Modern Practice in Mining. Vol. 2. The Sinking of Shafts. 8vo., pp. 292. *Longmans.* Net 7s. 6d.

Rogers, A. W. An Introduction to the Geology of Cape Colony. New edition. 8vo. *Longmans.* Net 9s.

Sheldon, Samuel, and others. Alternating and Current Machines; being the 2nd volume of Dynamo Electric Machinery. 8th edition. Completely rewritten. 8vo., pp. 366. *C. Lockwood.* Net 12s.

Shillito, F. W. Handbook of Corliss Steam Engines. 4th Edition. 8vo. *Spon.* Net 4s. 6d.

Smith, H. E. Strength of Material. 2nd edition. Cr. 8vo. *Chapman and Hall.* Net 5s.

Smith, J. Cruickshank. Oxide of Zinc, Its Nature, Properties, and Uses. Cr. 8vo., pp. 112. *Trade Paper Co.* Net 3s.

Stewart, A. W. Recent Advances in Physical and Inorganic Chemistry. 8vo. pp. 282. *Longmans.* Net 7s. 6d.

Steinmitz, C. P. General Lectures on Electrical Engineering. 3rd edition. 8vo. *C. Lockwood.* Net 8s. 6d.

Thorpe, Sir Edward. History of Chemistry. Vol. 1. (History of Science Series). 12mo. pp 148. *Watts.* Net 1s.

Thorpe, Sir Edward. History of Chemistry. (History of Science Series). Cr. 8vo., pp. 156. *Watts.* Net 1s.

Trautwine, J. C. Concrete. Reprinted from "Civil Engineer's Pocket Book." 12mo. *Chapman & Hall.* Net 8s. 6d.

Tutton, A. E. W. A Wave-Length Comparator for Standards of Length. An Instrument for Fine Measurements in Wave-Lengths of Light. 4to., sd. *Dulau.* Net 2s.

Van Nostrand's Chemical Annual, 1909. Edit. by J. Colesen. Cr. 8vo. *Constable.* Net 12s. 6.

Abstracts of Patent Applications.

(C.) 450/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in means for separating pulverised ore or other comminuted solid matter from liquid. 7.11.08.

This specification claims the combination of multiple settling surfaces and scrapers "substantially as described" for the separation of pulverised ore or other comminuted solid matter from liquid.

(C.) 109/09. George Hans Hirsch. Improvements in tube mill liners. 5.3.09.

This invention relates to tube mill liners consisting of channels formed of ribs of iron placed so as to cause automatic packing of a layer of the mill charge. These ribs may either be arranged as a single helix, as multiple helices, or as separate circles; in all cases they are held in position by specially designed locking bars, by wedges or by other suitable means.

(C.) 421/09. Henry Cawood Embleton. Improvements in or connected with pneumatic stamps for crushing ores and the like. 14.9.09.

1. This application covers a device for balancing the reciprocating parts of the stamp which are used for operating the stamp proper.

2. A device for revolving the stamp continuously, variably, and yielding, so that the stamp strikes the ore with a twisting motion.

3. A scheme for making up the length and weight lost by the wearing of the shoe.

4. A revolving feed drum with longitudinal grooves and various adjusting devices.

5 A Scotch yoke for doing away with an oscillating connecting rod with combinations of all of the above.

(C.) 505/09. Charles Allen. Slime separator and classifier. 6.11.09.

This application relates to a slime separator and classifier, the principal feature of which consists in automatic regulation of discharge. The apparatus consists essentially of a cone fitted with an annular buoy in floating suspension in a tank containing a liquid, the vertical rise and fall of which is dependent upon the proportion of solids to liquids contained in the cone; this vertical movement regulates the discharge opening which consists of a conical member, in the centre of which a fixed valve is placed, and also opens and closes the end of the discharge by its relation to a fixed valve plate.

(C.) 514/09. Harry August Stockman. Improvements in means or apparatus for disintegrating ores, minerals and other materials, and recovering the metal or other contents thereof. 12.11.09.

This application is in respect of a machine "for disintegrating ores, minerals and other materials and recovering the metal or other contents thereof.

The apparatus consists of a cylindrical pan having its axis inclined at 45° and carried at one end of a framed structure, the other end of which is carried on a fixed centre.

The end of the cylindrical pan travels on a circular path when the framing is moved around its fixed centre, and is constrained to revolve on its own axis.

The inventor proposes to use animal power to drive the apparatus, but points out that steam or other power may be used.

The material to be treated is shovelled into the pan as it revolves and a continuous stream of water is directed into the pan, the overflow passing off through a pipe in the centre and delivering on to a suitable table carried under the framing where the metal or other valuable contents are retained, the waste pulp passing off at the end of the table into a circular launder or trench which leads it away.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 593/09. Edward Emery Hardach. Improvements in stamp dies and means for fixing the same. 28.12.09.

(P.) 594/09. Alfred Berriman. Improvements in means for disposing of tailings and similar material on mine dumps and other depositing sites. 30.12.09.

(C.) 595/09. Percy St. George Kirke (1), Francis Sanders Morris (2). Improved method of and means or apparatus for producing and utilising a mixture of elastic fluids for motive power or other purposes. 31.12.09.

(C.) 597/09. Frederick Carl Siemens. Improvements in regenerative gas furnaces. 31.12.09.

(C.) 498/09. Dr. Carl Beindl. Improvements in and connected with the manufacture of hydrocyanic acid and cyanides. 31.12.09.

(C.) 599/09. Richard Scherl (1), Paul Frohlich (2). Improvements in or relating to gyroscopic mechanism. 31.12.09.

(C.) 1/10. Michael Woods (1), Thomas Jefferson Gilbert (2). Improvements in machines for dressing rails. 4.1.10.

(C.) 2/10. George Boulton Brown Elliott. An improved grating for mills for pulversing quartz and similar material. 4.1.10.

(P.) 3/10. William Edward Harding (1), Walter Bedford Thenyon (2). An improved chuck for rock drills and the like. 4.1.10.

(P.) 4/10. William Pill (1), Samuel Thomas (2). Improvements in and relating to linings for tube mills. 4.1.10.

(C.) 5/10. Arthur Reginald Angus. Improvements in and relating to devices for preventing collisions between railway trains "O.". 5.1.10.

(P.) 6/10. James White. Improvements in centrifugal pumps. 6.1.10.

(P.) 7.10. Edmund Paul Millington Jerrard. Improvements in and relating to mine signalling instruments and the like. 6.1.10.

(C.) 8/10. Arthur Eilert Drucker. Continuous Slimes vacuum filter. 7.1.10.

(P.) 9/10. Arthur Yeart Smith. An improved method for the construction of gates or gateways in fences and improved parts therefor. 12.1.10.

(P.) 10/10. Karl Hens. Portable ambulance dipping tank. 13.1.10.

(C.) 11/10. Joseph Storey Curtis. Demagnetisation apparatus for diamond drill rods. 13.1.10.

(P.) 13/10. Harold Arthur Tute. Improvements appertaining to the distribution of fluid pressure. 15.1.10.

(P.) 16/10. L. B. Woodworth (1), H. J. S. Heather (2), R. Curtis (3). An improved method of electrically altering the range of overload release devices on switch gear and the like. 21.1.10.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Anderson, A. G. *l/o* Germiston ; Rezende, Penhalonga, Rhodesia.

Atkinson, C. E., *l/o* Queque ; c/o C. L. Atkinson, 29, Tyrrel Street, Bradford, Yorks.

Barratt, R. L., *l/o* Bushtick Mine ; P. O. Box 101, Gwelo, Rhodesia.

Brown, G. D., *l/o* Barberton ; Burger Street, Krugersdorp.

Coaton, A. A., *to* P. O. Box 166, Fordsburg.

Cogle, C. T., New Kleinfontein Co., Ltd., Benoni.

Collings, B. I., *l/o* Exeter ; P. O. Box 80, Salisbury, Rhodesia.

Cooke, R. C. H., *to* P. O. Box 210, Salisbury, Rhodesia. Consulting metallurgist.

Hutton, C. E., *l/o* Benoni ; Fairwood, Orange Grove, Johannesburg.

Jordan, T. W., *to* City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.

Laret, H., *l/o* Zeerust ; Bushtick, Bulawayo, Rhodesia.

Melvill, G., *l/o* Luipaardsvlei ; c/o Geo. Baker, Watermeyer Street, Grahamstown, C.C.

Nicholls, R., *l/o* Bulawayo ; P. O. Box 2, Luipaardsvlei.

Patterson, P. F., *l/o* Penhalonga ; Golden Kopje Mine, per Private Bag, Salisburg, Rhodesia.

Piasecki, S., *l/o* Johannesburg ; Sub Nigel G. M. Co., Ltd., Nigel.

Reece, F. B., *l/o* Lead, South Dakota ; Socorro Mines, Mogollon, New Mexico, U.S.A.

Smith, S. A., *l/o* Benoni ; New Jagersfontein Mining and Exploration Co., Ltd., Jagersfontein, O.R.C.

Williams, G. W., *l/o* Sydney ; 90, Queen Street, Melbourne, Australia.

Yates, Prof. J., *to* Transvaal Mining Institute, P. O. Box 2824, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. FEBRUARY, 1910. No. 8.

Proceedings
AT
Ordinary General Meeting, February 26, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, February 26th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

56 Members: Messrs. W. R. Dowling, F. F. Alexander, E. H. Croghan, A. Richardson, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, A. Whitby, H. A. White, Prof. J. A. Wilkinson, R. G. Bevington, W. A. Caldecott, A. F. Crosse, E. L. Adams, H. A. Adams, J. F. Bagshaw, F. W. Bentley, W. Broom, J. M. Campbell, S. Caris, J. Chilton, F. W. Cindel, D. Coll, M. J. Doyle, A. D. Gillmore, B. J. Hastings, C. B. Hilliard, H. Howard, M. C. Inglis, T. Johnson, G. A. Lawson, H. Lea, J. Lea, W. D'A. Lloyd, C. E. Meyer, P. T. Morrisby, M. T. Murray, S. Newton, E. A. Osterloh, E. Pam, C. F. Parry, F. D. Phillips, E. T. Rand, W. H Roe, H. A. Searf, R. D. Scotland, G. H. Smith, B. T. Solly, S. H. Steels, R. Stokes, A. Thomas, J. R. Thurlow, J. F. Walker, F. Wells, and L. J. Wilmoth.

22 Associates and Students: Messrs. A. R. Adams, C. Y. Bruce, C. J. Crocker, G. J. V. Clarence, J. Cronin, C. L. Dewar, J. H. Hopkins, W. J. R. Hunter, E. G. Kenyon, A. King, E. Lee, C. Lipschitz, C. J. McCaffrey, G. Musson, W. S. New, H. B. Powter, F. J. Pooler, H. G. Roake, J. H. Smith, H. Stadler, A. Willcox and W. Waters.

20 Visitors (including Messrs. W. McDermott, H. G. Nicholls and E. C. Dyason), and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the January *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Newton and Roe were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Collins, Henry Stanley, Crown Mines, Ltd., P. O. Box 102, Fordsburg. Cyanider.
Hall, Charles Ernest, Crown Mines, Ltd., P. O. Box 355, Johannesburg. Cyanider.
Judge, Arthur Theophilus, Machavie G. M. Co., Ltd., Potchefstroom. Mining Engineer.
Middleton, Henry Victor, Crown Mines, Ltd., P. O. Box 102, Fordsburg. Cyanider.
Owen, Lionel Arthur, Crown Mines, Ltd., P. O. Box 1124, Johannesburg. Cyanider.
Short, Arthur Reginald, Crown Mines, Ltd., P. O. Box 36, Fordsburg. Cyanider. (*Transfer from Associate Roll.*)
Wood, John Egerton, A.M.Inst.C.E., A.R.S.M., Batavia, Java. Consulting Mining Engineer.
Wyly, Arthur James, Mijnbouw Maatschappij Siman, Lebong Tandai, Benkoelen, Sumatra. Metallurgist.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Chittenden, Edgar Darlington, P. O. Box 2235, Johannesburg. Claimholder.
Creagh, John, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.
Gillmore, William Duncan, Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg. Cyanider.
Nisbeth, Gosta Fredrik Arthur, P. O. Box 6221, Johannesburg. Mining Engineer.
Oliver, Edwin Letts, North Star Mines Co., Grass Valley, Cal., U.S.A. Metallurgical Engineer.
Summersgill, James William, Hlobane, Vryheid, Natal. Coke Manufacturer.
Sutherland, William Joseph Paterson, Dynamite Factory, Modderfontein. Chemist.

As Students.—

Gibb, Bertie, Sub-Nigel Ltd., Nigel. Cyanide Learner.
Taylor, Michael, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Cyanide Learner.
Thorne, Thomas Llewellyn, Aurora West United G. M. Co., Ltd., P. O. Box 97, Roodepoort. Cyanide Learner.

President's Return.

Mr. W. R. Dowling (*Vice-President*): I should like to take this opportunity of welcoming our President after a four months' holiday. He is looking very well and so must have been enjoying himself. I understand he has seen quite a number of our oversea members, and will be able to tell us about them.

The President: I thank you heartily for your welcome to me on my return, and can but say that it is a great pleasure to me to be back once more. In addition, I would take this opportunity of congratulating your Vice-Presidents on the excellent progress the Society has made during my holiday, and I feel that I am to be envied in having such capable leaders to help me through.

The most prominent feature of my holiday was the hearty welcome I received from mining and metallurgical men at home in my capacity as President of this Society It was a delight to me to find that amongst the most up-to-date in our profession, I needed no introduction, as our work here was well known and appreciated, and my only wish is that the friendship I received from them as your President, will be continued as an individual when my term of office expires. I met and was hospitably received by our corresponding Member of Council, Mr. G. T Holloway, who is, I am pleased to say, recovered from his regrettable accident; by one of our past presidents, Mr. W. R Feldtmann; by our honorary member, Mr. J. S. MacArthur, with both of whom it was a pleasure to resume past acquaintanceship; by Mr. T. A. Rickard, so well known in the journalistic mining world; and by the President and Council of the Institute of Mining and Metallurgy. From each and all of these I had many enquiries regarding *confrères* on these fields, and they all sent kind messages to our Society, and I can only encourage you all in our work by saying that the records of it, as embodied in our *Journal*, are highly appreciated and valued.

Coming back with these kindly feelings, I am lucky in being able to reciprocate in a slight degree to-night by welcoming here one of the past presidents of the Institute of Mining and Metallurgy, Mr. Walter McDermott.

As most of you are aware, this Society is intimately connected with the Institute of Mining and Metallurgy, and a cordial and friendly feeling exists, whilst papers of interest to both Societies' members are occasionally printed in the proceedings of each society. In addition the standardisation of screening and of weights and measures undertaken by the Institute of Mining and Metallurgy has been of very material use to our industry. Other fields are still open for efforts in similar directions and I feel that, in as much as we may view things somewhat differently from our friends in London, it would be to the advantage of the mining and metallurgical world if certain standards were adopted and endorsed by both societies.

Mr. McDermott was well-known to many of us by reputation a few weeks ago. He has now, I am pleased to say, cemented this in a more personal manner. His knowledge of working conditions has been considerably enhanced during his stay, and after viewing metallurgical and mining practice here, I feel sure he will have something interesting to tell us. It is the habit in some countries to waylay visitors on their arrival and get their impressions right away. We have been more charitable, or should I say lucky, as we have been able to reserve his remarks till the end of his visit. With your permission then I would call upon Mr. McDermott to tell us what he thinks of us, and as the traditions of this society are not against hard hitting, he need not fear if he criticises us in a hostile spirit.

Mr. Walter McDermott, who met with a most cordial reception, said: Mr. President and gentlemen, I am extremely touched by the kind reception you have given me, as indeed I have been by the reception I have met with throughout South Africa during my visit here. I should be well nigh overwhelmed if I did not recognise that the attention which has been paid to me, and the kindnesses I have received, are in a sense official and in my capacity as representing the Institution of Mining and Metallurgy, so that I do not have any false modesty in assuming that it is entirely personal, whilst at the same time I have had sufficient of that to give it a personal flavour. What your President has said is perhaps a little without justification as to what I can say to you. I have been caught quite young in the country, and there is nothing really that I can say except as to general impressions of the conflicting conditions which exist, and of the many unsettled questions regarding which it would be very presumptuous were I to attempt to give any conclusions of my own. I am still with an absolutely open mind on a number of points in which I have taken great interest, but about which I can truly say nothing but what I have heard, but to judge between them or say anything as to the actual tendency of the conditions here is quite impossible for me, even after a few weeks residence. I have seen a good deal, but not half as much as I should have done had it not been for my occupation and travels to different parts. I should like to have seen more of the typical mining plants than I have. Of course I have read a great deal and thus have a general idea of what questions are still open. I may say in connection with that, that I do find there is still this trouble of the diversity of opinion as to what is going to be the treatment in the future. I say nothing of mining as I have only been underground twice, and can therefore express no opinion. But I have been struck by the attention which is being devoted, not only to the details of crushing, but also to the treatment. I find that some of you are still occupied with the question of the continuous treatment, without classification, of the tails, that is the treatment of sand and slime together. This is nothing new. It has long occupied the thoughts of all of you but it has not been practically followed out owing to the difficulties of a continuous working process. Everyone recognises the advantages of the continuous working process, and if continuous settling could be combined with the continual operation of slime and sand together, there would be advantages connected with it. The question of the settling of slime itself almost necessarily involves further consideration. I find many interesting facts are abroad at the present time as to what is being done in this direction by more than one metallurgist on the field. Of course people abroad have sometimes unduly criticised the decantation system of slime settlement, very often because they have not seen it. Anyone who has seen what has been done with the decantation system gets his eyes opened to new facts, and its success here must be admitted in other parts of the world. Therefore any conclusions arrived at abroad would require modification so as to give expression as to what has been done here. Whether it is the ore or the skill of the metallurgist, certainly you produce effective settling and decantation, and the reduction of values in the solution and residues is accomplished. Therefore any changes from that will be in the nature of reforms coming from the universal effort made to reduce cost, since there is no more chance of increasing extraction. I do not want to go into particulars as to what investigations are being made. But any modification of a continuous system of settling will take time to develop and it is a matter on which I cannot begin to express an opinion. I only introduced it here as being one of the points which struck me as being interesting to us, and to which we and others connected with the Institution of Mining and Metallurgy look forward to information from your transactions and from actual work here with the greatest interest.

Your President has referred to the relations between the two Societies and I am pleased to corroborate his wishes on behalf of the Institution of Mining and Metallurgy, that these relations shall continue to be of a most intimate character, and be extended in any direction in which they can be made workable. One direction has been indicated to you by your President in his remarks to you just now, that is, that we should co-operate in establishing a uniformity of terms throughout the world. We have established a few standards and we think they have been successful, for at least they have brought uniformity of thought amongst men and the intercommunication of ideas. I should like to say that the basis on which we have worked on the other side is one that probably you are soon also to adopt in anything that relates to the uniformity of practice amongst the members of our profession all over the world. You enjoy advantages which we do not on the other side. Our membership is more scattered. We do not get collected together, as you do in this room, those men whose interests are largely in common as regards their daily occupations. I think there is hardly a subject that anyone of you may bring up for which you will not immediately get a very large and sympathetic attention. Whereas with us we may have papers read at a very large meeting in which there is comparatively small interest. At the same time the profession is wider than in Johannesburg, and the subjects of which we treat are perhaps more far reaching, in some respects, if less interesting. Therefore we find it necessary not to attempt from London to lay down any laws or even suggestions until we have first collected the opinions of engineers outside London. I should like to suggest therefore that in case we should co-operate in this movement as to uniformity of terms that you should be satisfied with our method, and first of all collect opinions by questions addressed to residents abroad, and not only amongst yourselves. When these matters have been submitted to engineers all over the world, and they have had an opportunity of expressing themselves or suggesting terms, then you will really feel that you have the force of the profession behind you in a way you would not have if you simply adopted a certain system of your own in Johannesburg, and I am sure it will meet with the approval of all active members of the Institution of Mining and Metallurgy. Therefore if you have suggested any collection of terms—and there was a proposal on our side to attempt to introduce such a thing though we have taken no steps yet—you being the first in the field, if you can suggest something of that nature for our consideration, and will join with us in circularising engineers all over the world, we should heartily co-operate with you.

Your President has suggested that I should say something on screens. There has been some misunderstanding about these screens. We attach no undue importance to their standardisation, and there was no common intention to have a screen used in any sort of way commercially. We found dissimilarity in practice, and when collecting opinions all over the world, we found there was a general consensus of opinion as to the adoption of a standard screen which would be the means of translation into the common language of all engineers. At the time we were obtaining opinions we received an immense number of suggestions, suggestions as to the scale on which we should grade them, the number of screens, the wire to be used as a standard, and we had every opportunity of collecting evidence from all over the world as to what appeared to be the most generally held opinion concerning the most useful standards. We placed them in arithmetical progression, and we then found out that we could have them manufactured at something like a uniform standard. But as you know commercial screens are very irregular. The one half of a screen may be slightly different to the other half of a screen, and it is impossible to purchase screens with any real regularity. We felt there was no possibility of converting a screen into a scientific instrument. In the webbing there is always a marked difference even with the greatest care. There is the difference in the use by the individual, the length of time the sample is rubbed through the screen, and the length of time the screen has been in use. It is only an instrument which can be used in a laboratory for rapid and approximate work, and to attempt to introduce reforms by the use of micrometers and going entirely on apertures is, in my opinion, slightly misleading in practice. Every man does not use micrometers, and the screen is not a scientific instrument at the end when you have done it all. So there is no reason in saying this is a standard which must be used in practice, but it is a means of interpreting into language certain results of working. There is no reason in giving us a standard mesh. Every man who works in mesh will know exactly what it means. We have tried to establish something which every metallurgist can obtain and every man can use in a laboratory. To show how difficult the question of screening is, I may say, it has taken months of time and has cost manufacturers a great deal of money to produce, even on a small scale, the number of feet necessary to supply laboratories with a real well sized screen. We have succeeded in that and have these to 120 mesh. It is true that some of the larger mesh are not absolute, but they carry out the progression and are useful for purposes of comparison. The Institution of Mining and Metallurgy screening is not used commercially, for you would never think of using such a thickness of wire commercially, but it carries out the system necessary to finer mesh. I may say in connection with this that what has been done by the Mines Trials Committee here is of very great interest to all the world, and the desire of the Committee to express the equivalent in a standard mesh will be very much followed abroad, but there is not the slightest desire that such a standard should be used by investigators where they have good reason to adopt another standard.

I think, gentlemen, that I have touched on all the points I wished to speak about and I thank you heartily for what you have said both for myself and for the Institution of Mining and Metallurgy.

The President: I am sure we appreciate very much what Mr. McDermott has said, and in parting with him I can only say we all wish him a very pleasant and safe voyage back to England.

I also wish to welcome in your name two visitors we have with us to-night from other parts of the world—Mr. H. G. Nicholls, with whom many of you will connect a certain slimes settler, and whose reputation has extended well over the United States; and Mr. E. C. Dyason, who is on a visit from Melbourne.

Mr. H. G. Nicholls (*Visitor*): With regard to impressions, there is one I can offer unhesitatingly, and that is the universal kindness I have experienced since I came here. I have been made to feel so much at home that I have half ceased to consider myself a visitor. In connection with slimes I hope I may be able, at some future date, to meet you again.

Mr. E. C. Dyason (*Visitor*): I am very glad indeed to be able to attend a meeting of this Society, and to pay a visit to this, the Mecca of gold mining engineers. I hope that after I have worshipped at the shrine I shall go back to Australia very much edified.

Prof. G. H. Stanley (*Member of Council*) here exhibited two laboratory sieves which had been introduced in accordance with the standards recommended by the Institution of Mining and Metallurgy (with apertures ·006 in. and ·01 in.), combined with an arrangement which enabled the screens to be replaced when worn.

CONGRATULATIONS FOR MR. CALDECOTT.

Prof. J. A. Wilkinson (*Member of Council*): I think the congratulations of the Society are due to Mr. W. A. Caldecott, one of our Past Presidents, who has just been awarded the degree of Dr. of Science by his *alma mater*, the Cape University.

Mr. W. A. Caldecott expressed his thanks for the kind reference.

FURTHER NOTES ON RAND MINING.

By Tom Johnson (Member).

It is well known that several mines have trouble with the roof of working places, and I thought a few remarks on how to deal with some of the problems would not be out of place, hence these notes. No doubt, much that I have to say to-night will be old to some of you, but from what I can see and hear, will bear bringing forward again. In some cases the roof shows weight over large areas, crushing the pillars in the stopes, the drive ribs, and sometimes even the large safety pillars of the shafts.

Personally, I like to see incline shafts and the drives under the reefs out of danger, but that will not help us with the present shafts and drives that are giving trouble now.

For the first case we will take an assumed one of where a large area is open, the drive ribs are getting crushed and also the stope pillars, causing the roof to break up. Around the drive it will look something like Fig. I. Here, in the lower figure, you will see that the ribs, although broken, have offered so great a resistance as to break the roof, leaving it ragged and dangerous. Now before things had reached this state the ribs should have been taken out and waste packs or sand packing put in as a substitute. This would steady the roof without breaking it up over the drive, thereby reducing the danger and trouble. The taking out of the ribs allows the elasticity of the roof to come into play so that it bends instead of breaks. In the upper figure the ribs have not been strong enough to bear the roof. If unfortunately the drives have got into the condition as shown in Fig. I (lower) (which many underground men are getting familiar with), then take out the ribs, rip down the loose rock, and pack. This is much better than trying to hold the drives with props or sets of timber, as timber, with a moving roof, would require renewing fairly soon.

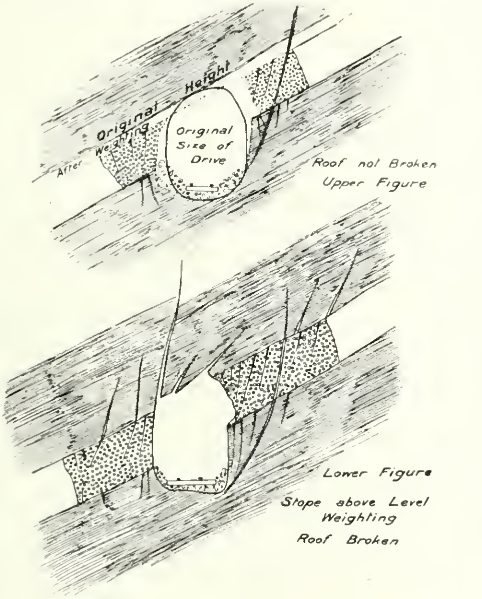


Fig. I.

Packwalls and Pigsties.—Packwalls should, like props, be built at nearly right angles to the plane of the reef and not vertical as so many of them are. If built as shown in Fig. II., lower figure, there is a liability (especially with a small pack 12 or 14 ft. square), for the whole to get squeezed "upbrow"; if a longer pack, the top of the lower wall will get forced inside the pack and bulge out the middle part of the wall, which would eventually fall, and the pack would soon be no better than a loose heap of waste. If built as in Fig. II., upper figure, the pack takes the weight more directly, and would not slide away so easily. I have been told many times that the walls would not stand if leaning over to the dip, but as I have seen miles of walling put in on the "rise" side of levels you will realise I am inclined to hold the opinion that they will stand; it is all in the manner of building them. In Fig. II., upper figure, props are fixed, with or without a lagging board here and there behind, helping to steady the walling until the weight comes on; these props need not be heavy. Note in upper figure that the walls are the height of the width of stope, but in the lower figure the walls are higher than the width of stope, a source of weakness. The small waste should be thrown on the walls as they are being built course by course and sifted in amongst the boulders to form a bedding, thus giving solidity to the walls.

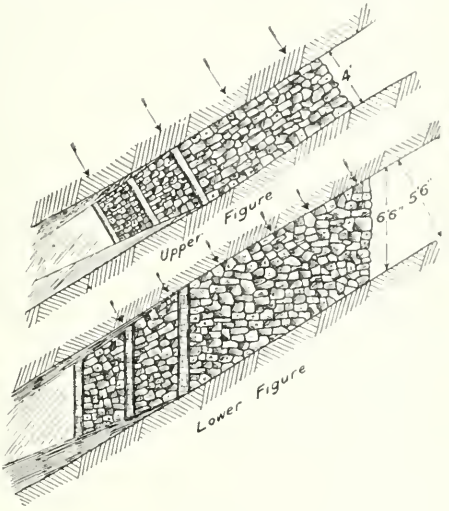
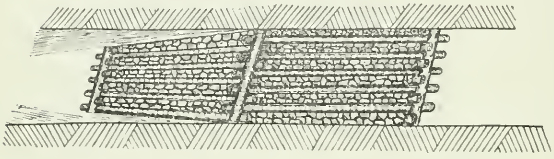


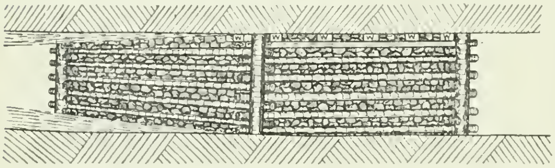
Fig. II.

It is a debatable point how a wedge shaped piece of rock should be put in a dry wall. There are some who think it should be put in with the thin edge outwards, as they think it has a better chance of not being crushed out, but personally I would sooner put it the other way about, packing over and under to level up.

It is a good plan to bind the walls to the inside of the pack with pieces of old pipes, rails, rope, boards, etc., particularly at the corners. Pigsties, or birdcages, are very good in places, but it is wonderful the number one sees that are practically of no use. It seems as if we follow fashions in the mines, as we do with our clothes, without making sure that we are doing the most suitable thing. Many pigsties can be seen that are practically useless except as receptacles for waste, as little thought is given to their proper use. A lot of trouble and expense is incurred in getting timber down the mine and into the working place. Then the whole thing is jerry built, like Fig. III., upper figure. Here the four corner posts are put up and the crib built in what appears to be the orthodox manner, but it is just here that failure arises from want of a little reasoning as to the "why" of things. If the corner props were taken out the pack would fall to pieces, as a study of Fig. III., lower figure, will show—apparently the same thing, but there is this slight difference, stones are built in between the different layers of sticks and weight is put on to the pigsty by wedging; since putting stones in between the layers of sticks prevents the sticks sinking into each other as much as they would do otherwise, and the pigsty will not give way so fast. A pigsty built in this manner, if free from the blasting, should have the props at the corners taken away as soon as it is wedged, as the props will take all the weight till broken. It cannot be too strongly impressed on men and bosses that pigsties have a distinct use and place in the scheme of mining and are not to be used just because someone else uses them. They should be used where a pack is wanted, and where there is not coarse rock enough to build the walls of a pack, or where a pack would be destroyed by the blasting: at the corners of very large packs, where roads have to be kept open a long time, and in places where props would soon be broken, or break up the roof, advantage should be taken of their area of support, and thus they must be well wedged as shown in Fig. III., lower figure. In the upper figure there is only a line contact between each row of sticks, which allows too much "give" with little weight.



Wrong Way Upper Figure



W = Wedges Right Way Lower Figure

Fig. III.

Speaking of supporting broken roof naturally brings to mind the question: "Is there no way of preventing the roof breaking, and can we get away from the effects of roof breaking"?

Now it is well known to colliery men that in dealing with a "tender" roof, or in other words one that breaks easily, it is of the greatest advantage to move the face on at the greatest speed, so as to keep exposing fresh roof and not give the roof time to break, and the same principle applies here. If we used eight or ten machines on a face we should not have so much worry with the roof in stopes, the time factor being so very important in this matter. In my previous paper "Notes on Mining," I advocated a larger number of machines per face and I am now more firmly of the opinion that this has to come.

Props.—One sees some strange prop setting in these mines. Sometimes a fine hitch is prepared for the foot of a prop and the head not half set, leaving a gap looking at you as at (*a*) Fig. IV. Then there seems little idea as to the proper angle a prop should make with the roof and floor. In horizontal places props should be vertical, whilst in inclined places the props should be nearly perpendicular to the roof and floor. The greater the inclination of the stope, the more the prop may be from the perpendicular. The amount varying from 0° for low dips, up to 10° as a maximum, say for a dip of 60°. See Fig. IV.

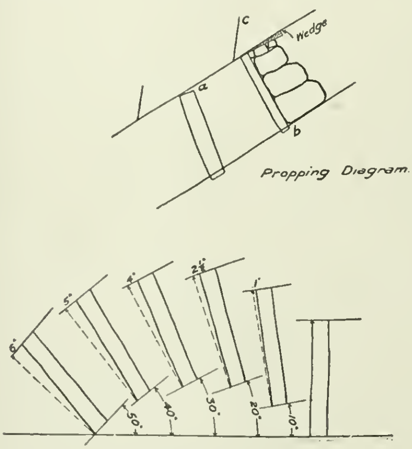


Fig. IV.

Where props have to stand the blasting they should be of good diameter, say 10 in. and upwards, and sometimes, when it is very troublesome to get heavy thick props into a stope, bunches of three or four lighter props can be put in and bound together with wire or hoop iron. In low places two or three pieces of rock built up as shown in (*b*) Fig. IV. and well wedged serve the purpose of a prop splendidly. This is very cheap, while it is not always necessary to use a thin prop as shown. In many places concrete props built in place would prove very useful. In setting props the roof should be examined for slips—the extended use of acetylene lamps being very useful in this direction. The prop should be set on the "upthrow" side of the slip as shown at (*c*) Fig. V., so as to catch the lower portion and also hold the upper portion. The angle of prop setting shown in Fig. IV. is for an unbroken roof, or for the lower side of a broken roof. On the top side of a broken roof the angle would be different because of the tendency of the roof to slide downhill, the idea throughout being to set the supports against the direction of movement.

Pillars.—Pillars in stopes should never have a fault slip passing through them except in the case of a vertical slip striking with the dip. With inclined slips and vertical strike slips, the pillar should be left under the slip as it runs in the roof as shown at (*a*) Fig. V. Also when two reefs are being worked close together, care should be taken to get the pillars in the top seam perpendicularly over the pillars in the lower reef. A look at most mine plans will show you something like Fig. V. Here we have the two reefs fairly close together, with drives on both the South and perhaps the Main Reef Leader or Main Reef, the reefs being worked out as shown. Now the whole of the reefs may as well have been worked out so far as the value as supports are concerned. The lower reef ought to have had all the pillar left on the lower side of the drive and the top reef all left on the top side of its drive; another way would have been to stope on the lower side of the lower drive down to the point (*c*) then leave the pillar under the ribs of the top drive.

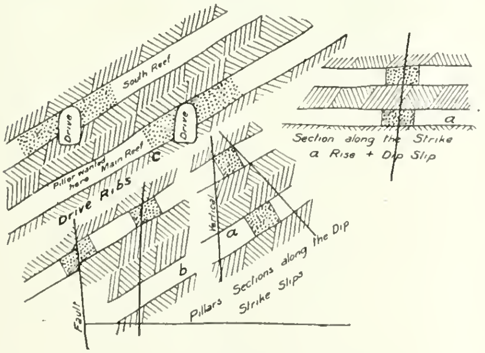


Fig. V.

Crushing Incline Shafts.—Where the shaft reef pillars are being crushed in a timbered shaft, help the pillars by building large packs on the stope side of the pillars, using sand packing as it gives least; also weaken the post timbers in the shaft by tapering them at the foot, which will allow them and the roof to settle gently. If there is not room for settling, then there is no help but to make room either in the floor or roof or by using a lower skip. In some cases the roof plates could be cut out between dividers; in other cases sets could be taken out without increasing the danger. If there is plenty of room *i.e.* height in the shaft, the pillars could be stoped out and the place packed with water-borne sand, the stopes behind the pillars being packed first. It may appear foolish at first sight for anyone to suggest taking out shaft pillars to save the shaft, but it must be understood that when the small pillars have gone in the stopes, the shaft pillars have a much greater weight to carry. Now if the stopes are packed and the pillars left in, they are still the strongest and have to carry the weight until some settlement takes place, and during this time the pillars get crushed and in some cases the roof also gets broken, while if the pillars are taken out and the space packed, the roof can settle bodily without breaking up. In the case of an incline shaft, where stopes have been worked underneath and are caving, these stopes should at once be packed close, preferably with water-borne sand, and for a good distance on either side of the line of the shaft. The same thing applies to the bottom of vertical shafts that are crushing because of insufficient area of pillars; and whether caving has commenced or not the water borne sand is best, for it will flow into places where other packing cannot be put In the case of vertical shafts there may be trouble with the timber should the ground be breaking. In cases of this kind the sets below the break should be strung together by some means, so that they support one another, and hung to good ground above. The sets immediately above the break also need hanging from some point higher up the shaft, the studdles between the two lots of timber taken out and provision made in the guides for the movement that is taking place. Someone might say crushing happens quickly and before it could be fixed up to handle water-borne sand, but I think not. However, if the danger be not appreciated or understood, or one wished to keep down the working costs so that the trouble became imminent, there are air pipes or water pipes in every working shaft and I should make use of these in getting over the trouble. An objection might be offered that it would be going rather far to take either the air column or pump column for the purpose of passing sand down, but I am one of those who hold that a great many things are permissible before going so far as to shut down.

In cases of incline shafts that have bad roof in places and are heavy on timber, it would probably be cheapest to use concrete arches over each compartment instead of timber.

Machine Air Cost.—There has been quite an amount of talk about machine air costs, and it seems that some people are as mad about this so-called economy as they are about running three or more machines per man. They do not bother about the half-crown they lose if they can gain threepence in the air costs. To me it appears that the machine that drills a few feet more per shift is the one to be after, even if air costs are 20% higher. For instance air costs per machine shift are, say, 6s. ; then say 30% is leakage, etc., leaving 70% against the machine itself. This equals 4s. 3d. The machine drills 24 ft. per shift and breaks $\frac{2}{3}$ of a fathom of ground. Now suppose we change the machine and use one that takes 50% more air, making the cost 6s. 4d. against the machine itself. Leakage will be the same, and how much more work must the new machine do to pay for its extra cost. To get at this we must know the total cost of breaking ground per fathom. This we will take at say 70s. ; even with the three machines per man, 70s. equals 46s. 8d. per shift, equals 1s. 11·3d. per foot, so to pay for the 50% increase of cost of air for the machine we only need another foot of drilling from the machine, despite all the worry about getting a machine to save air. I think looking at the air costs alone is wrong if we really wish to lower the machine costs. The same reasoning applies to whatever machines a mine may be using. Do not worry particularly about the cost of air per machine shift, but look to the cost per fathom, and strive to get the best from the air. Do not let your machines get into the rattletrap stage, but spend money on keeping them in order. If one machine fitter has a hard job to keep the machines in fair running order, have two and keep the machines in good order, the total cost being the point to watch. Let any individual item rise if by so doing the total is lowered. We have at the present time some very good drilling machines on the Rand, but we do not get the work out of them that they are capable of doing. It is well known that there is a very great difference between the lowest and highest cost per fathom on any mine. I figure it that some men can do fairly well on three machines and others cannot. Why then give the poor man three machines? Give him the same price per fathom and let him try with fewer machines. I have often seen men make as much money stoping with one machine as two, the same price per fathom being paid.

The moral is get a machine that can drill, and a man at the back of it to see that it does drill. Then machine costs will come down even if air costs are higher. Some use the argument that if a machine uses less air we can use more machines with the same compressor plant, but I say, get the same footage from a smaller number of machines even if each takes more air. Money is saved, because there are less machines to buy and less machinemen required, leaving us the chance of keeping out some of the men who have not yet learned their business.

Breast Driving.—For those who wish to drive on the reef a modification of a coal mining method may be resorted to, which will help to solve the difficulty of broken roof in drives. In this method (see Fig. VI.) the drive would be breasted in 25 ft. to 30 ft. wide, putting in packs on one or both sides of the track. Ordinarily, I think, the top side would suit us best (if the lower side ground was to be left in for a long time after the top side was worked, then a pack should be put on a lower side also), the track or tracks are laid on the lower side. The pack, made from the roof waste, would form a brattice behind which the return air would make its way to the last opening to the level above. If enough waste was not produced in the drive to form a decent pack, waste from other work could be used, say,

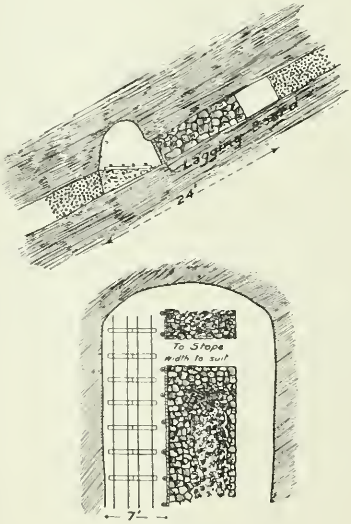


Fig. VI.

from cross-cuts, ore passes, etc., for there is nearly always waste to be found that could be used instead of sending it to the mill. This drive, as can be seen by reference to Fig. VI., is breasted in on the reef; about nine machines would be in use and upwards of 500 ft. per month advance could be made. Very good arrangements for handling the broken rock would have to be made with a reef thickness of 3 ft., as 6 tons per foot of advance would require moving, besides the waste for packing.

These drives should be run on lines, the direction of which are determined from some other drive already driven. In case of a fault throwing the reef out, the drive would be narrowed down to a sufficient width for the tracks and a fair sized ventilating pipe. The stopes should be started at the back of the pack, no rib being left. This method of driving would be useful in a developing mine, if in *good ground*, for getting a mill going sooner. Development rock, as a rule, is not fit for milling alone, due to its poor value, but if done as sketched out above, the rock would be of as good value as the stope rock. Seven or eight faces would keep 50 stamps going. Also, as we must expect greater trouble with the roof in the deeper levels, we will have to consider the advisability of driving to the boundary and stoping home, as in this way we should escape some of the roof trouble. In conjunction with sand packing it would be an ideal scheme. I know the time and cost factors have been against this being done, but if we could get faster driving, say, 500 ft. per month at the lowest, the time and cost factors would be so reduced as to make this system practicable.

Six months would see the drive 3,000 ft. in, which is farther than most drives have gone. In the old days, when it took from two to three years to get to the boundary, one could not wait all that time before commencing to stope; but with the higher rate of driving we have, and which will be improved upon, it is quite feasible to get out to the boundary before stoping commences. As we should only have four boundary faces of 250 ft. on each level, giving, say, 400 tons from a stoping width of 4 ft., this would not be sufficient if we wished to keep to a reasonable number of levels, say, three or four. I should therefore set out the winzes as near as practicable 700 ft. to 800 ft. apart, and work the blocks outwards and inwards, taking about twelve months to work out a block; this would give eight faces with about 800 tons of rock, single shift, per level, causing us to use coal mining method of transport, the bugbear of gold miners. A way in which we could stope without pillars or sand packing, is to break the floor for rock to make ordinary packs, this would be cheaper than pillars.

Driving under the Reefs.—If drives are put in under the reefs and cross-cuts driven to the reefs, there is little fear of trouble on the drive: the drives would only need to be single with an occasional by-pass, and the cross-cuts would form the gathering stations No driving on the reef would be done, and a track would follow the stope face along the bottom, on which the rock broken from the bottom lift of the stope would be handled. A steep boxhole would be put up from the side of the cross-cut through to the top reef; this would cut the stope at some distance from the bottom and would form the boxhole for the first cross-tracks and the brake roads in the stope. Winzes would be sunk on the reef, one over the other, and the cross-cuts would be driven to intersect these winzes, so forming the points of attack of the stopes. This system is, of course, for mines where the cross-cutting and facilities of working would prove cheaper than driving on each reef. The boxholes could be put up from the top side of the drive and at various angles so as to split up the stopes as required. The drive should be widened to form a shunt, "inbye" and "outbye" of the box.

Drill Sharpening Furnaces Underground are not so much used as they should be considering how cheap they are and how easy and comfortable they make the handling and sharpening of drills. Of course, it is no use putting a furnace underground and then bringing the drills in the skip to the level on which the furnace is, as when drills are once put in the skip they may as well go to the surface. To arrange things properly a furnace should be on each level and the drill sharpeners go to each; that is, they travel to the drills instead of the drills travelling to them. Not only does this save time and handling, but it brings the sharpener and user into much closer contact, which is a most desirable thing. It also keeps down capital expense of steel, thus providing the capital for the furnaces. Although the men do not and must not be expected to do as many drills underground as on the surface, my experience is that it is still cheaper, for, as mentioned above, less steel is wanted, whilst the machine men are much better served, and things go more smoothly.

Shaking Chutes.—Many of these very handy and useful ore conveyors are in use, but are not being fully taken advantage of. I think it was about nine years ago that I first used these hung from a rope. This is much easier and better than hanging every length from separate pegs in the roof. When hung from separate pegs, some are nearly sure to be out a little in distance, whereas when hung from a rope they may be adjusted to a nicety, and are much easier to change to some other point. The manner of doing the job is to first stretch a rope along the roof, supporting it from pegs in the roof where it sags too much, which may be 30 ft. or more apart, being nearest in narrow stopes. On the rope are fastened clips the exact distance apart of the lengths of chutes, hole to hole; the chain swingers are hooked at the back or top side of the clips, and the rope is tightened by a right and left handed screwed tightener before the shutes are hung on it, Fig. VII. At the end of the shift, when the shutes need shifting prior to blasting, they are unhooked from the rope and taken away, and in most cases the rope may be left as it is, taut against the roof.

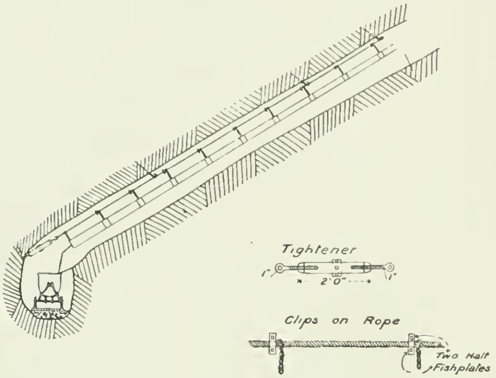


Fig. VII.

In case of shifting to another spot closer to the face it is only necessary to drill a few holes in line without reference to distance apart. The chutes are unhung, the rope slackened, and the top end taken to the new point and made fast. The rope is again tightened and the chutes rehung, the clips on the rope serving again to space them properly without any alteration. No labour is necessary except the shovelling boss and his boys. Chutes should deliver right into the box or into the car, not on to the floor of the stope and then boys employed to shovel the rock into the box, as one notices being often done; the boxes should be steep enough to allow the rock to just about run, as if too steep they give trouble to the trammers by spilling, and if too flat they also give trouble; an ordinary width stope box 6 ft. to 8 ft. long will cost about 2d. per ton to get the rock through; these remarks apply to ordinary stope boxes. Where more or less permanent boxes are put in, they should have radial doors if possible, as then the steepness of a box and box-hole does not matter.

Mr. A. Richardson (*Member of Council*): Our thanks are due to Mr. Tom Johnson for his suggestive sequel. It is not often that a practical mining man comes forward to give us the "worm's-eye" view of underground conditions instead of the bird's-eye view with which most people are acquainted, and therefore I hope he will give something more to ponder over when his powers of reply have been stimulated by plenty of adverse criticism.

Mr. C. F. Parry (*Member*): It gives me great pleasure to second the vote of thanks. Many mines on the Rand are having a great deal of trouble at present, and when they get deeper the trouble is going to be increased—the trouble of keeping the hanging wall where it ought to be. For that reason I think Mr. Johnson's paper is most opportune, and I hope it will lead to considerable discussion.

THE TUBE-MILL CIRCUIT AND CLASSIFICATION.

By G. O. Smart (Member of Council).

In view of the increased attention which has of late been paid to classification for tube-milling purposes, it may be of interest to members of this Society, to lay before them the results of work which has been carried out during the past year. It is now generally admitted that for the most efficient tube-milling, it is necessary to maintain constant conditions as regards tonnage and moisture in the pulp entering the tube-mill. The following remarks are intended to point out how such a desirable uniformity of feed can better be secured, than by the classifiers in common use on these fields.

Before discussing the details of classification it may be desirable to illustrate and briefly refer, to the general arrangement of the tube-mill circuit, and classification system of which the principle was introduced by Mr. J. R. Williams at the Glen Deep, in the early days of tube-milling on the Rand, although it is by no means universal in other mining countries. The following figure illustrates schematically the principle of the tube-mill circuit as now in use on some of the Gold Fields companies.

With this system only those particles in the tailing pulp can overflow the classifier and pass away to the cyanide plant, which are not sufficiently heavy to fall through the classifier and so pass into the tube-mill. The size of the particles overflowing the classifier naturally depends upon the coarseness and volume of the pulp entering the classifier, or in other words, upon the relation between the tonnage of ore milled, and the total capacity of the crushing plant in stamps and tube mills. Since the classifier determines, subject to the foregoing limitations, the size of the particles overflowing it, an important difference may be noted between such a determining factor and that of the battery screen, the pulp leaving which, has a size of its maximum particle determined by the diameter of the screen aperture, and not by the combined effect of diameter of particle and its specific gravity, as is the case in hydraulic classifiers. In the latter case the particles of richer and specifically heavier pyrites, are reduced to a finer state of division before leaving the crushing plant, than the poorer and lighter silicious grains, and consequently that portion of the ore which best repays regrinding is automatically reduced finer than the less valuable portion. In order to obtain full efficiency from tube-milling, it is necessary that the tube-mill outflow should contain a fairly large proportion of coarse grains, such coarse grains being elevated to return again into the tube-mill, while the portion which has been sufficiently finely comminuted by its first passage through the tube-mill, overflows the classifier. Another advantage of the tube-mill circuit, is that, although a low ratio, say, 2 of water to 1 of sand, is sufficient for the purpose of amalgamation on the shaking tables, yet this thick pulp is diluted with much thinner pulp from the battery, and the resultant mixture consequently is more easily pumped and classified than would be the case with the tube-mill returns, or mill pulp separately. The tube-mill circuit likewise serves as a most efficient trap for "floured" mercury and amalgam, escaping from either the battery plates or the shaking tables. Any particles of such material other than of impalpable fineness sink down through the tube-mill classifiers, and lodge upon the tube-mill shaking tables. Should this not occur during its first passage such a particle merely returns again until it is caught. It may be incidentally observed that particles of metallic iron likewise drop down and accumulate in the tube-mill circuit, and such would no doubt also be the case with metallic gold, if amalgamated plates were not installed after the tube-mills. A magnetic separator which would readily and efficiently remove such accumulation of metallic iron from the tube-mill circuit, would be gladly welcomed, since at present no such means is known for removing this material, as readily as metallic gold can be removed by amalgamated plates.

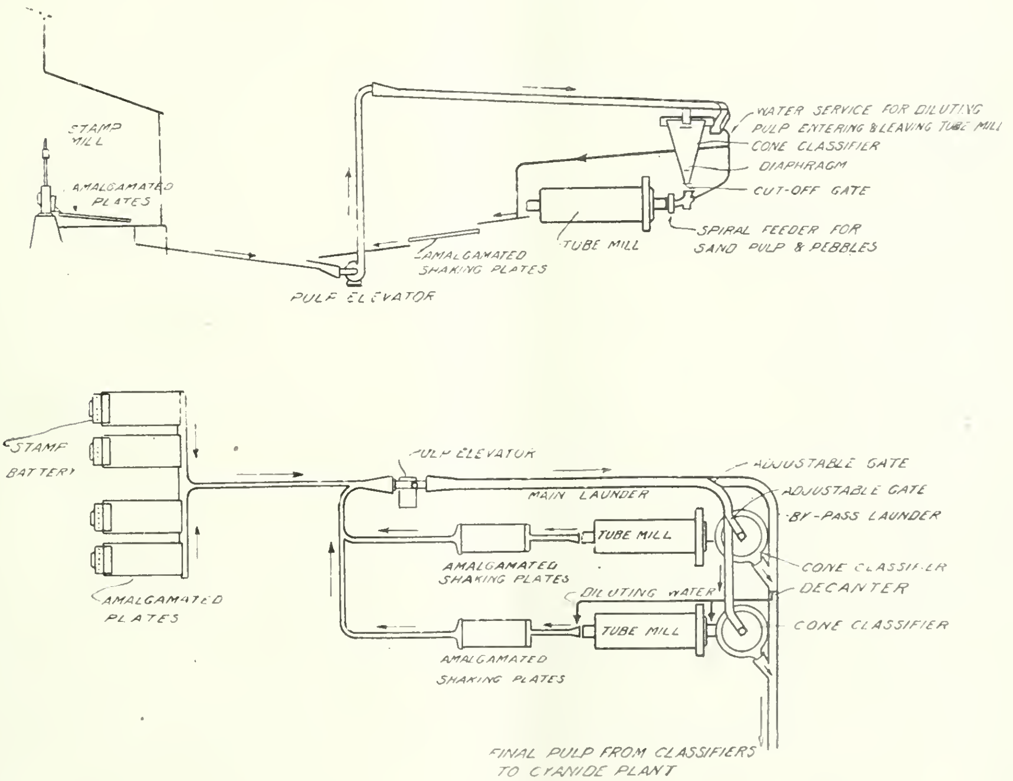


Fig. I.

Tube Mill Circuit, with Single Cone Classifier and Diaphragm for each Tube Mill.

It is advisable in the design of a metallurgical plant that the appliances for any given purpose shall be as few and as simple in their operation as possible, and concentrated at one point, rather than scattered throughout the plant. In accordance with this requirement all classifiers should be placed together, instead of being distributed throughout the reduction works at various points. The flat nature of most mill sites on the Rand, necessitates elevation of the pulp leaving the crushing plant to a considerable height for it to gravitate to the cyanide plant. In the diagram (Fig. I.) the number of classifiers has been reduced to the minimum possible, viz., one per tube mill, without interfering with efficient classification, but on the contrary, as the figures quoted later will show, considerably increasing it. To deal in further detail with the precise system of classification now to be described, reference should be made in the first instance to Fig. II.

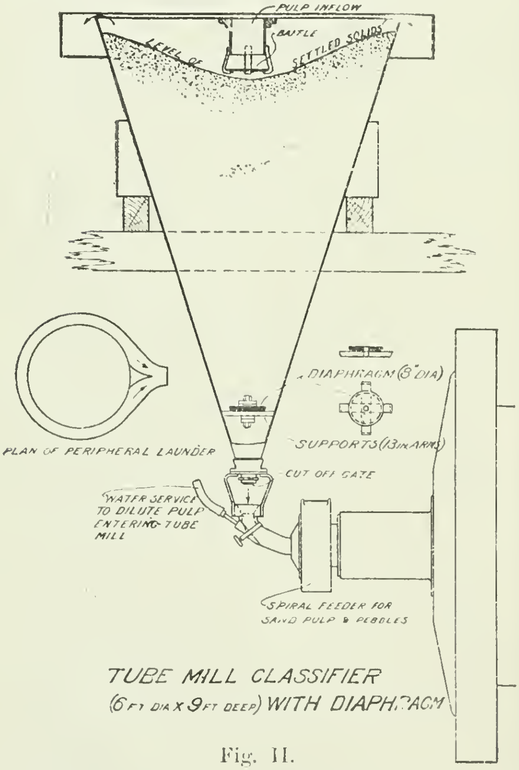

PULP INFLOW
BAFFLE
LEVEL OF SETTLED SOLIDS
DIAPHRAGM (8" DIA)
SUPPORTS (13 IN ARMS)
PLAN OF PERIPHERAL LAUNDER
CUT OFF GATE
WATER SERVICE TO DILUTE PULP ENTERING TUBE MILL
SPIRAL FEEDER FOR SAND PULP & PEBBLES
TUBE MILL CLASSIFIER
(6 FT DIA X 9 FT DEEP) WITH DIAPHRAGM

Fig. II.

The rapid increase in the number of cone classifiers employed on these fields, and elsewhere in mining districts, in place of the old fashioned wooden pyramidal spitzkasten and spitzlutten is due to their greater efficiency, inasmuch as the length of their overflow, which is more than three times as large as that in the usual pyramidal classifier, reduces the velocity of the stream in inverse proportion, so that more efficient settlement is obtained. The use of conical classifiers with peripheral overflow launders, naturally involves a central inflow, in place of the side inflow and overflow over the opposite side usual with pyramidal classifiers. This arrangement reduces the velocity of the overflowing pulp, and consequently allows more tranquil and better settlement.

The disadvantages of the ordinary wooden pyramidal classifiers are well known. They are costly to erect, and the small diameter of the underflow outlets renders them liable to choking, with consequent filling up of the classifier with sand, and resultant disorganisation of classification and overflow of coarse material. In addition, unless a large proportion of water is allowed to pass out with the coarse sand in the underflow, it is necessary to introduce additional water under pressure to prevent choking of the underflow nozzle. The labour and discomfort involved in the attendance upon the old type of classifier is considerable, and they are frequently built so that all parts are not readily accessible. If such constant and uncomfortable work is not conscientiously carried out, their choking is liable to pass unrectified with disastrous results to classification. The usual dewaterer delivering an inflow into the tube-mill with up to 50% and over of moisture, possesses the great disadvantage of a constantly varying underflow, whilst the tonnage of solids cannot be changed without likewise changing the ratio of water to sand

The cone classifiers shown are considerably larger than those in use elsewhere on these fields, being 6 ft. in diameter and 9 ft. deep. This is in accordance with the principle of employing units proportional to the size of the plant, and likewise because each tube-mill having but one classifier, its superficial settling area must be sufficient to ensure the settlement at once of all grains in the pulp entering which require further crushing. The principle of operating this classifier differs from that of other hydraulic classifiers, in that it must be operated with a considerable depth of settled sand. It will likewise be seen that the upper and lower portions of this sand are separated by a disc near the bottom, of which the edges are 2½ in. from the side of the cone. In its original form this disc was serrated, and was supported by the sides of the cone, but a plain disc supported by projecting arms extending to the sides of the cone, as shown in Fig. II., is now used. When a wooden disc is used weights are required, also shown in Fig. II., in order to keep the disc in position until the cone fills up with sand. This disc or partition is termed a diaphragm,* on account of its dividing the upper portion of the cone from the lower, and, as will be seen, its function is entirely different from baffles as used for diverting the course of the stream of fluid pulp. While supporting the shifting mass of descending solids, it permits of a continuous flow between its periphery and the sides of the cone, and at the same time prevents a vertical channel from the top of the cone forming through the mass of solids to the outlet, which would allow thin fluid pulp to descend and escape. Further, the diaphragm relieves the pressure of the mass upon the outlet at the apex of the cone, and therefore should be made fairly strong. The relieving on the outlet of such direct pressure, prevents the undue packing of the solids at the apex, which would result in the pulp issuing very slowly even with a free vertical discharge. This slow rate of underflow would necessitate a very large outlet at the bottom to pass the required tonnage, with consequent liability to formation of a central channel, or bore, and sudden discharge of a very large and unmanageable volume of fluid pulp. Such an underflow, if unchecked, speedily results in the discharge of all the solids in the cone owing to increased velocity. Further, if any attempt be made to run even a moderately thick pulp underflow from a classifier without using a diaphragm, solids settling on the sides are liable to periodically slip and fall like miniature landslides, and thus even if the outlet is not completely choked, the tonnage and ratio of water to solids is continually varied. Since it is obviously impossible to observe what is happening inside a cone classifier, the foregoing explanations are advanced from continued observation upon the behaviour of such appliances. As Mr. Walter McDermott recently observed at a meeting of a kindred society,* the action of the diaphragm is difficult to understand, and is subject to different explanations, but as regards the result that it ensures a steady underflow, in large volume, of thick pulp there is no doubt. The fact that the particles of sand must descend round the periphery of the diaphragm, ensures a more uniform settlement of the solids in the cone than if no diaphragm were used. The upper surface of the settling solids is concave, the edges being higher at the side of the cone. Since the coarser grains naturally settle nearer the centre than the finer grains, and since the former constitute the bulk of the settled particles, it follows that in the upper portion of the cone, the faster downward movement of the descending solids, is nearer the centre than the side of the cone. At the same time particles of insufficient size or density to settle near to the centre, but which the velocity of the current is insufficient to transport over the edge, settle before overflowing, and gradually creep down the sloping side of the cone. If the ratio of water to solids in the pulp entering the classifier decreases, the finer particles in the settled solids accumulate at the side near the top, until the cross-sectional area of the basin is of such dimensions that the velocity of the pulp flowing over its edge, and over the edge of the cone, is sufficient to carry away all particles of such a size as are not specifically heavy enough or large enough to descend. But if with a decrease in the volume of water entering the classifier, there is likewise a decrease in the tonnage of solids, or in the average size of particle in the pulp, both the overflow and underflow under these conditions are correspondingly finer. If, however, whilst the volume of water is decreased, the tonnage and coarseness of the solid particles remains the same, and merely the ratio of water to solid reduced, then the average fineness of both overflow and underflow remain constant. Similarly, if the volume of pulp entering the classifier is increased, but the tonnage of solids and average size of particle remain the same, only the ratio of water to solid being raised, the cone again automatically adjusts itself to the changed conditions, and yields the same grading of overflow and underflow as before. This is due to the area of the basin being increased, by extending its edges towards the side of the cone, and thereby reducing the velocity of the overflow stream at the point where the solids are discharged, to the same as before. If the average size of particles in the pulp entering the classifier be greater, both overflow and underflow are proportionately coarser. In the practical operation of classifiers, it is not only necessary that a thick underflow of the desired tonnage be secured, but that there should also be an equal volume of pulp of the same consistency delivered to each classifier, as otherwise classifiers receiving more than their proportionate share of the total pulp or solids, will overflow a coarser product than others receiving less. It is therefore necessary that the overflow of each classifier should be sampled and tested, to secure uniformity. From the foregoing it will be apparent, that unless the volume of pulp entering is greater than the capacity for which the cone was designed, which will result in the velocity of the stream being greater than will allow of the settlement of the bulk of the desired particles, it follows that the classifier will automatically adjust itself to all conditions. The object of the horizontal baffle shown near the top of the cone, Fig. II. below the inlet, is to prevent the force of the stream disturbing the settled solids, and to divert the flow of pulp in a horizontal direction towards the periphery of the cone.

---

* W. A. Caldecott, "Classification for Tube-Milling," *Journal S.A.A. of Engineers*, December, 1908, p. 101; also this Society's *Journal*, Vol. ix., March, 1909, p. 312.

---

* See *Journal South African Association of Engineers*, January, 1910.

Owing to the low percentage of moisture and consequently viscous nature of the underflow, it is essential that it should have the freest possible discharge from the outlet of the classifier. Any attempt to discharge the underflow through a pipe with a right-angle turn, or through a curved pipe attached to the bottom of the cone, or even through a vertical pipe of the same diameter as the usual outlet, will be unsatisfactory. The best method at present known for discharge of the underflow, is a free vertical fall from the outlet at the apex of the cone through a nozzle as shown in Fig. III. It will be observed that this nozzle is practically a continuation of the sides of the cone, and terminates in a sharp edge so that in no portion are there parallel sides. The friction is consequently reduced to a minimum. This nozzle constitutes the wearing portion of the outlet, and consists of a simple piece of hard cast-iron, which lasts about three weeks, and can be readily replaced at the cost of a few pence. To shut off the underflow of the cone when required, or to regulate the flow, several devices are suitable, one of which, as designed by Mr. S. R. Adams, and shown in Fig. III., does not obstruct the free vertical discharge. The diameter of the outlet to the classifier, shown in Fig. III., is $2\frac{3}{16}$ in., which will pass about 400 tons of sand, with 26% of moisture in 24 hours. This large size is required on account of the slow rate of the flow of the viscous pulp. Incidentally, the wear on the nozzle by abrasion, caused by such slow rate of flow is considerably less than with the usual type of spitzlutten, delivering pulp at a greater velocity. The thick underflow described contains too little moisture for efficient tube-milling and is delivered at so slow a velocity, that the method of feeding the tube-mill by means of the horizontal as jet commonly employed is impracticable. To ensure the delivery of such pulp into the tube-mill, it is necessary to employ either a fixed hopper with gland at the end of inlet trunnion, or preferably a spiral form of feeder, the feeder shown in Fig. II. was designed by Mr. L. Pryce. The additional water required to thin down the pulp for efficient tube-milling, is of course of some assistance in the delivery of the underflow into the interior of the tube-mill. This water may be derived from the mill water service, or may be conveniently obtained from the upper portion of the stream in the tailing launder, in the latter case this water also serves to dilute tube-mill outflow before amalgamation, and is withdrawn from the upper portion of the tailing pulp, by means of a small adjustable decanter, placed in the launder, with an intake opening facing up stream, Fig. I. The amount of water required both at inlet and outlet of the tube-mill is exactly regulated by valves, or cocks, on the pipes from the decanter. With the contemplated removal of amalgamated plates from the stamp-mill, and with very coarse crushing the importance of continuous and reliable classification becomes very great, so as to prevent any but the very finest gold overflowing the classifiers, and to ensure that nearly all the metallic gold liberated by crushing, shall reach the amalgamated plates in the tube-mill circuit. The settling area of a 6 ft. diameter cone is 28·3 sq. ft., so that with the high ratio of tube-mills to stamps, needed with coarse stamp milling, the total settling area in the classifiers becomes proportionately large, and the liability for coarse particles of metallic gold to overflow the classifiers is thus reduced.

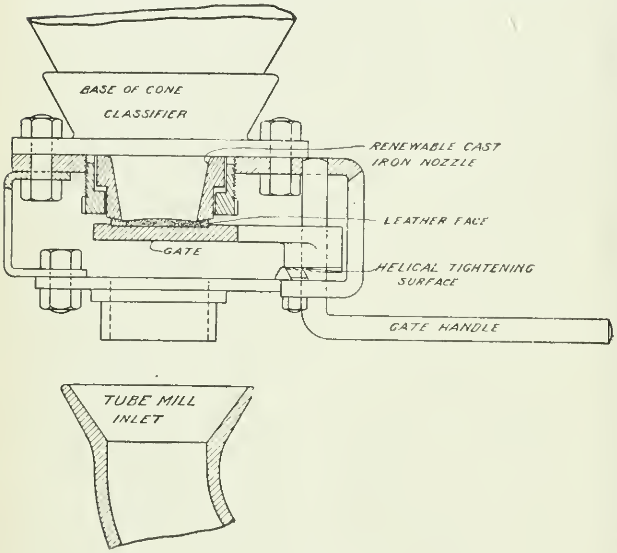


Fig. III.

Samples for moisture or grading analyses of the underflow are readily taken, whilst tonnage determinations may be made by diverting the underflow by means of a launder or large pipe, for a definite time, into a vessel of known capacity, for subsequent weighing or estimation, from the ascertained weight of a cubic foot of such settled sand. Assuming that coarse settled sand occupies 20·3 cub. ft. per ton, the amount of moisture needed to fill the interstitial spaces is 21% on the moist product. The actual percentage of moisture in the underflow of the classifier being as low as 26%, shows the efficiency of the dewatering, the small percentage of excess moisture, being probably due to water filtering down through the descending sand in the classifier, at a faster rate than the sand. The moisture in the underflow decreases with increased coarseness of this product.

Since it is possible to steadily maintain a uniform consistency of underflow, which may be diluted as required and thus to vary the tonnage of solids or moisture independently, this important requisite, as emphasised in the second report of the Mines Trials Sub-Committee on tube-milling efficiency, is better secured, as likewise the minimum amount of fine particles in the tube-mill feed.

On consideration of what has been already stated, and by reference to the diagrams shown, it will be seen that several advantages are secured by the method of classification described. In the first place, the simplicity of the general arrangement and the fewness of the units involved are obvious, and consequently the initial cost of installation, attendance, and the liability to derangement are much reduced. The large outlet at the bottom of the single cone for each tube-mill, precludes the possibility of choking, even when an inclined $\frac{3}{4}$ in. mesh screen in the tailing launder is used, in place of the usual $\frac{1}{4}$ in. mesh, which latter necessitates continual labour in removing ore fragments and debris, under penalty of disastrous results. The saving in elevation by the use of a single cone may be of great importance, especially in an existing plant, when the fall available is insufficient to permit of the installation of the usual spitzlutte, launders, and dewaterer for additional tube-mills. This has been illustrated at the Simmer and Jack, where the satisfactory installation of a sixth tube-mill was thus rendered possible, without change of launders or pumping arrangements, as also at a mining company in Mexico. At this latter plant an additional tube-mill (5 ft. × 24 ft.) was installed, and as little fall was available, there was erected as the tube-mill classifier, a cone 4 ft. 10 in. deep and 4 ft. in diameter, with a diaphragm 8 in. from the apex, allowing 1·5 in. annular space between diaphragm and cone. This classifier was run completely filled with sand, and was fed from the battery at the rate of 240 tons of tailing per 24 hours, produced from screen of 256 holes per square inch. The following were the results obtained from the classifier:—

Gradings.	Inflow.	Overflow.	Underflow.
+ 30	18·5	0·5	34·5
+ 40	10·0	1·0	16·5
+ 80	21·5	8·0	33·5
+100	1·0	1·5	1·5
+150	12·0	11·5	8·0
+200	3·5	9·9	2·0
– 200	33·5	68·5	3·5
Percentage of Moisture	90%	93%	34%

The foregoing overflow is fairly fine, but if a finer final product were required, it would be necessary to reduce the volume of pulp entering the cone, or to employ a larger classifier. As under the conditions of this plant, 38% of moisture gives a considerably higher tube-milling efficiency, than higher or lower dilution, a satisfactory tube-mill product was in this way secured, whereas the usual classifiers had failed to reduce the moisture in the underflow below 48%.

The following table represents the effect produced upon the efficiency of classification for tube-milling, by a recent change at the Simmer and Jack plant, whereby the classification for four tube-mills was converted from the ordinary pyramidal spitzlutten, launders and dewaterers, to a single cone per tube-mill, single cone classifiers having previously been installed for the other two tube-mills.

	Pyramidal Spitzlutten and Dewatering Cones.		Single Cone Classifiers with Diaphragms.	
Period	1st to 8th Dec., 1909		1st to 8th January, 1910	
Tons ore milled...	18,208		17,934	
Grading analyses of tube-mill feed	+ 60	61·8	+ 60	67·1
	+ 90	22·3	+ 90	20·7
	– 90	15·9	– 90	12·2
Grading anrlyses of mill tailing...	+ 60	43·8	+ 60	45·2
	+ 90	11·8	+ 90	11·7
	– 90	44·4	– 90	43·1
Grading analyses of final pulp entering cyanide plant	+ 60	10·9	+ 60	7·4
	+ 90	16·0	+ 90	15·9
	– 90	73·1	– 90	76·7
Relative internal capacity of tube-mills	100		102·5	
Tons – 90 product per tube-mill per 24 hours ...	115·2		126·5	
Relative tons – 90 product per unit of tube-mill capacity	100		107·3	

This comparison of grading analyses and efficiency would appear still more favourable, were it not that only two-thirds of the classification plant was changed, the remaining third having been changed previously to the single cone system. It will be seen from the foregoing, that the tons of – 90 mesh product per tube-mill per 24 hours were raised by 11·3 tons, or allowing for 2·5% greater internal capacity of tube-mills owing to longer average life of liners, the tube-milling efficiency was increased 7·3%. This is due to the better separation in the classifiers of the coarser particles requiring regrinding, and to the thicker and steadier feed of pulp. The low percentage of – 90 mesh product entering the tube-mills from the single cone classifiers will be noted, as also the reduction of the + 60 product from 10·9 to 7·4% in pulp entering the cyanide works. This is due to the fact that the low percentage of moisture carries down very little finely divided solids or slime.

Owing to the less labour and attention required by the classifiers two men were transferred to other work.

I wish to thank the General Manager and the Consulting Metallurgist, for their permission to give the results obtained at the Simmer and Jack, and also Mr. C. O. Schmitt for supplying the drawings.

Mr. S. H. Pearce (*Past President*): I should like to move a hearty vote of thanks to the author of this paper, the subject of which, I think, is interesting to most of us on these fields and probably at no time more than the present. As one of the Committee who are considering the methods to be adopted to obtain greater efficiency in our tube-mills, I may say that the valuable contribution just given, will go a long way towards giving us some pointers on the subject. There are one or two things which, at first sight, strike me as being a little contradictory, about which I would like to speak later on. I do not say this in any carping spirit, for I feel that Mr. Smart has done work and given opinions on a question many of us might have been afraid to tackle so boldly.

Mr. R. G. Bevington (*Past President*): I have great pleasure in seconding the vote of thanks to Mr. Smart. It is evident he has done a great deal of hard work in his investigation of this subject which is interesting to all of us who have to do with tube-mills. There are two very important points upon which Mr Smart has touched; the one is the need for finding some sure method of delivery to each classifier; and the other is the keeping away from the tube mill of as much as possible of the − 90 product which is ready for the cyanide works and which one does not want in the tube, unless, of course, an "all slimes" process is to be resorted to.

THE BARBERTON GOLDFIELD.

(Read at October Meeting, 1909.)

By A. Richardson, M.I.M.M. (Member of Council).

REPLY TO DISCUSSION.

Mr. A. Richardson (*Member of Council*): In reply to Mr. Tudor G. Trevor, the evidence on which my statement with regard to ancient workings was based is to be found recorded in the books to which I made reference.

Mr. E. Browne's interesting description and Mr. W. A. Caldecott's remarks show what efficient work can be done, and what remarkably low working costs can be attained when natural facilities are present.

THE ASSAY OF CYANIDE SOLUTIONS AND SLIME RESIDUE CARRYING DISSOLVED GOLD.

(Read at October Meeting, 1909.)

By A. Whitby (Member of Council).

DISCUSSION.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution in the absence of the author:

Mr. H. R. Jolly (*Member*): The ordinary procedure for conducting an assay by the above method has been dealt with very fully by Mr. Whitby, but the writer has occasionally found that the results are inclined to be slightly low, due possibly to the gold in solution being in a higher state of oxidation than usual, or to the oxidation of the precipitate brought down, which causes it to go into solution again carrying some of the gold with it. (See Dr. Moir's discussion on Mr Whitby's paper, *Journal* for Dec., 1909.) Some experiments were then made with a view to getting over the difficulty. It was decided to effect the precipitation in the presence of a reducing agent, namely, sodium sulphite.

The samples were mixed thoroughly before any tests were made. The experiments were as follows:—

Method No. 1.—20 A.T. of the solution were measured out, 15 to 20 c.c. sodium sulphite added, the whole heated to boiling point and then the solution precipitated in the ordinary way by adding potassium cyanide, potassium ferrocyanide, copper sulphate, 15 to 20 c.c. sulphuric acid and sodium sulphite. The results were satisfactory as the following table shows:—

Solution.	Evaporation. A.V. dwt. per ton.	Method No. 1. A.V. dwt. per ton.
Top Slime	·44	·44
,, ,,	·45	·43
,, ,,	·42	·41
,, ,,	·40	·41
,, ,,	·44	·42
,, ,,	·44	·44
,, ,,	·44	·44

It is generally recognised that heating a solution is a tedious process and should be avoided if possible.

Method No. 2.—Some assays were then made by adding 15 to 20 c.c. sodium sulphite to the solution first, allowing to stand for a few minutes and then precipitating as before but without heating. The same solutions were assayed by the evaporation method and also by Method No. 1. The following are the results :—

Solution.	Evaporation. A.V. dwt. per ton.	Method No. 1. A.V. dwt. per ton.	Method No. 2. A.V. dwt. per ton.
Top Slime	·44	·42	·42
,, ,,	·44	·44	·44
,, ,,	·44	·44	·46
,, ,,	·46	·46	·48

It is evident from these results that heating the solution is a waste of time, so Method No. 1 can be discarded.

The next table is a comparison of results between the evaporation method, the ordinary precipitation method and Method No. 2.

Solution.	Evaporation. A.V. dwt. per ton.	Ordinary Pr't'n Method. A.V. dwt. per ton.	Method No. 2. A.V. dwt. per ton.
Black Sand	16·30	16·42	16·77
,, ,,	·37	·39	·39
,, ,,	·40	·42	·43
,, ,,	·50	·52	·53
Top Box 1	6·88	6·95	7·06
,, ,, 3	2·16	2·21	2·21
Top Slimes	·44	·41	·46
,, ,,	—	·38	·39
Black Sand	—	3·22	3·46
,, ,,	—	1·94	1·99
,, ,,	—	·60	·61
Average % Difference	97·12%	97·55%	100%

The above results show that the addition of sodium sulphite before and after precipitation gives higher values than either of the other two methods.

The final precipitate is much more granular, not so bulky and filters quicker than that from the ordinary method and the colour is also different. It is much lighter and in some cases is light yellow, as compared with the dark brown colour of the other. The filtrate is clearer and remains so.

Great care was exercised in all the evaporation assays in order to get the highest results, in fact, new evaporating dishes were used for the tests.

Some experiments were carried out with the addition of sodium sulphite at the beginning only, but the results were not so satisfactory.

REPLY TO DISCUSSION.

Mr. A. Whitby (*Member of Council*): My only critic, Mr. White, deals with the subject of my note in his usual manner. He ignores the principal motive, the use of the method in a recent foreign publication as an original one, and conveys the impression that it is not entitled to consideration on account of its resemblance to De Wilde's process of gold recovery. That is manifestly unfair. One might as well say that Alexander's method of determining lead is not entitled to bear his name because the insolubility of lead molybdate was previously well known. I was well aware of the remarks of Dr. Loevy and Mr. J. R. Williams on De Wilde's method, but so far as I know, neither of these gentlemen ever made any practical use of it in laboratory work. Virgoe was the first to publish a method of *assay* by precipitation with copper salts, and I followed Virgoe and, although I am open to correction, I do not think Mr. White said or did anything with his application of De Wilde's solution till my method was already established. My objection to using cuprous chloride in acid solution was based on its instability. Some experiments recently conducted show that with care it gives results almost as good as my own process, but I still do not think it sufficiently reliable. I should like to know why Mr. White so needlessly uses 80 assay tons of even box end solutions. Surely a 2500 c.c. beaker must be very awkward to handle, and he can get quite sufficiently accurate results with an extreme of 40 assay tons in special cases, even with the older type of balance. I do not think further that a series of 2500 c.c. beakers would find room in most assay offices.

My critic's remarks on the slime residue assay are equally airy, and if he only found differences of ·05 dwts. between his treated pulp and the untreated, he can hardly have had effective precipitation, or his residues were extremely low. At the time when these experiments were carried out it is hardly likely that slime residues were often below ·4 dwt. with the old two decantation ratio of solution to dry slime, and the soluble gold was somewhere in the neighbourhood of ·25 dwt. Almost the whole of this is lost in simple drying, although when old pans are used, it is possible to adventitiously include some gold from previous samples. In this connection it is somewhat to be regretted that Mr. Rusden did not publish the results of the experiments spoken of by Mr. Pearce and reverted to by Mr. White. Why the subject of slime residue assays, on which most assayers have something to say, should be more or less "taboo" in this Society, ever since Mr. Caldecott's paper many years ago, passes my comprehension. It would almost seem as if that class of result which shows a higher percentage of gold recovery in the reduction plant, is sometimes the most acceptable.

I have to thank Dr. Moir for his kindly little note on the chemistry of the process and Mr. Jolly for his interesting experiments, showing the necessity of continuity in the maintenance of reducing conditions.

EXPERIMENTS ON THE ASSAY OF ACID WASHES RESULTING FROM THE CYANIDE "CLEAN UP" BY THE USE OF BISULPHATE.

(*Read at October Meeting, 1909.*)

By L. J. Wilmoth (Member).

REPLY TO DISCUSSION.

Mr. L. J. Wilmoth (*Member*): Considering the very little discussion that has taken place on my paper, a very few remarks will do by way of a reply. Mr. Crosse's question as to the state of the gold in these acid washes I have already replied to by saying I consider most of the gold is in suspension, the rest being in solution. On January 15th I assayed a sample of acid wash after thorough shaking of the bottle and obtained a result of 0·36 dwt. per 2,000 lb. The rest of the sample was put on one side and left without being disturbed until the 16th of this month, some 29 days later, when it was again assayed. This time, however, only the top portion of the sample was taken, the lower portion being left undisturbed. The assay result was ·07 dwt., the difference undoubtedly was due to the separation of the gold values from the solution due to settlement during the 29 days. As to whether a longer period of settlement would have reduced this ·07 dwt. still lower I cannot say, but it is more than likely this value represents the gold in solution. The effect of simple settlement in the treatment of acid washes for their gold values might be worth investigating. An extreme case of gold in suspension I mentioned in the original paper when a sample was reduced in value by only a couple of hours settlement from 1·27 dwts. to 0·23 dwts. In regard to the condition of the small proportion of dissolved gold, Dr. Moir, in his brief contribution, throws some light.

Mr. Whitby has objected to the use of the term "Standard Method." Perhaps the term might have been omitted, but in all experiments of this type some definite point must be used and hence the term "standard." I think it is also further justified by the general use of this method along the Rand.

Once again I would disclaim any originality in any of the methods mentioned. The method referred to as the "standard method" has been in use on the reef, as far as I know, some four years.

With Mr. H. A. White I disagree, when he considers we assayers should measure our solutions and not weigh them. This measuring is perfectly admissable with ordinary cyanide solutions, but, with solutions having a specific gravity of 1.12, as quoted by the President, it is absolutely wrong. A very little trouble on the part of the cyanide manager will secure him the actual weight of his acid wash. The assayer is asked for the assay value, and unless his instructions are to the contrary, he should report his values by weight and not by volume.

In reply to Mr. Watson, the alkaline sulphide in the acid solution was used as a matter of convenience.

In conclusion I would like to thank all those who took part in the discussion on the paper, and once more appeal to assayers to come forward with any experimental or other work bearing on assaying.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

PART I.—CRUCIBLE EXPERIMENTS.

(*Read at November Meeting, 1909.*)

By Prof. G. H. Stanley (Member of Council).

DISCUSSION.

Mr. F. J. Pooler (*Associate*): We all await with more than ordinary interest Prof. Stanley's practical results from Pretoria, but as an instalment, preparatory to these, his crucible experiments are more than interesting, and serve to whet our appetites for what is to follow.

His experiments seem to indicate that Ti_2O_3 as a base has been the previous source of trouble, but one would like to know more definitely what information is available as to the existence in the slags of a titano-silicate such as $Ti_2O_3\ 3SiO_2$ and it is to be hoped that further attempts to form it, as in experiment (*j*), will be more successful and less costly than that one. For it seems fairly obvious that unless the existence of some such compound can be definitely demonstrated, the conclusion that Ti_2O_3 is functioning as a base can not absolutely be relied upon. Is it not possible that Ti_2O_3 and SiO_2 are both acting, the former functioning as a base and the latter as an acid? In support of this I may say that I have made various calculations in an endeavour to make up a formula for the slags mentioned, and treating all titanium, as Ti_2O_3 have failed to get silica enough to go round the various bases present, in several cases, the presumption being that some of the titanium is acting as TiO_2 forming titanates to make up for the deficit in silicates, unless the slags obtained had considerable excess of the bases. This, of course, can only be settled by analyses. Again as Ti_2O_3 was thought to be formed, why are analyses of experiments I. to VII. given as TiO_2? It has occurred to me that the presence of TiO_2 might be ascertained quantitatively by reducing some well powdered slag in a current of hydrogen, as is mentioned in Roscoe and Schoelemmer, p. 643 (3rd edition), when analyses before and after reduction would indicate what change, if any, had taken place. (The oxides of iron must first, of course, be removed.) There may be some experimental reasons against this, and I shall be glad to be told of such by Prof. Stanley in his reply.

If the above surmise is correct, it seems that a reducing atmosphere is an essential for any considerable success in the larger experiments to be undertaken, as otherwise the formation of TiO_2 in considerable quantities seems highly probable, seeing that, as is well known, Ti_2O_3 is not altogether formed, if titanium ores are reduced and then cooled in other than a reducing atmosphere.

Another method of finding out if TiO_2 is present would be as follows: pass dry chlorine over heated powdered slag, made into balls with powdered charcoal, exactly as is done in preparing $SiCl_4$; but it seems uncertain if the time such experiments would take will in any measure repay the authors.

In making calculations on the reduction of $2TiO_2$ to Ti_2O_3 and Fe_2O_3 to $2FeO$, a peculiar relationship is noticed, which may complicate matters considerably, if all iron is not removed before the titanium peroxide is reduced, thus

$$\begin{array}{cccc} & 2TiO_2 & \rightarrow & Ti_2O_3 \\ & \underbrace{2(48+32)}_{160} & \rightarrow & \underbrace{96+48}_{144} \\ \text{and} & Fe_2O_3 & \rightarrow & 2FeO \\ & \underbrace{112+48}_{160} & \rightarrow & 2(56+16) \\ & & & 144. \end{array}$$

It will be seen that the same proportion holds for the loss of weight in the two cases.

ENDLESS ROPE HAULAGE.

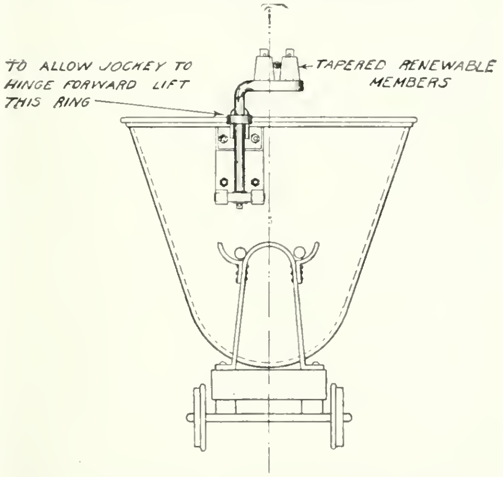
(Read at December Meeting, 1909.)

By H. G. Kay (Associate).

DISCUSSION.

Mr. G. A. Webb (*Visitor*): The very interesting paper read before you by Mr. Kay describing endless rope haulages, and the methods adopted at the Premier Mine, should provoke considerable discussion, as it is to the adoption of automatic and labour-saving devices we have to look for a further reduction of costs on these fields.

The form of haulage mentioned as being "very suitable for tailings dumps" with the jockey fixed to the side of the trucks might be all right in some cases, especially if the plant was designed to suit the haulage but even then it has the drawback as applied to the average gold mine that if any accident should occur to the full trucks, the empty ones are also held up and *vice versa*. Another point which appears to me to be objectionable is the tendency of the leading near side wheels, as also the back off side wheels, to be continuously flanging, tending to run off on account of the distorting pull on the frame.


TO ALLOW JOCKEY TO HINGE FORWARD LIFT THIS RING
TAPERED RENEWABLE MEMBERS

I certainly agree that "slogging out" boys should be dispensed with, and should not be necessary where proper attention is paid to the jockeys. The frame as shown in the diagram, but without the runners, is the usual construction for sands dump haulages. With these runners a fixed type of jockey must be used, which, of course, would not be suitable where they have to be taken out to pass under tanks where there is not sufficient head-room, unless they are hinged.

The very important items of onsetting, and releasing points, methods of attaching, speed, etc., should be discussed with advantage.

I was particularly interested in the description of the jockey in use at the Premier Mine (Pack's patent) on which the jaws are turned 90° to get four wearing surfaces for each jaw. I have brought some jockeys here to-night to show you which have similar properties with the added advantage that the gripping parts can be easily made to follow the wear of the rope and are so simply detached.

As it is with maintenance costs you are all deeply concerned, it may interest you to know that with the type of jockey shown the costs can be considerably reduced showing a saving of over 100% compared with the ordinary U type.

On haulage having a mean grade of 15·7% roughly 1 in 6·36 by experimental tests which have been conducted over 18 months (although four months' costs were taken for these figures) it was shown the costs could be reduced in this instance from £13 14s. 3d. to £6 - and this allowing to replace the whole wearing blocks each month, the average life of these wearing blocks (as per samples) was over six weeks.

(Mr. Webb demonstrated his remarks by an exhibition of the jockeys and runners invented by himself.)

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Solubility of Lime in Water.—"Determinations of the exact solubility of lime, prepared by igniting purest calcite, were made by the authors, using an apparatus specially designed to avoid all sources of error as far as possible. The following table shows the solubility of pure lime in water at the temperature stated :—

Temperature. °C.	Weight of Water in Grams required to Dissolve 1 Gram CaO.
2	768·5
10	786·8
15	804·3
20	826·4
25	868·7
30	908·2
40	988·1
50	1083·0
60	1179·0
70	1274·8
80	1368·1

—G. T. Moody and L. T. Leyson, *Trans. Chem. Soc.*, 1908, *93*, 1767-1772.—*The Analyst*, Feb., 1909, p. 73. (E. H. C.)

Chemical Methods for Distinguishing Carbon Monoxide-Hæmoglobin from Oxyhæmoglobin.—"The author has investigated a large number of the proposed methods for the differentiation of carbon monoxide-hæmoglobin from oxyhæmoglobin, and has selected the following modifications of Zaleski's and Welzel's tests :—The blood containing carbon monoxide and a control blood containing oxygen are diluted a hundred times with water, care being taken to agitate the carbon monoxide blood as little as possible, since the compound is unstable in the air. The cylinder must be tightly corked. If the test is required to yield a rapid result with a transient effect, 5 c.c. of each of the solutions of blood are placed in two test-tubes, and treated with 5 drops of concentrated copper sulphate solution. The contents are mixed by inverting the tubes once, and the carbon monoxide blood should then show a purple-red colour, whilst the ordinary blood is coloured green. Small differences are best detected by looking at the contents of the tubes from above. In a few minutes the carbon monoxide blood also turns greenish, but the difference may be made to last longer if only 1 drop of copper sulphate be employed. If the test is not required immediately, but is desired to be permanent, 5 drops of a freshly-prepared solution of tannin are used instead of the copper sulphate. In the case of carbon monoxide blood, the flocculent precipitate which is formed remains rose-red in colour; in the case of the ordinary blood, it gradually becomes dirty red and finally brownish, and the difference remains perceptible for months. The tube containing the carbon monoxide blood must be tightly corked whilst the test is being effected."—K. Bürker.—*Zeit. biol. Tech. und Methodik*, 1908, 1, 146.—*Zeit. anal. Chem.*, 1909, *48*, pp. 205-207.—*The Analyst*, April, 1909, p. 161 (E.H.C.)

Changes in Dilute Solutions of Permanganate.—"With regard to these changes, attention is again called to the observation that the changes take place whether the solutions are subjected to the action of light or whether they are kept in the dark, although light accelerates the changes. One explanation is that the dissolved gases, the atmospheric gases, particularly carbon dioxide, may have been the influencing causes. Against this view is the fact that carbon dioxide usually converts manganates to permanganates very rapidly: and oxygen would scarcely act as a reducing influence, whilst nitrogen is completely inert. Nor could the changes have been caused by any dissolved organic substances, for the solutions were made in distilled water, and from pure crystallised salts; and also, all the glass vessels were well washed before use with a warm mixture of hydrochloric and nitric acid. Nor could any organic substances have entered the solutions, for the latter were always kept well covered up. Nevertheless, in order to eliminate the action of any extraneous gases or dirt, another series of solutions of potassium permanganate was made, and the glass bottles were filled to overflowing. The contents were isolated from the air by well ground stoppers. The bottles of one series were filled in the dark, and they stood in cupboards to which no light could penetrate. A similar series of bottles stood in a well lighted room and they were exposed to the full action of light. The changes in the colour of the solutions were exactly the same as in the earlier observations; but those of the solutions exposed to the action of light were much quicker. Besides the colour changes, a small bubble of gas collected in the bottles. The absorption spectra were not observed.

The explanation which seems to account for the changes is that the dissociating force of the water slowly acted upon the dissolved permanganates, so that the ionic condition of $\overset{+}{R}$ and $\overset{-}{MnO_4}$, where $\overset{+}{R}$ represents the metallic ion, broke down. One may imagine that the $\overset{-}{MnO_4}$ ion undergoes further dissociation into MnO_3 and O, or into MnO_2 and O_2. In such dilute solutions the tension between the molecules of water surrounding the ions must have been very considerable, and it may have been so great as to destroy the MnO_4 condition, so that further changes took place which became visible in changes of colour, in changes in the width of the bands and in the liberation of oxygen; and these changes would be accelerated by the vibrations of light. The intermediate stages of the changes might be represented by the narrowing of the bands 1, 2 and 3 and the widening of the bands 4 and 5, and these might correspond to changes of the MnO_4 ions into MnO_3 and O ions. The complete change in colour from the well known permanganate tint to a light brownish yellow colour in the more diluted solutions, the disappearance of the absorption bands, and the increased general absorption represented the complete change into MnO_2 and O_2. The MnO_2 would be dissolved in the colloidal condition, and it would be represented by the complete disappearance of the bands, and the increased general absorption, whilst the oxygen partly escaped." — J. E. PURVIS, M.A. — *Proc. Cambridge Philosophical Society*, May, 1909, p. 255. (H. A. W.)

ESTIMATION OF NICKEL IN NICKEL-STEEL BY MEANS OF DIMETHYLGLYOXIME.—"The following method is stated to be rapid and trustworthy for the estimation of nickel in nickel-steel. Five grams of the drillings are dissolved in 50 c.c. of hydrochloric acid (specific gravity 1·125), the mixture being gently heated to aid the solution. Concentrated nitric acid is then added, drop by drop, until the solution is clear and dark red in colour, and the whole is concentrated to a volume of about 25 c.c. The residual solution is rinsed into a separating funnel with hydrochloric acid (specific gravity 1·125) and shaken with 100 c.c. of anhydrous ether, which removes the greater part of the iron, the solution being then shaken with three successive quantities of 5 c.c. of ether containing hydrochloric acid. The acid solution containing the nickel is heated until most of the ether remaining in solution has been evaporated, filtered, the filter washed with hot water, and the filtrate diluted to a volume of about 500 c.c. After the addition of a little tartaric acid, the solution is neutralised with ammonia, then rendered slightly acid with hydrochloric acid, and dimethylglyoxime, in the form of a 1 per cent. alcoholic solution, is added. On heating the mixture and adding ammonia, scarlet-red nickel oxime is precipitated. The precipitate is collected on a weighed filter, and the clear filtrate is passed through a filter of equal weight to the first; both filters are washed with hot water, dried, and weighed together in a weighing-bottle. The empty filter is now removed, the bottle and full filter are re-weighed, and, after removing this filter, the bottle is weighed empty. The weight of the precipitate thus obtained, multiplied by 0·2031, gives the quantity of nickel."—A. IWANICKI. —*Chem. Zeit. Rep.*, 1908, *32*, p. 644.—*The Analyst*, Feb., 1909, p. 74. (E. H. C.)

ESTIMATION OF SULPHIDE IN ALKALI CYANIDE. —"Of the three methods studied, that of titration with lead nitrate, with lead paper as an external indicator, is the most rapid and convenient, and if suitably performed and corrected, it is sufficiently accurate for technical purposes with quantities of potassium sulphide below 0·1 per cent. *G. W. Williams's modification** of the colorimetric process (with sodium hydroxide and lead acetate) is more accurate, but less rapid, and it is necessary to use boiled water and to check the strength of the standard solution very frequently. Lastly, Feld's method, depending on the expulsion of sulphuretted hydrogen by distilling with magnesium chloride in an atmosphere of carbon dioxide, is the most accurate, but too laborious for technical purposes. The author describes the following procedure for performing the titration method with lead nitrate: —10 grams of the powdered cyanide are stirred quickly with 15 c.c. of ordinary water, any lumps being broken up with a glass rod. Any small undissolved residue will generally go into solution before the titration is finished. The lead nitrate solution (1 c.c. equivalent to 1 mgm. of potassium sulphide) is then run in quickly, so long as it produces an increase in colour; after this it is added gradually, the solution being tested by placing drops of it in contact with drops of lead nitrate solution on thick blotting paper. The author has carefully investigated the errors involved in the lead titration method, due to oxygen in the water, oxidation whilst the substance is being dissolved, and oxidation during titration. These errors are very considerable; but if the procedure described above be adopted, one and a half to two and a half minutes being required for dissolving, and three to four minutes for titrating, the errors are sufficiently constant at from 18 to 26 per cent. of the sulphide present. If then the results be multiplied by the factor 1·25, or if, as is more convenient, a solution containing 2·4 grams of lead nitrate per litre be assumed to correspond with 0·10 per cent. of potassium sulphide per 1 c.c. used, the results are accurate to 2 or 3 units in the third place of decimals, and compare favourably with those obtained by Williams's method."—T. EWAN.—*Journ. Soc. Chem. Ind.*, 1909, *28*, pp. 10-13. *The Analyst*, March, 1909, p. 117. (E. H. C.)

VOLUMETRIC METHOD FOR THE ESTIMATION OF TITANIUM AND OF TITANIUM AND IRON IN THE PRESENCE OF EACH OTHER.—"In one part of a solution containing titanium and iron, the titanium is estimated, after reduction by zinc and hydrochloric acid, by titration with a solution of methylene blue, which is standardised against a standard titanium solution. In another part of the original solution, iron (in the ferric state) is estimated by titration with titanous chloride solution. Iron salts have no action on methylene blue, but the lower oxides of vanadium and tungsten reduce it. The reduction of the titanium is conveniently carried out in a flask closed by a rubber stopper, carrying a Bunsen valve and two short glass rods. The top of the valve is closed, not by the usual glass rod, but by a short glass tube closed by a small rubber stopper. Through this rubber stopper a piece of platinum wire is inserted, the lower end of which, inside the flask, carries a small rod of zinc, which effects the reduction. When this is complete, the zinc is raised above the liquid by means of the wire, carbon dioxide is passed through the flask by removing the two small glass rods, the zinc rod is rinsed with water, and the titration effected. The maximum error of the method is about 0·3 per cent." —E. HIBBERT.—*Journ. Soc. Chem. Ind.*, 1909, *28*, pp. 189-190.—*The Analyst*, April, 1909, p. 182. (E. H. C.)

DIRECT VOLUMETRIC METHOD FOR THE ESTIMATION OF ALUMINA.—"The precipitate of alumina and iron oxide, obtained as usual, is dissolved in the smallest possible quantity of concentrated hydrochloric acid. To this solution 2 to 5 grams of sodium peroxide are added, so as to yield a strongly alkaline solution containing aluminium as aluminate. The solution is filtered from ferric hydroxide, and exactly neutralised with hydrochloric or sulphuric acid, after the addition of phenolphthalien. The liquid is then heated almost to boiling, methyl arange is then added, and then standard sulphuric acid, until the aluminium hydroxide previously precipitated is entirely redissolved, and a permanent acid reaction is obtained. The quantity of standard acid used corresponds to the alumina present. The method is inapplicable in the presence of lead or zinc. Sodium peroxide is used to make the solution alkaline, instead of sodium hydroxide, on account of its greater purity."—W. H. SEAMON, *Western Chemist and Metall.*, Feb., 1909.—*Chem. Engineer*, 1909, *9*, p. 116.—*The Analyst*, June, 1909, p. 291. (E. H. C.)

---

* See this *Journal*, Vol. vi., Nov., 1905, p. 170.

SEPARATION OF ANTIMONY FROM TIN.—"The quantitative separation of antimony and tin is one of the most difficult in analytical practice. Of the methods hitherto proposed, that of Vortmann and Metzl (*Analyst*, 1906, *30*, p. 281), is the simplest, and gives useful results. The author, however, has found that a ready means for the separation of the sulphides of these two metals may be based on their different solubilities in hydrochloric acid of different concentrations. Antimony sulphide, though soluble in hot concentrated hydrochloric acid, is insoluble in the 15 per cent. acid, whilst tin sulphide is completely soluble. When both metals are present in an acid solution of such concentration, saturation with hydrogen sulphide at the ordinary temperature throws out a little of tin sulphide in association with the antimony, but this may be avoided by performing the precipitation at a temperature of 50° to 60° C. A mixture of antimony and stannic salts is treated with strong hydrochloric acid sufficient to give a concentration of 15 per cent. of the acid. The beaker containing the solution is then placed in a water-bath, and the temperature is maintained at 50° to 60° C.; a strong current of hydrogen sulphide is then passed through the liquid for thirty minutes, and the antimony sulphide separates readily as a scarlet precipitate. The liquid is then cooled below 30° C., and a moderate stream of hydrogen sulphide is passed for ten minutes. The clear liquid is decanted off through a Gooch crucible, previously dried at 110° C. and tared. The precipitate is rapidly collected, washed with 50 c.c. of 15 per cent. hydrochloric acid saturated with hydrogen sulphide, then with water and hydrogen sulphide, and then in succession with alcohol, a mixture of alcohol and carbon bisulphide, carbon bisulphide, again with alcohol, and finally with ether. It is dried at 110° C. and weighed. The tin remains in the filtrate, which is partially neutralised with ammonia, diluted with water, warmed and saturated with hydrogen sulphide. The results are exact, and are independent of the relative quantities of the two metals. The method is applicable in all cases where antimony is present in the form of a solution of the trioxide."—G. PANAJOTOW.—*Ber. deut. Chem Ges.*, *42*, pp. 1296-1299.—*The Analyst*, June, 1909, pp. 291-2. (E. H. C.)

RAPID VOLUMETRIC METHOD FOR DETERMINING CERIUM IN PRESENCE OF OTHER RARE EARTHS.—"Cerous salts in sulphuric acid solution are quantitatively oxidised by sodium bismuthate to ceric salts, and the latter can then be again reduced by means of ferrous sulphate added to known amount, the excess of this salt being determined by titration with permanganate. The permanganate solution is approximately one-fortieth normal, and is prepared by dissolving 10 grams of potassium permanganate and 50 c.c. of strong sulphuric acid in water, and making up to 1 litre. The value of this solution in terms of Fe, × 3·08415, gives the equivalent of CeO_2.

The method is accurate, as shown by the following results:—CeO_2 taken (1) 0·1901 gram (determined by oxalate precipitation; found (*a*) 0·1897, (*b*) 0·1907 gram CeO_2. Taken (2) 0·2102 gram; found (*a*) 0·2103, (*b*) 0·2104 gram CeO_2, (3) Taken, 0·1928; found (*a*) 0·1927, and (*b*) 0·1922 gram CeO_2."—F. J. METZGER.—*J. Amer. Chem. Soc.*, 1909, *31*, pp. 523-525.—*The Analyst*, July, 1909, p. 335. (E. H. C.)

DETERMINATION OF VANADIUM.—"*Method for Products Low in Silica.*—Weigh out from 0·250 to 1·000 gm. of the finely powdered material and mix in a nickel crucible of about 25 c.c. capacity with 3 gm. of sodium peroxide; cover with about 1 gm. more of the peroxide and heat over a Bunsen burner; a dull red heat being sufficient. When the darkening which gradually takes place on the surface is complete, the operation may be considered finished.

Fill a 500 c.c. beaker half full of cold water and transfer the cold crucible containing the melt to it. Care should be taken at this point as the reaction between the peroxide and water is violent. It is well to float the crucible in the beaker, cover with a large watch glass and add water to the contents of the crucible a few drops at a time from a wash bottle. When solution is complete, remove and wash the crucible, heat to boiling and pass a current of carbon dioxide through the solution until saturated. Remove from the heat, allow to settle and pour through a filter supported on a platinum cone. Wash three times by decantation and three times on the filter using hot water containing a little sodium carbonate. Suction can be applied toward the end of the operation if found necessary.

The filtrate is brought just acid with nitric acid, boiled and the vanadium precipitated with lead acetate; the slight excess of nitric acid being corrected by the addition of a gram or two of sodium acetate.

The precipitate of vanadate of lead is separated by filtration, and dissolved in a small quantity of dilute nitric acid. To the solution add 10 c.c. concentrated sulphuric acid, evaporate to fumes, cool, add cold water and separate the sulphate of lead by filtration, catching the filtrate in a flask. The filtrate which may amount to 150 c.c., is boiled and reduced by sodium sulphite, using 1 gm. and adding in successive small portions. The excess of sulphur dioxide is removed by boiling; for convenience in testing this point, fit a stopper and delivery tube to the flask. When the escaping steam no longer decolorises a dilute solution of permanganate of potash, held in a small beaker, the boiling is stopped and the hot solution titrated with standard permanganate.

The iron factor multiplied by 0·916 equals vanadium.

If the material contains arsenic, it will be necessary to remove it. A convenient point at which to do this is after the first reduction by sodium sulphite. Pass hydrogen sulphide, filter, boil out excess and titrate.

A correction from the burette reading should be made as follows: To 150 c.c. of water in a flask, add 10 c.c. sulphuric acid and 1 gm. sodium sulphite. Boil out the sulphur dioxide and titrate. The amount of permanganate used to colour the solution should be deducted from the assay. The correction usually amounts to 0·2 c.c.

Method for Ores.—Treat 1 gm. of the finely ground ore (less if high grade) with 80 c.c. of aqua regia and evaporate to dryness. Add 10 c.c. nitric acid, 1·20 sp. gr. and digest on the hot plate for a few minutes. Filter off the silica, etc. Nearly neutralise the filtrate with caustic soda and pour it with constant stirring into a hot solution of caustic soda contained in a large beaker. A stick of caustic soda dissolved in 200 c.c. of water is about the right strength. Heat the solution to boiling and pass a current of carbon dioxide gas. From this point the method is carried out as described above.

For Products containing much Silica and not readily decomposed by acids, the method of heating with sodium peroxide may be used, but in this case it will be necessary to remove the silica. The filtrate from the iron, after slight acidification with nitric acid, is made alkaline with ammonia and 3 gm. or 4 gm. of ammonium carbonate added. The beaker is put on a warm plate and allowed to stand for an hour with occasional stirring. The voluminous precipitate of silica is separated by filtration on a 12 cm. filter supported on a platinum cone using suction. This precipitate, although somewhat inclined to be sticky, is not difficult to filter if suction is used. The washing with hot water, containing a little ammonium carbonate must be thorough. Ten or twelve times is not too much, churning up the precipitate well with the stream from the wash bottle. The filtrate now freed from the greater part of the silica is again made slightly acid with nitric acid and the precipitation with lead acetate made as before.

When using the acid method of decomposition, it is not always safe to assume that the vanadium is all extracted by treatment of the ore crushed to 80 mesh or coarser, with dilute nitric acid, as is sometimes recommended. Some vanadiferous sandstones yield to this treatment, but not all and the writer, has repeatedly found vanadium in residues treated in this manner. To make sure of complete extraction, the ore should be finely ground in an agate mortar and decomposed first with aqua regia.

The direct reduction and titration of the vanadium in the presence of the accumulated salts of the analysis is inaccurate, the end point being uncertain and successive reductions generally failing to check ; so it is always necessary to separate the vanadium from these by precipitation with lead acetate, the titration of pure vanadium solutions being exact. If rapid approximate results are wanted, the filtrate from the iron residue may be acidified with sulphuric acid and the solution reduced and titrated.

The method as described has the advantage over the ammonium carbonate method in that the separation of the iron and vanadium is performed, in nearly every case, at a single operation, and obviates the necessity of a re-solution and reprecipitation of the iron residue. A clean separation can be made on iron vanadate, using 300 mg. of substance.

The carbon dioxide is used to prevent bumping, although in some cases its use apparently gives a cleaner separation, due possibly to the tendency of the alkaline carbonates formed to hold vanadium in solution."—H. F. Watts.—*Western Chemist and Metallurgist*, Nov. 1909, p. 408. (H. A. W.)

METALLURGY.

The Hunt Continuous Slime Filter.—"The Hunt Continuous Slime Filter is designed for the separation of liquids from finely divided solids, and is especially adapted to the cyanide, chlorination, and other chemical processes for treatment of ores. It is also most suitable for the dewatering of tailing in localities where water is expensive or where it is desirable to follow wet crushing by cyanidation or other chemical treatment.

In its general design the machine consists of an annular chamber which is connected with the vacuum pump and the top of which constitutes the filtering surface. The filter used is a form of sand filter, which the inventor first applied over 15 years ago and has used most successfully in leaching ores.

A great deal has been published recently regarding the troubles incidental to the use of canvas or filter cloth in filter-presses and vacuum-filter machines. With a sand filter properly arranged these troubles do not exist. A clear filtrate is obtained and no subsidiary filter is required.

In making the filter-bed, slats of triangular section are placed side by side but not touching, a space of about three-sixteenths or one-quarter of an inch being left between the bases of the slats. Carefully screened gravel, as nearly uniform in size as possible and too large to pass through the openings between the bases of the slats, is then placed in the lower part of the bed. Above these, finer gravel or coarse sand is spread in a layer about 1 in. deep to form a floor on which is spread clean sand of 8 to 12 mesh size, to a thickness of an inch or so above the tops of the triangular slats. This forms the permanent filter-bed and is supported strongly enough to remain perfectly rigid on the application of a vacuum in the chamber underneath. This chamber is connected with a wet vacuum-pump which withdraws the solution and maintains the vacuum. The walls of the structure may be made of concrete and those being raised above the top of the filter-bed constitute circular tracks on which a carriage revolves. This carriage has an iron scraper in front, which is of the same width as the filter-bed, and removes the layer of residue from the permanent filter-bed. Following close behind the scraper, on the carriage, is placed a distributor similar to that used on a Frue vanner, which distributes the sandy portion of the pulp. Behind this is a second distributor which spreads the slimy portion of the pulp over the layer of sand just laid down by the first distributor. On the central column of the machine are two concentric hoppers or funnels, which are connected by pipes with the two distributors and which receive the pulp. Before reaching the filter machine the pulp is roughly classified, so that a portion of the fine sand is separated in a clean condition, and this clean sand is fed upon the filter-bed by the first distributor, the rest of the pulp going to the second. In this way the slimy portion of the pulp is distributed over a layer of clean fine sand. As the carriage moves continuously the deposited layer of sand and slime is also continuously removed by the scraper, while the first material is deposited by the distributors, as described above. Pipes to spray wash solution and wash water follow the carriage at suitable distances, so that the residue can be thoroughly washed.

The machine has an extreme diameter of 15 ft., the annulus being 3 ft. wide. This gives a filter surface of 113 sq. ft. On the supposition that the layer of residue contains 50% moisture and weighs 109 lb. per cub. ft., if it is ¼ in. thick, it will amount to 2·26 cub. ft. and will weigh 256 lb., or 123 lb. dry. At a speed of one revolution of the carriage per minute, this represents an output of 3·69 tons of dry material per hour, or over 80 tons per day.

The vacuum-chamber and base are made simply of concrete, the bottom of the vacuum-chamber being arranged so as to drain to the outlet. The triangular slats are put together in sixteen sections which fill the annular space and form the support for the filter bed of sand. The carriage is driven from a heavy gear which revolves on ball bearings on the central pillar. The gear is made amply strong to meet all strains and stresses. The pinion is driven by a worm-gear with cut teeth running in oil, thus securing the necessary speed reduction direct from an ordinary line shaft. The arm which connects the carriage with the driving gear is built of channel irons and is very strongly braced. The carriage carries the scraper in front which is set at an angle of 60° with the radius so as to discharge the residue over the outer edge of the filter-bed. Overhanging the edge of the filter-bed is a bucket which receives the residue as sent over by the scraper. At one part of the circle is fixed a tripper, which overturns the bucket as it passes; by this means the residue is discharged at one point and can be removed by a travelling belt or a car. When water is abundant this bucket is not required, but the residue may be discharged direct into a tail-race which surrounds the machine."—*Mexican Mining Journal*, Sept., 1909, p. 27. (G. H. S.)

CYANIDING IN AMERICA.—"In some respects America is ahead and in others it is behind in milling and cyaniding. The new 100-stamp mill of the Roodepoort United company, now under construction in Africa, has 1,900-lb. stamps, which are expected to crush 12 tons each per day. The capacity record now held there by the Cinderella Deep is over 9 tons per 1,600-lb. stamp. There is nothing to compare with these figures in the United States or in Mexico. The limit in stamps seems to be 1,250 lb., and this weight is not common. The favourite weight is 1,050 lb. when new, with an average running weight of 1,000 lb. The nearest thing to the capacity figures quoted above is the record of the Mexico mill at El Oro, Mexico, of 7 tons per stamp, though this is not a fair comparison, as the battery screens at El Oro are 6 mesh, while in Africa they are 30 to 40 mesh. And this is the best in Mexican practice. The average tonnage may be stated as 3·5 tons and the worst, with 1,000-lb. stamps and 40-mesh screen is 2·5 tons. Allis-Chalmers company has furnished 1,600-lb. stamps for Africa, but of the many hundred stamp mills they have supplied, in North and South America, none have exceeded 1,250 lb.

On the other hand, a recent exhaustive report of A. Heymann on the metallurgical practice of the Midas Deep, Ltd., says among other things: 'A sample was also taken from the slime dam of the current slimes, which, assayed at my laboratory, gave a result of 2·5 dwt. This would not pay to treat on a small scale, and would only be payable with a very considerable increase in the size of the battery, and consequent increased tonnage.' One gathers from this statement that the slime from this 20-stamp mill, which consists of 35 tons per day of $2.50 slime is not treated because it will not pay! What plant here would throw away such an asset in the form of an easily treated slime? An American or Mexican cyanider, of a speculative turn of mind, apparently could make a fortune cyaniding slime ponds in Africa.

Ways of preventing losses.—A feature of construction, which is deserving of great consideration and which has received little attention in the technical press, is the common or garden variety of rim launder. The ordinary iron launder as used for cyanide tanks need cost at most 75c. per running foot. If I were building a plant with the expectation of having it operated by the best three shiftmen you know, I should expect an occasional forgetfulness or oversight and should provide for it. Every tank in a cyanide plant which may contain pulp or solution should have a rim launder. Formerly it was considered advisable to cement the floor of the entire plant to provide for such accidents. But such cement floor is advisable even with rim launders, if for nothing but to prevent leakage from faulty construction of plant. Even the tailings-pond floor should be cemented. All these extra launders and cemented floors should conduct the 'accidental overflows' to a sump tank specially provided. The least experienced plant operator knows that this pays for itself in short order. It can be easily understood that if the night shift forgets and allows a number of tons of solution to overflow, there will be no boasting about it in the morning. But when the 'accident sump' catches this overflow, even though the pumping back is not reported, the loss is prevented, which is the main thing.

In this connection, it is amazing what losses some companies will stand, when the simple expedient of returning the tailings-pond drainage will pay handsomely. One plant manager in Mexico returns settled solution from the top of a slime dam and obtains 500 pesos daily. In this case, the query is: What amount of money soaks into the ground? What amount could be obtained by tunnelling under the pond and catching this drainage?

Another company sends its tailings into a small lake, pumping back and securing many thousands of dollars from the surface of the lake. Where sand leaching is practised, charges of sand are more liable to go out without thorough washing than are slime charges with modern methods of slime washing. At one mill, nepotism requires the employment of a young man who is better able to cope with his cups than with classifiers. This often results (particularly on night shift) in slimy charges of sand and consequent utter inability to treat or wash. It is a notorious fact that the drainage from the pond of this plant, except during the short period of sluicing out the sand tanks, averages $1.655 per ton. The point I wish to make is that, if not from one cause, then from another, accidents happen in the plant and that, so far as possible, provision should be made against resulting losses.

Tube Mill Practice.—Tube mills are being improved daily. It is but a step from one man-hole in the shell centre to two at opposite ends and on opposite sides, then another from silex set flat in cement, to edgewise setting, to iron or steel plates, and from these to the honey-comb and ribbed lining. Where formerly linings lasted three months and required three weeks for renewal, the life is now twelve months and replacements are made in two eight-hour shifts. At first the mills were filled and charged laboriously by removing a single man-hole and dumping in sacks of pebbles. Then convenient bins were built over the mills to chute the pebbles rapidly into the mill and thus save time in this slow and time-wasteful operation. Now such a scheme as a fixed spiral in the discharge opening, or a feeder large enough to pass the pebbles into the feed end of the mill, has magically removed such troubles, and tube mills are run 97% of the possible running time.

The modern operator has gone even further and is dispensing altogether with flint pebbles. One Mexican company had several car loads of pebbles at Tampico for several months which were for sale cheap. In other words, it was not worth while to haul them to the mine. It is an interesting fact that while pebbles are consumed at the rate of from 1 to 2 lb. per ton of sand ground, under similar circumstances, from 15 to 20 lb. of quartz lumps are required. With man-hole charging it would have been difficult, expensive and tedious to have supplied such a quantity of grinders to the mill, but with the simple methods of feeding now being adopted this is not difficult. So the cost of the pebbles is saved and incidentally the capacity of the plant is increased about 1%.

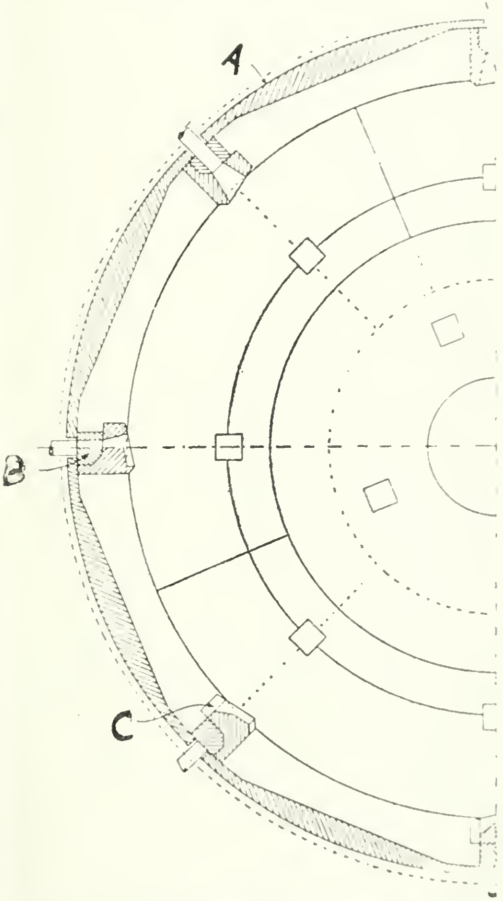
A ratio of one tube mill to 20 stamps is considered crowding in Africa. At the Waihi Grand Junction 40 stamps are followed by four tube mills, and six more are to be added without increasing the number of stamps. The remarks of the chairman of the company are significant. He says: 'The addition of these six tubes together with the subsidiary plant will enable from 350 to 400 tons to be treated daily without the erection of any further stamps, our experience being that it is much more economical, both from the view-point of capital expenditure and working costs, to reduce the ore by tube mills than by stamps.'

Precipitation Practice. - In precipitation, zinc shavings are favourites at long odds, though zinc dust is being successfully used where Mr. Merrill has had personal control of its use, as well as at Mercur, where for a while a change to shavings was considered, but finally given up. In treating the precipitate, sulphuric acid has fallen into disfavour and no wonder. It has been found that passing the precipitate through 40 to 60-mesh screens is all that is required for producing a reasonable grade of bullion, and that by arranging a set of shallow trays in one compartment, or, better, in a spare precipitation box, the 'shorts' can be made to serve for precipitation of solutions.

The tilting furnace wins friends wherever tried. A new one recently put on the market is simplicity itself. I have seen a No. 60 crucible heated to 900° C. (a temperature which is well above that necessary for cyanide bullion) in less than one hour. The fuel is coal oil, distillate or crude oil, and no pressure or blast is required, natural draft alone being used. I neglected to say that the above temperature was attained, starting with a cold furnace and crucible. This is not an advertisement but an appreciation, and I should be glad to supply the name of the manufacturer to any one interested.

Where we are weak in this branch of metallurgy is in our method of slime settlement. We need a genius to prove that slime is actually as heavy as sand by settling it as rapidly, with no thought of ions or electrons."—MARK R. LAMB.—*Mines and Methods*, Oct., 1909, p. 73. (H. A. W.)

TUBE-MILL LINERS. "F. C. Brown, superintendent for the Waihi Grand Junction and Komata Reefs, describing the tube-mill liners invented by him, which have been continuously in use since 1906, is now able to give details of their work, wear and cost. The accompanying illustration shows a half cross-section of the liners. The longitudinal plates A are of hard steel and are thickest in the middle, where the wear is greatest. The longitudinal shoes C, also of hard steel, serve to raise the pebbles and ore and throw them down. The bars B, shoes C, and plates A are held in position against the inside of the tube-mill by means of bolts. The bars B are not subject to wear and are made of tough iron. The ribs are arranged in such a way that there is no slip of pebbles on the liners and the correct rate of revolution has to be determined, so that the cascade shall be well regulated and the pebbles not thrown too far. For a 4 ft. mill the correct rate is 27½ revolutions per minute, and for a 4 ft. 6 in. mill, 24 revolutions give the best result. In a 4 ft. mill there are eight ribs, and in a 5 ft. mill ten ribs. Each rib for a 4 ft. mill measures 3 by 2½ in., and for a 5 ft. mill 3½ by 3 in. It will be seen that this system of liners is much thinner than silex liners, and in a 4 or 5 ft. mill the working surface is thereby made nearly 20% greater than in a mill of the same outside diameter lined with silex. The ore at the Komata Reefs is very hard. It is coarsely crushed in the stamp-mill, the woven-wire screen having only five apertures per linear inch. There are two tube-mills in operation, each is 16 ft. by 4 ft., and they treat between them about 100 tons per day. The grade of the feed is: Retained on 6-mesh, 5%; 40-mesh, 46%; 90-mesh, 26½%; 200-mesh, 8½%; through 200-mesh, 14%. After grinding, 60% passed 200-mesh. The results during 1908 showed that the consumption of flints was 2·38 lb. per ton of ore and the cost, at 0·47d. per lb., was 1·11d. The consumption of steel in the liners, at 16s. per cwt., was 0·82d. per ton of ore. The total cost of wear was therefore 1·93d. per ton. It was found that a set of liners will last about 22 months. At the Waihi Grand Junction there are four tube-mills, having a total capacity of 205 tons per day. Two measure 19 ft. 1 in. by 4 ft. 8 in. internal diameter, one measures 16 ft. by 4 ft. 3 in., and one 12 ft. 6 in. by 4 ft. 3 in. The grade of the charge during 1908 was: Retained on 10-mesh, 6·2%; 20-mesh, 5·6%; 40-mesh, 19·8%; 60-mesh, 8·5%; 80-mesh, 7%; 100-mesh, 5·4%; 150-mesh, 10·3%; 200 mesh, 3·0%; passed 200-mesh, 34·2%. After leaving the tube-mills the grade was: Retained on 100-mesh, 0·6%; 150-mesh, 6·7%; 200-mesh, 5·7%; passed 200-mesh, 87%. The flints consumed cost 1d. per ton of ore; and the liners, 0·92d.; a total of 1·92d. In order to get efficient reduction, return-



ing-boxes in the nature of spitzkasten are employed at both the Komata Reefs and the Waihi Grand Junction, so that underground particles in the discharge are returned to the feed."—*New Zealand Mines Record*, April, 1909.—*Mining Magazine*, Oct., 1909, p. 153. (J. A. W.)

Committee on Fuel.—"The appointment of the committee on coal is of such recent date that it has not had time to much more than make a hurried, general survey of the field, and to make preparations for gathering together literature on the subject.

It may be asserted that in spite of the really enormous amount of literature published on coal, less is known chemically of this commodity than of any other single substance. Fully 75% of the literature consists of repetition of stock tests or experiments. For example: the literature appertaining to the economic combustion of coal in boilers would, if collected together, weigh many tons. The published analyses of coal amount to many thousands, but when it comes to the chemistry of coal, the literature might be put into a very small space, and of this at least 90% consists of inconclusive experiments which practically prove nothing. No scientific mind of the first class has ever seriously attempted to work out the chemistry of coal.

However, within the last few years, there is a distinct tendency to get out of the rut that previous experimenters have been travelling, and it seems as if the question of the scientific examination of coal is becoming recognised among scientific men as worthy of their attention. In view of the fact that considerably less than 10% of the coal produced becomes avalable for useful work, it would certainly appear as if the study of coal is one of the most important scientific and economic questions of the day.

The first work that your committee proposes to undertake and, in fact, has already commenced, is to gather together as far as practicable all the literature relative to coal and its use. To abstract this literature will be its first task. In doing this work the needs of the western chemists will be kept strictly in mind, and while some attention will be paid to foreign and eastern coal, it will be more for the purpose of being able to institute comparisons between those coals and those found here in the West.

The next work will be to make investigations into the methods of making coal analyses. The committee has obtained the report of the Committee on Coal Analyses appointed by the American Chemical Society, and has subjected it to a rather close examination. This report was made a number of years ago, and has been accepted as a standard by chemists all over the United States. Fresh examination of the report, however, demonstrates that even in the minds of the chemists of the American Chemical Society a large number of questions were left open, and some of these questions the committee proposes to investigate. We will concern ourselves, however, chiefly with the methods of sampling, determination of moisture, determination of volatile gases, and will also pay some attention to the question of occluded gases, with which subject the report of the American Chemical Society did not deal.

Sampling.—It appears to us that the method of sampling recommended by the American Chemical Society is open to criticism. The method recommended gives hardly much more than a grab sample. If coal is to be purchased on the basis of British Thermal Units contained, the samples obtained must be above suspicion as to their correctness. Your committee will attempt to sample coals by several different methods, and will subject these methods to careful examination.

Moisture.—The official method of determining moisture is by drying the powdered coal in a toluene bath. Toluene boils at 113° C. at a barometric pressure of 30″. The boiling point at this altitude must be considerably less, and it is questionable in the minds of your committee whether a sufficient temperature can be obtained by the toluene bath to eliminate the moisture. We will introduce a comparison of the various methods.

Volatile Gases.—The determination of volatile gases recommended by the Committee of the American Chemical Society is the fractional distillation of one gram of powdered coal over the full flame of a Bunsen burner for seven minutes. At this altitude the distillation of low-boiling-point tarry matter must go on much more rapidly than at sea level. The violent distillation in the opinion of your committee is apt to entail a considerble mechanical loss of coal and probably the distillation of high boiling point hydrocarbons, not distillable at sea level. Moreover, the great majority of western coals are neither bituminous nor lignite and are non-coking in character. These coals have been officially named subbituminous. Such coals are much more likely to dust than bituminous coking coals, and we propose to determine whether the mechanical loss is not more serious than the committee of the American Chemical Society was inclined to believe. It may be pointed out here that this question of the testing of non-coking coals is as yet an open one.

Occluded Gases.—Of course, all chemists are familiar with the fact that a great deal of gas contained in the coal is occluded. Recent tests, however, go to show that a considerable volume of the gases is contained in fractures or fissures in the coal. This brings up the question whether samples of such coals, after being crushed and kept for a short time at the low atmospheric pressures which prevail at these altitudes, do not suffer a considerable loss. This is a question which your committee will endeavour to determine.

It is well known that coal suffers considerable deterioration by keeping. This is true, even at sea level. At some of the coaling stations of the British Government coal is stored under water, and it is claimed that by so doing the coal suffers less deterioration than if stored in air. It occurs to your committee that by getting into communication with the railroad companies it is possible that some data might be obtained of interest along these lines. At altitudes of 5,000 to 10,000 feet the losses of occluded gases would seem to be greater than at sea level.

During the last few months the United States Geological Survey has published a great deal of literature on a number of coal fields of which very little is known, even by western men. Some of these new coal fields are of extreme technological interest. It might be interesting to note, as showing the enormous importance of these coal fields, that the Book Cliff and White River fields contain an aggregate of twenty-two billion tons of coal, and it is probable, in view of the little-known geological conditions of these fields, that they may contain many times this amount, but taking the present rough estimates, these two fields alone contain sufficient coal to supply the entire United States with coal for the next fifty years at its present rate of consumption, and these are but two of the coal fields out of the scores yet to be investigated in the West.

Your committee reports that it has secured the co-operation of the city of Denver boiler inspector for the purpose of investigating the industrial and domestic use of coal in cities. Your committee has proffered its aid to those bodies interested in the mitigation of the smoke nuisance. Your committee will endeavour also to secure the co-operation of those engineers in charge of the large office buildings of Denver, and will endeavour to secure data on the character of the various steam equipments at the different buildings, data relative to the coal used and the conditions and character of the waste gases sent into the atmosphere above the city.

Your committee wishes to state that as a matter of convenience it will devote its main attention at this time to coals mined in the State of Colorada. This course is not only more convenient, but inasmuch as our primary investigation goes to show that the State of Colorada contains every class of coal from anthracite to a low-grade lignite, it believes that the problems solved here will be of general application all through the West. The committee states, however, that it will not confine its tests merely to Colorado coals.

The question of the artificial modification of coal is one of extreme interest, and considerable work has been done along these lines during the last four or five years. Besides its heating value, coal contains a great many other valuable properties. It is possible that the process of modification of coal will result in giving a product of more economic value as a fuel than coal, besides giving by-products of value to chemical manufacturers.

While this point of view is probably too advanced for conditions as they are in the West to-day, your committee believes that conditions are changing so rapidly that some attention should be given to this phase of the subject.

Some very excellent work recently has been done in the Experimental Station of the University of Illinois. This work deals with the enriching of producer gas, or rather the co-efficients of the velocity of conversion of carbon-dioxide into carbon-monoxide. This work is of peculiar interest, inasmuch as it for the first time takes into account the work of Van't Hoff, and deals with the transformation from a point of view of the velocity of chemical reactions and the influence of mass. This piece of work is of extreme interest, and is a very fine example of technological chemical research.

Your committee appreciates the fact that it has undertaken a very large amount of work, but it is of such extreme interest that it will be a pleasure and not a task. It invites the co-operation of the members of the society, not only here, but in other States.—A. W. Warwick.—*Western Chemist and Metallurgist*, Oct. 1909, p. 389. (H.A.W.)

The City Deep Mill. "In the ordinary way of things the cam shaft, subject as it is to the destroying action of incessant vibration at the same time that it is carrying an irregularly distributed load, which ranges between three and four tons, usually breaks down in about six months or so. With the increasing weight of stamps which is the vogue to-day, and which does not yet appear to have reached the limits of finality, this unpleasant habit of the cam shaft is a matter of serious import to the mechanical engineer; it is, in fact, the chief obstacle to the building of the very heavy stamps which are necessary in order to obtain the general milling efficiency which Rand engineers are aiming at.

The arrangement is briefly this. Instead of the usual heavy timber king-posts upon each side of the mortar-boxes for ten stamps, there are built up from the concrete blocks upon which the boxes rest reinforced concrete piers, about 6 ft. across the base, 7 ft. high, 4 ft. across the top, and 14 in. wide. Upon this is set a single steel casting, held down to the blocks by six 1¾ in. rods, 14 ft. long. These bolts are sufficiently long to give the whole the required elasticity, and to complete the necessary resilience the casting is supported upon the concrete piers by wooden blocks, 14 in. × 12 in. × 4 ft. In addition to the guide frames, the casting carries a fish-bellied girder of cast steel, upon which intermediate bearings 5 in. long are set between the cams. The cam shaft, therefore, is supported at every few inches of its total length, instead of, as usually, at the ends only. It is obvious that a structure of this kind is capable of supporting stamps of enormous weight, and instead of a life of several months only for the cam shaft one foresees a life of several years. There appears to be no reason why stamps should not be weighted up to a couple of tons if necessary. The guaranteed weight of the City Deep stamps is 2,000 lb., and the minimum crushing duty at the start of the proposed plant is fixed at 11 tons. With the same weight of stamps and a suitable mesh there is no reason why the crushing capacity per stamp should not rise to 20 tons per stamp.

Long Stamp Heads.—Another feature of the battery will be the length of the heads. These are about 46 in. long and 9½ in. in diameter, and, together with the shoe, will have a total length of 60 in. Since they will project considerably above the top of the mortar-box, wooden guides to receive them will be fitted right into the top of the box. All the pulp from the mill will be taken directly from the boxes through launders to the tubes, except that percentage which is already sufficiently fine, of course."—*The South African Mining Journal*, Sept. 11, 1909, p. 771. (A. R.)

Electrical Smelting of Iron Ore in Sweden.—"'Bergverksnyt,' referring to the operations conducted for three months during last summer, and the committee appointed by Government to report on the process, have come to the conclusion that the process is one of great importance, especially for countries where water power is cheap and freight easy. According to the committee, the two most important economical factors are: (1) For iron ore carrying 60 per cent. or more iron, about ⅓ h.p. is required per ton of iron; (2) about 300 kgm. of coke is necessary for every ton of iron."—*London Mining Journal*, Oct. 23, 1909, p. 120. (A. R.)

Mercury in Italy.—"According to a report by Mr. Nicou, the columnar deposits of cinnabar in the Monte Amiata district contain 1 per cent. mineral (60 per cent. mercury) (*La Metallurgie*, 21st April, 1909). The ore is screened, and all the small fragments less than 1½ in. are roasted in Cermak furnaces. The chief mine is that of Abbadia San Salvatore, which turns out about 25,000 tons of 1 per cent. ore yearly. Altogether, the number of furnaces running is 28, among which are seven Cermak furnaces with a capacity of 24 to 30 tons, four of

12 to 15 tons' capacity, four of 2 to 8 tons' capacity, and 13 ordinary 6-ton furnaces. The average recovery from a large furnace is 95·2 per cent. of the total mercury contained in the ore. The latter costs 13s. 6d. to 22s. 6d. per ton at the mine, and the cost of classification, conveyance, and treatment is 3s. 3d. per ton, making the total cost 16s. 9d. to 25s. 9d. for the $12\frac{3}{4}$ lb. of mercury recovered, or 1s. 4d. to 2s. per lb. Although the quantity of the ore is diminishing, the use of improved appliances enables the output of mercury to be maintained, and even increased, so that Italy contributes nearly 400 tons out of a total of about 3,600 tons produced throughout the world."—(H. A. W.)

TREATMENT OF ORES OF PRECIOUS METALS.—[*] "The pulverized ore is submitted to the action of a solution containing cyanide, a soluble iodide, and a substance capable of producing nascent oxygen in presence of the mixture, such as a soluble bisulphate,[†] ozone, or nitrogen peroxide. The process is hindered if the solution be too strongly acid, but goes forward in feebly acid solutions. The most suitable proportions of the constituents must be determined by preliminary trials, but in practice the mixture may be composed as follows:—Potassium cyanide, 450 grms.; potassium iodide, 225 grms.; sodium bisulphate, 1350 grms.; lime or soda, 225 grms.; the whole dissolved in 900 kilos. of water, and employed in the proportion of two parts of the solution to one of ore. The same solution may be used over and over again, provided the strength in cyanide and bisulphate be maintained constant. No further addition of iodide is necessary, the iodine being regenerated by the bisulphate added from time to time. Between each operation it is desirable to pass the liquid over the zinc shavings in order to extract the dissolved metals. The bisulphate may be replaced by ozone or by nitrogen peroxide, the continuous regeneration of the iodine proceeding in an exactly analogous manner. Refractory ores containing reducing agents (such as iron sulphides) or tellurium may be treated at once by the solution, no preliminary roasting being necessary."—*Clancy Metals Process Co.*, Fr. Pat., 400-700, March 13, 1909.—*Journal of the Society of Chemical Industry*, October 15, 1909, p. 1044. (W. A. C.)

MINING.

ELECTRICALLY-DRIVEN WINDING ENGINES.—"One of the essential features of the flywheel storage arrangement for electrically-driven winding engines is the automatic arrangement for securing the fall in speed of the motor and flywheel when the maximum power is being taken. It is at the moment of this fall in speed that the energy of the flywheel is given up to the load, and thus the power taken from the generating plant is equalised. Immediately the load comes off the flywheel regains its energy by speeding up again, and when the maxima and minima come near together the action has to be quick.

With a direct-current motor the fall in speed is easily arranged for, by fitting it with a compound winding, the series or compounding coils being so connected that when current is going through them they magnetise the poles in the same direction as the shunt coils, or in other words, are cumulative in action. The increase of magnetism of the poles causes the speed to fall in just the same way as arranging the series and shunt coils in opposition would give an increase of speed by reason of the field being weakened.

With a three-phase motor the problem is not so simple, although the motor itself is a simpler piece of mechanism than the direct-current motor. The reason for this is that a three-phase motor is essentially a constant-speed machine, and to get the speed down, to give it 'slip' as it is called, a resistance has to be inserted in the rotor circuit. Of course, such a resistance means that power will be lost when current passes through it, and this power will be proportional to the 'slip.' Therefore in order to keep up the efficiency of the combination the 'slip' should be as low as possible. This means that a heavier flywheel is required than with the direct-current motor set, for it has to give up the same amount of energy with a smaller drop of speed. It should be remembered that the stored energy of a flywheel is proportional to the square of the speed at which it runs. The direct-current motor having a compound winding has no loss of power when the fall in speed takes place, and other things being equal should therefore be the better arrangement. But that is just the point, there are other things which are not equal.

Of course, 'slip' resistances may be made to go in and out automatically by means of an extra piece of apparatus, so that it is only connected in the rotor circuit when the load, and therefore the current, exceeds a certain amount. A good deal depends on the time taken by maximum load. A long-period maximum means a much heavier flywheel, and there comes a time when a flywheel does no good and may as well be omitted.

It will be seen from the above that there are points for and against both direct-current and three-phase for winding engines. These can only be thoroughly appreciated by a trained electrical engineer, and colliery officials and steelworks managers who have a mind to buy such plant on their own responsibility are dealing with a problem that is not at all simple."—E. KILBURN SCOTT.—*Iron and Coal Trades Review*, Sept. 10, 1909, p. 389. (A. R.)

HYDRAULIC STOPE FILLING AT THE ROBINSON MINE.—"A launder 6 in. × 6 in. in size, is laid from the foot of the dump to the top of a pass, which has been carried down to the 9th level, and which was holed through with this object in view as far back as December last. The launder is graded at about 15° at the top of the pass, where it connects with another of similar dimensions which runs down into the mine. A small quantity of water, sufficient to keep the sand moving easily, is supplied at the dump, and occasionally a little lime is added to neutralise the acidity of the drainings below. At the top of the stope to be filled, or wherever necessary, a further supply of mine water is added as required in order to facilitate the distribution of the sand in the stope. To retain the material in place, the back of the level is packed first of all with rock of ordinary walling size, and, following this, with smaller rubble which is covered finally with old cocoa-nut matting, the whole making a kind of filter, through which the water readily drains away. Only a small percentage of slime finds its way through the matting. The mass quickly settles as a close and compact filling, which may be relied upon to afford solid support at every point to the roof above it. The average dip of the reef, it may be mentioned, is from 40° to 50°. So far only about 800 tons of sand have been packed in the mine, but owing to the fact that the Main Reef, which lies some 2 ft. or so below the Main Reef Leader, has yet to be worked upon a considerable scale, the sand filling process seems likely, sooner or later, to be brought into large requisition. Taking the Main Reef Leader at an average thickness of 6 ft. to 7 ft., the footwall parting at 2 ft., and the Main Reef below at, say, 6 ft. or 7 ft., the resulting distance from wall to wall when the whole is broken away will amount to something like 15 ft or 20 ft. Later on the sand filling method will probably be extended to the lower levels of the mine in such a way that the Main Reef may be taken out below the old Main Reef Leader stopes, commencing from the bottom and going upwards in much the same way that the faces are carried in coal seams."—*The South African Mining Journal*, Sept. 11, 1909, p. 773. (A. R.)

---

[*] THOS. B. CROWE: Researches upon the Telluride Gold Ores of Cripple Creek (Colorado). See this *Journal*, Vol. ix., May, 1909, p. 398.

[†] This should be *persulphate*.—ED. COM.

NOTES ON DIFFERENT METHODS OF STOPING.—"In dealing with this subject the intention is not to write a detailed description of the various methods in vogue on the Kalgoorlie Field, but rather to give a brief outline of three of the methods which are most universally in force, and from which, ultimately, a few comparisons may be drawn.

The three methods which the writer proposes embodying in this article are known as 'The Rill,' 'The Flatback,' and 'The Shrinkage.' For comparative purposes each method will be applied to a block of ore 500 ft. in length, and opened up by levels 200 ft. apart.

The Rill System.—It is necessary in this system, for facilitating mullocking, to make connections between levels by rising and winzing at about every 100 ft. or 150 ft. along the ore body. After the leading stope is beaten off for the full length of the drive, the level is timbered. Single stulls are put in up to a width of 14 ft. If the span is too great for stulls, set timbers or saddleback stulls are used. Stulls are lagged over with 4 in. gimlet poles, and wooden chutes are built at intervals of 50 ft. Ore passes are cribbed up through the filling directly above each chute, and are constructed of 7 in. round timber or 9 in. by 3 in. square timber, the passes measuring 4 ft. × 4 ft. in the clear. Occasionally a double pass may be carried up, one side serving as a travelling way.

Stoping is commenced off the winzes, and rills are gradually formed, one on either end of each winze. The rills are generally at an angle of 37° off the horizontal, this allowing for the boring of all waterholes. After the rill is beaten through, and the ore, which gravitates down the rill to the ore pass, is cleaned up, machines are transferred to another portion of the stope, ore passes built up, and filling is proceeded with. The filling, which usually consists of residues, is dumped down the winzes from the level above and gravitates down the rill into position, although at the extreme points of rills a certain amount of handling is required.

'*The Flatback' System* is similar to the rill, the difference being that the stope is carried horizontally, and not rilled as in the previous method. As the ore does not gravitate, but lies where it is broken, it is necessary to have ore passes at shorter intervals. Filling is also more expensive on account of the extra handling required. This method is usually put in force where the ground is heavy, and bulk heads, etc., are required to keep up the back or walls, as the flat bed of mullock forms a solid foundation for the bulks.

'*The Shrinkage' or 'Lay' System.*—In this system one connection between levels at about the centre of the block is all that is needed. The level is timbered similarly to the 'Rill' system, except that the chutes, which are put in every 20 ft., are constructed of ⅜ in. steel. These chutes practically consist of a steel lip protruding into the level, the bottom of the chute being formed by the broken ore itself. Any rocks which are too large to pass through the opening can be blasted without damaging the chute. The stope is rilled at an angle of 23°, all waterholes being bored. Sufficient of the broken ore, approximately 44%, is drawn off to allow comfortable working room between the back of the stope and the top of the broken ore, which acts as filling until the stope is completed. Two ladderways are cribbed up through the ore, one near either end of stope, built of 6 in. half-joggled logs, and measuring 3 ft. × 3 ft. in the clear. These manways are commenced between two chutes, so that the shrinkage of the ore directly over the chute will not affect them. When blasting, only a few holes are fired in a round, so as to allow the large rocks to be broken down sufficiently small to pass through the chutes.

After the stope is beaten through to level above, all ladders and air pipes are removed from passes, and the balance of the ore remaining in the stope is drawn off. The passes then collapse, and the timbers are regained through the chutes, to be utilised in other places. Filling is then a very simple matter, as, the stope being open, the mullock may be dumped at whatever point required.

Where conditions permit of the satisfactory working of the Shrinkage system the writer certainly prefers it. In the Rill system, in opening up the block of ore, three or four winzes are essential to economical mining, whilst in the Shrinkage system one winze is sufficient. A considerable saving in the cost of development is thus effected. Expensive ore passes, which have to be installed and maintained in the Rill system, are entirely dispensed with in the Shrinkage system. Loss of time, experienced in shifting machines preparatory to filling, is avoided, because in the Shrinkage system the stope is not filled until after it is beaten through and the ore drawn off. In the Rill system a certain percentage of ore, no matter what care is exercised in cleaning up rills, is lost—embedded in the filling—whilst in the Shrinkage system a certain amount of low-grade material scales off the walls and reduces the grade slightly. This can be reduced to a minimum by strict supervision in the breaking of the stope.

There are certain conditions under which the Shrinkage system, if applied, would be unsuccessful, such as in flat veins or in stopes where the walls and backs are bad; but where the lode is at all nearly vertical and the walls fairly good the writer is satisfied that it is a cheaper and just as satisfactory a method as the Rill system. In isolated patches of ore, which probably would not be worth considering if a lot of dead work were necessary to deliver the mullock in the stope, this system can be used to advantage."—J. CHEFFIRS.—*Queensland Government Mining Journal*, Sept. 15, 1909, p. 443. (F. A.)

MISCELLANEOUS.

ANALYSES OF COLONIAL OATS, AND THE DETERMINATION OF THEIR PHOSPHORUS CONTENT.—"The percentage weight of kernel in the oat-grains from different districts in Cape Colony varied from

54·3 to 71·1, the highest figure being that given by a fine sample probably grown from Canadian seed. The author gives the following average results of his analyses of oats from various districts of Cape Colony, as specified in the table, together with the composition of oats from other countries for purposes of comparison :

District.	Number of Samples.	Moisture.	Fat.	Proteins.	Digestible Carbohydrates.	Crude fibre.	P_2O_5	CaO.
Piquetberg	6	10·37	6·12	9·92	59·45	10·80	0·57	0·093
Malmesbury	6	9·50	6·16	9·56	61·62	9·92	0·48	0·091
Cape	2	10·13	6·16	8·00	62·06	10·00	0·46	0·108
Stellenbosch	1	10·35	5·85	8·06	62·35	10·05	0·50	0·118
Caledon	6	9·78	6·58	8·45	60·66	10·81	0·46	0·095
Bredasdorp	6	10·48	6·63	8·50	61·48	9·43	0·47	0·102
Robertson	5	9·66	6·08	8·85	60·52	10·94	0·67	0·112
Alexandria	3	10·05	5·79	9·79	60·26	10·59	0·57	0·107
King William's Town	2	8·98	6·31	9·39	62·19	9·71	0·48	0·086
Cathcart	10	9·59	5·56	10·50	60·55	10·09	0·62	0·104
Queenstown	5	9·69	5·33	11·09	60·20	9·51	0·76	0·120
Average for Cape Colony ..	52	9·85	6·03	9·53	60·77	10·20	0·56	0·102
Middle and North Germany ..	103	12·45	5.30	10·82	58·23	10·25	—	—
South and West Germany ..	44	13·39	5·30	11·36	58·12	9·93	—	—
Austria-Hungary	34	11·83	5·84	11·41	56·40	11·01	—	—
France	196	13·50	5·46	9·52	60·47	9·18	—	—
England and Scotland ..	16	13·74	6·15	13·05	53·16	11·89	—	—
America	22	10·67	4·96	11·26	59·35	9·33	—	—
(Various sources) ..	(Large)	13·30	4·77	10·32	58·19	10·32	0·69	0·100

From these results it is evident that Colonial oats are equal in value as regards organic food material to those of other countries, but their content of phosphoric oxide is somewhat lower."—J. LEWIS, *Cape of Good Hope Department of Agriculture, Bull. 35,* 1908.—*The Analyst,* Feb., 1909, pp. 55-56. (E. H. C.)

HOW MUCH WOOD IN A CORD?—" School arithmetics say that a cord of wood is 128 cubic feet, or the contents of a pile 8 feet long, 4 feet high, and 4 feet wide. Wood is marketed on this basis. A pile whose length, breadth, and height multiplied together gives this number of cubic feet fills this requirement, no matter whether the sticks are long or short, straight or crooked, round or split, unless there is an understanding to the contrary. The cubic feet of solid wood in a cord, however, may vary widely according to conditions.

The following table compiled from a publication of the United States Department of Agriculture, Forest Service, shows how much difference it makes whether the wood is cut long or short, chopped or sawed, and whether the measurements are to be made while the wood is green or after it is seasoned :—

Solid Wood in One Cord, or a Pile Measuring 128 Cubic Feet.

	Cubic Feet.
Hardwood, 1-foot lengths.	85.0
Hardwood, 2-foot lengths...	84.0
Hardwood, 3-foot lengths...	83.5
Hardwood, 4-foot lengths	83.0
Hardwood, 12-foot lengths	73.0
Conifers, soft wood	90 to 96

Losses.	*Per Cent.*
Loss from sawdust in sawing to 4-foot lengths	.5
Loss from chips in chopping 6-inch sticks to 4-foot lengths	6 to 8
12-foot hardwood cut to 4-foot lengths, loses by closer piling (conifers being straighter lose somewhat less)	12
Green wood loses in bulk from seasoning ...	7 to 11
Green wood loses in weight from seasoning approximately	50

A cord of large diameter logs has a greater content of solid wood than a cord of smaller or split logs."—*Mines and Minerals,* October, 1909, p. 140. (A.R.)

MINING REPORTS.—" Whereas, the over-valuation of mining properties by investors and the public, due to ignorance of mining conditions and a lack of the real nature of the investment, tends to increase unduly the profits of mine promoters and speculators, and to increase unnecessarily the financial risks taken by mine investors, to the ultimate disadvantage of the mining industry :

Resolved, that it is the opinion of the Mining and Metallurgical Society of America that, for the protection of shareholders and investors, every mining company should publish an annual report within 90 days of the close of its fiscal year, and such report should incorporate the following information :—

1. A brief review of the past history of the property, the work accomplished, and the results obtained, with tabulated statement of expenditure and receipts from the beginning, marketable products made each year, and the sums received from the sale of same, the annual net earnings and the disposition made of such earnings.

2. A similar review, but in more detail, of the work of the year, with statements of the assets and liabilities (these statements to show all details as to capitalisation of the company ; the number and classes of shares outstanding at date of the report ; the respective rights of these shares ; the number of shares remaining in the treasury ; any options or contracts on such shares ; any bonded indebtedness), receipts and disbursements, cost sheet and other information as to work accomplished and results obtained.

3. A statement of ore reserves at the date of the report, compared with the reserves of the previous year, with an estimate, by competent authority, of the probable life of the mine."—*S.A. Mining Journal,* October 30, 1909, p. 194. (A. R.).

SOLDERING PLATINUM.—" Soldering ordinary platinum vessels is ordinarily done with platinum itself as the soldering material, or with pure gold ; using the oxyhydrogen blow-pipe or its equivalent melting flame ; but the latter has the great disadvantage that it melts at a lower temperature than the platinum, and under temperatures to which crucibles or other vessels may be subjected, it melts and leaves the crack open.

In the *Central-Zeitung für Optik und Mechanik* it is recommended to close the crack with a mixture of platinum powder and turpentine oil, then to heat the joint to a white glow, and in this condition to hammer all together. Another method, applicable where the crack is on the edge, consists in hanging a narrow strip of platinum sheet over the crack, then placing the article in the fire, heating it white hot, and then hammering if necessary.

According to the *Metall-Arbeiter* a small hole may be closed by a rivet made out of platinum wire, welded after hammering both heads closely. For larger holes a patch is made of platinum sheet, riveted on, and then welded."—*Metal Industry*, Sept. 30, 1909. (E. H. C.)

APPLICATION OF ELECTRICITY TO AGRICULTURE. —"It might be of interest to briefly glance at a new development in the application of electricity for the promotion of the growth of plants. The discovery that electrification of the atmosphere immediately above the plant stimulates in certain cases its growth, is already thirty years old, but only recently has this idea been practically developed—first by Professor Lemström in Sweden, and more recently still by Sir Oliver Lodge, in collaboration with Mr. J. E. Newman and Mr. R. Bomford, on an experimental farm at Bitton, Glos., and on a farm of about 40 acres at Evesham, near Birmingham, and at other places. A network of galvanised iron wires is stretched over the field to be treated, and suspended 18 ft. from the ground from wooden posts and oil insulators. The posts are placed 70 yds. apart, so that about one post per acre is required. The network consists of No. 11 S.W.G. wire placed over the posts in parallel rows 100 yds. apart, and No. 20 S. W.G. wire cross-connecting the stouter wires every 10 yds. The attachment of the latter to the insulators is not direct, but by means of short links and tension insulators to reduce the electrical stress on the insulator itself. The network is positively electrified to from 60,000 to 100,000 volts by means of an induction-coil mercury gas break and Lodge rectifying vacuum valves. The induction coil is worked on the primary side by continuous current obtained from an ordinary dynamo. Mr. Lionel Lodge, who is the electrical expert for the various installations, informs me that there is no difficulty in keeping the potential in the network up to the figures given except in misty weather or during rain, when it may fall as low as 20,000 volts; also that the amount of primary power required per acre is very small, namely, from 10 to 20 watts. The installation is run for five or six months during eight to ten hours each day, and the total expenditure of energy is only about 20 B.O.T. units per annum per acre. Under this treatment the increase in the yield per acre is about 30%, but under certain conditions it may be even more. The system is in use on several farms in this country, on six farms in Germany, and on one farm in Holland."—Dr. GISBERT KAPP, *Presidential Address, Inst. of Elect. Engineers*, Nov. 11, 1909.—*Electrical Engineer*, Nov. 19, 1909, p. 719. (J. A. W.)

METHODS OF RESUSCITATION.—"A method of resuscitation, applicable equally to cases of electric shock, "gassing," and drowning, which is said to be simpler and more effective than those previously in in use, was devised in 1903 by Prof. E. A. Schafer, F.R.S., after considerable researches into the efficiency of various systems, and has been adopted by the Royal Life Saving Society.

The first difficulty with a person suffering from electric shock is to obtain a clear passage through the patient's lungs—always a difficult operation—the teeth are frequently clenched and the tongue at the back of the throat, stopping the airway, and under methods usually adopted the first thing done is to open the patient's mouth and free the airway by grasping the tongue. In Schafer's method the patient is laid face downward, thus avoiding the necessity of attending to the tongue. 'The face downwards position not only insures the falling forward of the tongue, but also facilitates the escape of mucus and fluid from throat and mouth by natural gravitation, and prevents waste of valuable time which would be otherwise occupied in preliminary operations, and would delay the commencement of artificial respiration.'

The advantages of the Schafer method may thus be enumerated;—

1. The ease with which artificial respiration may be performed, hardly any exertion being required.
2. The efficiency with which the exchange of air in the lungs can be produced.
3. The extreme symplicity of the procedure.
4. The impossibility of the air passages being blocked by the falling back of the tongue.
5. (In cases of drowning). The readiness with which water and mucus are expelled from the air passages through the mouth and nostrils.
6. It involves no risk of injury to the congested liver or any other organ.
7. It is very easily remembered, and can be brought into operation by one person.

The method may be described as follows:—

The patient should be at once placed face downwards on the ground, and, without stopping to remove clothing, artificial respiration should be commenced immediately, as every instant of delay is serious.

Rough usage, especially twisting or bending of the limbs, should be avoided, and under no circumstances should the patient be held up by the feet.

In the event of respiration not being entirely suspended, it may not be necessary to imitate breathing; in that case the patient may be placed on his side, and natural respiration may be assisted by the application of an irritant substance to the nostrils and tickling the nose. Smelling salts, pepper, or snuff may be used.

To effect artificial respiration one should put one's self astride or on one side of the patient's body, in a kneeling position, facing his head. Placing one's hands flat in the small of the back, with the thumbs nearly touching and the fingers spread out on each side of the body over the lowest ribs, lean forward, and steadily allow the weight of one's body to fall over upon them, and so produce a firm downward pressure, which must not be violent. By this means the air (and, in cases of drowning, water, if there be any) is driven out of the patient's lungs. Immediately thereafter swing backward, rapidly releasing the pressure, but without lifting the hands from the patient's body. This backward and forward movement (pressure and relaxation of pressure) is repeated every four or five seconds. In other words, one sways one's body forward and backward upon one's hands twelve or fifteen times a minute, without any marked pause between the movements. By these means an exchange of air is produced in the lungs similar to that effected by natural respiration—every pressure forces air out of the chest, every relaxation of pressure causes it to pass in.

This procedure must be pursued until natural respirations are resumed. If they are resumed, and, as sometimes happens, again tend to fail, the process of artificial respiration must be resorted to as before.

Whilst the operator is carrying out artificial respiration others may, if there be opportunity, busy themselves with applying hot flannels to the limbs and body, and hot bottles to the feet, or promote warmth by friction, but no attempt should be made on the part of the operator to remove the clothing or to give any restoratives by the mouth until natural breathing has re-commenced.

When natural respiration is once established, one ceases to imitate the movements of breathing, and the patient may then be turned face upwards, and the treatment for the promotion of warmth and circulation proceeded with.

Friction over the surface of the body must be resorted to, using handkerchiefs, flannels, &c. By these means the blood is propelled along the veins towards the heart.

The friction along the legs, arms, and body should all be towards the heart, and continued after the patient has been wrapped in blankets or some dry clothing.

As soon as possible after breathing has been established remove the patient to the nearest house, and further promote warmth by the application of hot flannels to the pit of the stomach, and bottles or bladders of hot water, heated bricks, &c., to the armpits, between the thighs, and to the soles of the feet. If there be pain or difficulty in breathing, apply a hot linseed-meal poultice to the chest.

On the restoration of life, a teaspoonful of warm water may be given; then if the power to swallow has returned, very small quantities of wine, warm brandy and water, beef tea, or coffee administered, the patient kept in bed, and the disposition to sleep encouraged.

The patient should be carefully watched for some time to see that breathing does not fail.

If the patient has been carried to the house, care should be taken to let the air circulate freely about the room, and crowding round the patient prevented"—*General Report and Statistics of Mines and Quarries, Mining Engineer*, Nov. 1909, p. 261. (A. R.)

TELLURIDES.—"In *Economic Geology* for October, Victor Lenher contributes 'Some Observations on the Tellurides.' He has collected the few facts known concerning these interesting minerals, and has added results of experiments made by himself. Although the tellurides are of little use for industrial or metallurgical purposes, they are important by reason of their association with ore deposits rich in precious metals. Native tellurium is credited to Boulder County and to Cripple Creek, Colorado; the latter part of this statement is open to doubt; the most notable recent find of this metallic element should be credited to Vulcan, in Gunnison county, Colorado. Calaverite (named after the county of Calaveras, in California), and krennerite (after its discoverer Krenner) are the tellurides richest in gold. It is obvious that Mr. Lenher has not examined many specimens of calaverite from Kalgoorlie, otherwise he would not describe the mineral as 'silver white or slightly bronze coloured.' The West Australian calaverite is a rich lustrous yellow. When pure it contains 44% gold and 50% tellurium. Hessite is the silver telluride; good specimens have come from the La Plata mountains (Colorado), as well as from Nagyag and Veraspatak, in Transylvania. The lead tellurides altaite and nagyagite are named after the localities where they were first found, but they have an academic interest only. The telluride of mercury is coloradoite and it is allied to kalgoorlite, which is a West Australian variety containing gold and silver. Tetradymite is the bismuth telluride and rikardite is the copper telluride. The last of these was determined in 1902, from specimens found in Colorado; since then it has been recognised in several districts. The study of the chemistry of the tellurides derives importance from the light it throws on the origin of ore deposits containing gold and silver. Tellurium reduces the salts of the precious metals, especially gold. The gold tellurides all precipitate gold from a chloride solution. When tellurium itself is treated with ferric chloride the tellurium dissolves, reducing the ferric chloride to a ferrous condition. This reaction has been used as the basis for the extraction of tellurium from telluride ores. When roasted, the tellurides lose their tellurium, which tends to become fixed by combination with ferric oxide, as an iron tellurate. Tellurium can be transported in nature as an alkaline tellurite or tellurate. As a sulphide it is readily soluble in alkaline sulphides, and so long as free oxygen is absent, it can be transported in this form. As Mr. Lenher says: 'The close association of the tellurides with the more deep-seated sulphides would suggest that it is possible that these sulphides furnish sulphide solutions that are capable of carrying the tellurium, and that on contact with the gold solutions from lateral sources they are able to produce the telluride deposits.'"—*Mining Magazine*, Dec. 1909, p. 323. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ORE DRESSING. By ROBERT H. RICHARDS. Vol. III. and IV. 8vo., pp. 850. Illustrated. McGraw-Hill Book Co., New York. Price 42s.

"The publication of these two volumes supplements the two-volume book issued in 1903. The author is professor of mining engineering and metallurgy in the Massachusetts Institute of Technology, at Boston, and needs no introduction to those engaged in mining. By preparing these two volumes, together with an index (which constitutes a separate and fifth volume), Mr. Richards has completed a monumental task, which will ensure his abiding fame as a specialist in the wet treatment of ores. It will be open for technical men to criticise the voluminous character and the consequent cost of this tremendous treatise, but these drawbacks are inevitable from the bigness and intricacy of the subject. Moreover, the time necessary to the preparation of the successive volumes, or even chapters, has been so great that some of the earlier portions have been out-of-date by the time the concluding paragraphs went to press. Even in preparing these last two volumes the author was compelled to enlarge his matter from a small supplement or appendix to dimensions fully equal to the original publication. Meanwhile the revision of Vol. I. and II. was prohibited by the fact that the details are so interwoven as to preclude any method not involving re-writing. Indeed, we sympathise with the author, who is known to be as painstaking as he is accurate, for new concentrating devices and even new principles of ore dressing have succeeded others so rapidly as to embarrass a conscientious student desiring to be a trustworthy teacher. In order to avoid the revision of the first two volumes while yet adding the latest available information, the author has given us two additional volumes in which the subject matter of Vol. I. and II. is supplemented chapter by chapter. The fifth volume is an index that binds the four volumes together. Thus the reader is practically compelled to purchase Vol. III. and IV. in order to obtain the corrections and amplifications of the matter appearing in Vol. I. and II.; at the same time the buyer of Vol. III. and IV. will have only a lordly fragment of technical information until he completes his purchase by obtaining the first two volumes. The index is supplied free to purchasers of Vol. III. and IV. We confess that we do not like this feature of the publication, but we suppose that it was unavoidable. Certainly the cost of books of this kind is insignificant compared to the information they give; a single page, even a paragraph, will often contain data worth a hundred times the price that an engineer may have paid for the privilege of owning so comprehensive a technical guide. In one chapter Mr. Richards gives details concerning 94 representative mills in the principal mining centres of the world; he publishes data concerning cost, power, extraction, arrangement, and so forth, in such detail as to give figures and facts of the greatest value to anyone engaged in this branch of metallurgy. In another chapter the cost of milling is elucidated with elaboration, affording a wealth of practical information. We venture one criticism: the author practically ignores the air-cushion stamp. In the first edition the working of this stamp, owing to its misleading name—the 'pneumatic stamp'—was misinterpreted. In the second edition, some information was given concerning one or two stamps of this type that had proved to be failures. In Vol. III. just published, the whole subject is dismissed in a few lines, with the remark that this type of stamp is not gaining ground. As a matter of fact, the air-cushion stamp has run successfully at Dolcoath for several years, and an additional installation is now being erected. Similarly the experience at East Pool has induced the engineers to increase their plant. In South Africa, it is being tried at the New Kleinfontein, as was mentioned in our September issue. The air-cushion stamp has its limitations, but being in successful operation, it deserves a place in an encyclopedia of ore-dressing. We congratulate the author on fulfilling his purpose in giving technical literature a treatise so comprehensive and so trustworthy."—*The Mining Magazine*, Oct., 1909, p. 160. (W. A. C.)

DIE DIAMANTFÜHRENDEN GESTEINE SÜDAFRIKAS. By DR. ING. PERCY A. WAGNER. Pp. 207 with 29 illustrations and 2 tables. (Berlin, Deutschland: Gebrüder Bornträger.)

"This work will, no doubt, be eagerly perused by a number of people interested in the South African diamond industry, who are always on the look out for fresh facts in connection with the diamond fields, both from a practical and scientific aspect; but although the author shows himself possessed of a very intimate knowledge of the subject he deals with, yet the facts which he places before his readers are for the greater part derived from efforts other than his own, and consequently already previously brought to their knowledge. Nevertheless, such data as the author claims to be his own are certainly well worth consideration and study, and the author's ability of assimilating and combining facts mentioned by other writers on the subject are worthy of appreciation and deserve commendation in every respect. It is but seldom that one finds a work of this description, in which a number of facts concerning a particular object are so well put together, and the author is modest enough to acknowledge his indebtedness to others throughout the book. The frequent references and cross references that occur in the text certainly render the reading of the book somewhat difficult for such students as have not all the literature referred to either at their fingers' ends or in the shelves beside them; but on the whole the usefulness of the work is much enhanced by such references to the standard works and writings of other authorities.

Dr. Ing. Percy A. Wagner is without doubt an authority on the subject he deals with, and a translation of this book into our language would be a boon to students who feel that there is still room for further literature on the subject, and who wish to gain a practical and useful knowledge in this direction without having time or opportunity to read all that has been written on the diamond fields of South Africa, as in the short compass of this work a great many useful, interesting matters are considered which one hardly would look for in a book of its size.

To revert to the work itself. The first few pages are devoted to a somewhat exhaustive list of the literature published in a variety of countries on the same subject, in all numbering 138 books, papers, articles, etc., to which reference is made in the book, and this in itself should prove to be a useful compendium for the student.

Although the writer deals only in a very small portion of his work with the German South-West Africa diamond fields, yet the facts he mentions are interesting. According to Dr. Wagner, the field has a great future before it, in spite of the small size of the stones found hitherto, and he tells us that from May, 1908, to the end of December of that year 39,762 carats, representing a value of M. 1,100,000, have been found. Whether the very sanguine opinion that the stones, averaging as they do from 4 to 5 carats, are of exceptionally fine quality, will be maintained in the future, is yet a question of considerable doubt.

We can congratulate Dr. Wagner on his efforts now placed before us, and shall certainly look forward to further works from his pen."—*London Mining Journal*, Oct. 23, 1909, p. 149. (G. H. S.)

Bairstow, Leonard. The Elastic Limits of Iron and Steel under Cyclical Variations of Stress. 4to, sd. *Dulau.* 1s. 6d.

Ballinger, W. F., and Perrot, E. G. Inspector's Handbook of Reinforced Concrete. 12mo. *Constable.* Net 4s.

Bedell, Frederick. Direct and Alternating Current Testing. 8vo., pp. 276. *Constable.* Net 8s.

Bell, James. Introductory Science. Cr. 8vo., sd. *Normal Press.* Net 1s. 6d.

Corbin, Thomas W. Modern Engines. Cr. 8vo., sd., pp. 135. *C. A. Pearson.* Net 1s. Cloth, net 1s. 6d.

Cotterill, J. H., and Slade, J. H. Lessons in Applied Mechanics. Vols. I. and II. 12mo. *Macmillan.* Each 3s.

Dienhardt-Schlomann Series of Technical Dictionaries. Vol. 7. Hoisting and Conveying Machinery. Compiled by P. Stulpnagel. 12mo. *Constable.* Net 10s. 6d.

Franklin, W. S. Electric Waves. 8vo. *Macmillan.* Net 10s.

Gray, Frank J. The Computation of Area. A Simple Treatise. 12mo. sd., pp. 64. *St. Bride's Press,* Net 1s.

Grover, Frederick. Modern Gas and Oil Engines. 5th Edition. Revised and Enlarged. Cr. 8vo., pp. 382. *Technical Publishing Co.* Net 5s.

Guldner, Hugo. The Design and Construction of Internal Combustion Engines. Translated by H. Diederichs. Illustrated. 4to. *Constable.* Net 42s.

Hobart, H. M. Electricity. 8vo , pp. 228. *Constable.* Net 6s.

Lawn, J. G. Mine Accounts and Mining Bookkeeping. New Edition. 8vo., pp. 158. *C Griffin.* 10s. 6d.

Leeds, F. H., and Butterfield, W. J. A. Acetylene. The Principles of its Generation and Use. 2nd edition. Revised and enlarged. Cr. 8vo., pp. 408. *Griffin.* Net 8s. 6d.

Levin, A. M. The Modern Gas Engine. 8vo. *Chapman and Hall.* Net 17s.

Loney, L. An Elementary Treatise on the Dynamics of a Particle and of Rigid Bodies. 8vo., pp. 384. *Cambridge University Press.* 12s.

Lowe, H. Paints for Steel. 5th edition. Cr. 8vo. *Chapman and Hall.* Net 4s. 6d.

Lunge, George. Coal-Tar and Ammonia. 4th and Enlarged Edition. Vols. 2, 8vo., pp. 1212. *Gurney and J.* Net 42s

Lunge, George. The Manufacture of Sulphuric Acid and Alkali, with the Collateral Branches. 3rd Edition, much Enlarged. Vol. 2. Parts 1 and 2. 8vo., pp. 1042 *Gurney and J.* Net 42s.

Maurice, William. The Shot-Firer's Guide: A Practical Manual on Blasting, etc. 8vo., pp. 214. *Electrician.* Net 3s 6d.

McQuade, W. Engines and Boilers Practically Considered. Cr. 8vo., pp. 88. *Dawbarn and W.* Net 3s. 6d.

Middleton, G. A. T. The Elements of Reinforced Concrete Building. Cr. 8vo., pp. 120. *F. Griffiths.* Net 4s.

Morgan, J. J. Blast Furnace Practice. Cr. 8vo., pp. 54. *C. Griffin.* Net 1s. 6d.

Murray, James Erskine. Wireless Telephones and How They Work. Cr. 8vo., pp. 76. *C. Lockwood.* Net 1s. 6d.

Norman, G. M. Systematic Practical Organic Chemistry. (The Organised Science Series.) 2nd edition. Cr. 8vo., pp. viii-104. *Clive.* Net 1s. 6d.

Poincare, Lucien. Electricity, Present and Future. 8vo., pp. 324. *Milner.* Net. 5s.

Peckham, S. F. Solid Bitumens. Roy. 8vo. *Spon.* Net 21s.

Redwood, T. T. Lubricants, Oils and Greases. 8vo. *Spon.* Net 6s. 6d.

Ripper, William. Heat Engines. Being a new Edition of "Steam." Cr. 8vo., pp. 322. *Longmans.* 3s.

Science Year Book, The. Diary and Scientific Summary, 1910. 8vo. *King and Sell.* Net 5s. Abridged ed. limp. Net 1s.

Shillito, F. W. Handbook of Corliss Steam Engines. 4th edition. 8vo. *Spon.* Net 4s. 6d.

Thompson, A. Beeby. Petroleum Mining and Oil Field Development. 8vo., pp. 382. *C. Lockwood.* Net 15s.

Thomsen, T. C. Internal Lubrication of Steam Engines. Cr. 8vo., pp. 98. *Technical Pub. Co.* Net 2s. 6d.

Thornton, Alfred Augustus. Thornton on Patents, British and Foreign. Roy. 8vo., pp. 574. *C. Jones.* Net 21s.

Tutton, A. E. H. Crystalline Structure and Chemical Constitution. 8vo., pp. 212. *Macmillan.* Net 5s.

Wodiska, Julius. A Book of Precious Stones. The Identification of Gems and Gem Minerals, etc. Cr. 8vo., pp. 382. *Putnam.* Net 10s. 6d.

Wright, F. W. The Design of Condensing Plant. A practical Treatise. Cr. 8vo., pp. 208. *Technical Pub. Co.* Net 3s. 6d.

Year Book of the Scientific and Learned Societies of Great Britain and Ireland. A Record of the Work done in Science, Literature and Art during the Session 1908-1909. By numerous Societies and Government Institutions. Compiled from Official Sources. 26th Annual Issue. Cr. 8vo., pp. vi.-356. *C. Griffin.*

Abstracts of Patent Applications.

(C.) 500/08. A. J. Arbuckle. Improvements in apparatus for the treatment of crushed ore products for the recovery of the metal contents. 19.12.08.

The above application claims apparatus for the treatment of crushed ores comprising the combination of dewatering cones, previously patened by this applicant, and air-lift dissolving vats. The process described involves features such as air-lift dissolving vats and return of overflow solution for elevating thickened pulp before pumping.

(C.) 112/09. William Arthur Caldecott. Improvements in or connected with dissolving vats for use in the treatment of ores. 6.3.09.

The object is the acceleration of the dissolving of gold in sand practically free from slime.

The improvements consist of the combination of various devices for the aeration and mechanical agitation of ore pulp with the application of the Caldecott cone diaphragm No. 231, 1908.

This application is the combination of the Caldecott cone diaphragm for the treatment of sands with two systems of pump agitation, with a system of air agitation, and also with a system of mechanical agitation by means of a screw working in a verticle pipe known as the "Hendryx Agitator."

(C.) 167/09. Alexander John Arbuckle. Improvements in means for introducing and distributing mixtures of solids and liquids in settling or other vats and vessels. 5.4.09.

This specification claims for the purpose of facilitating the settling of solids from pulp certain arrangements of internal cones and rings with a verticle shaft provided with helical scraper blades "substantially as described and shown."

(C.) 218/09. Thomas Millar Smith. Improvements in means for obtaining better combustion of the fuel in the furnaces of steam generators and the like. 8.5 09.

This refers to the heating of air to a high temperature in passages or tubes located in the combustion chambers of a steam boiler before mixing it with the volatile gasses issuing from the grate.

(C.) 289/09. Karl Sophus Sturup (1), Theodore Wilfrid Rill (2). Improvements in means for distributing sand on mine dumps, also applicable for other similar purposes. 18.6.09.

This application relates to means for delivering tailings over the edge of a dump, and comprises a power-driven conveyor belt moving horizontally at right angles to the line of the track for the trucks. This belt is mounted on a carriage travelling on rails. Slide rails are provided for raising the trucks above the conveyor, where the contents of the trucks are tipped into a hopper.

(C.) 547/09. Oscar Clarence Beach. Improvements in grinding mills. 26.11.09.

This refers to numerous improvements in small plate grinding mills for assayers' samples, including means of getting better alignment of discs, complete discharge of all the ore, saving of dust, and better lubrication of the machine.

(C.) 548/09. Robert Lemmon (1), Robert Patrick Holmes (2). Automatic filter press. 26.11.09.

This machine consists of an automatic continuous filter press, having filter chambers on an intermittently revolving table. The filter chambers are fitted with air-tight plungers, reciprocating vertically into and out of said chambers, thereby compressing air on the surface of the slime pulp and forcing the liquid through a filtering medium placed in the bottom of the chamber. The pulp is admitted through a launder into the chamber directly underneath it, pressed by means of the air-tight plunger, then moved forward in the chamber by means of a 1-12th revolution of the table to the position of the next plunger. This operation of pressing is repeated four times. In the fifth position of the chamber washing and agitation of the pulp takes place, thence pressing is repeated twice, followed by another agitation and wash, and again pressing is repeated three times.

The final position of the chamber is the point of discharge of pressed slime, which operation is performed by automatically lifting the filter-bed to the level of the top of the chamber and removing the slimes by means of a scraper. As there are twelve chambers in all, there will thus be one chamber filling, eight simultaneously pressed, two simultaneously agitated and washed, and one being discharged.

(C.) 570/09. Alfred Nollitt Sedgley. Improvements in centrifugal pump, filters, and strainers. 10.12.09.

Two skeleton drums are mounted vertically on a tubular frame connected to a vertical tubular axis between the drums: around the drums is passed a suitable filter cloth in the form of a belt, and drawn tight on the drums by two rollers.

The system revolves on the central axis at a suitable speed, and the pulp is fed down the axis, passes through the tubular frame to the interior of the two drums, and is thrown against the filter cloth, which passes round them, and retains the solid portion of the pulp while allowing the solution to pass through and be collected in an outer fixed casing in which the drum frame revolves.

An arrangement is provided for rotating the drums on their own axis, thus causing the filter belt to travel and pass two scrapers, which clear it of the sands and drop them into a suitable receptacle beneath.

(C.) 574/09. John Collins Clancy. Treatment of precious metalliferous ores. 10.12.09.

This application refers to a process for the treatment of ores containing precious metals by means of sulphocyanide solutions, and using ozone as the oxidizing agent. Lime, caustic soda, and soluble iodine compounds might also be used in connection therewith. It is necessary that the mass under treatment should be in a non-acid condition, in which state the inventor claims his process has greater affinity for the precious metals than for the base metals which remain undissolved in the ore mass.

(C.) 577/09. Henry Livingstone Sulman (1), Henry Howard Greenway (2), Arthur Howard Higging (3). Improvements in or relating to the concentration of ores. 15.12.09.

The above application refers to a modification of the "flotation" method of concentration, which consists in the substitution of certain specified organic substances for oil or oily liquids previously used in conjunction with slightly acidified water, as mineral frothing agents. The substances mentioned comprise certain organic acids, ethereal salts, aromatic hydroxy-compounds, and ketones.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 17/10. R. B. Ballantine (1), M. Wagner (2). An improved process for the reduction of cassiterite and apparatus therefor. 21.1.10.

(P.) 18/10. C. A. Parsons. Improvements relating to the governing of fluid operated motors. 21.1.10.

(C.) 19/10. E. Richardson (1), J. T. Marshall (2). Improvements in slide valves for fluid pressure, steam and other engines. 21.1.10.

(C.) 20/10. W. Burgess. A composition or compound for neutralising the noxious fumes produced by blasting. 21.1.10.

(P.) 21/10. E. J. Way (1), A. J. Arbuckle. Improvements in vats or vessels for the treatment of crushed ore products. 22.1.10.

(P.) 22/10. James Garvie. Double faced valve and joint connection. 27.1.10.

(P.) 23/10. James Edward Thomas. Improvements in or relating to tube-mill feeding devices. 27.1.10.

(P.) 24/10. John Henry Houghton. An improved mixture for neutralising the noxious fumes produced by blasting in mines. 28.1.10.

(C.) 25/10. Georg Schwabach. Improvements in regulating devices for gas and the like supply plants. 28.1.10.

(C.) 26/10. Ernest Armstrong. Improvements in or relating to files, rasps or similar tools. 28.1.10.

(C.) 27/10. Godfrey Benington Johnson. Improvements in and in apparatus for producing corrugated metal sheets for roofing and other purposes. 28.1.10.

(C.) 29/10. Adolf Andersson. Improvements in rock drilling machines.

(P.) 30/10. Fred. Herbert Brislin. Improvements in or relating to fingers or devices for putting the stamps of stamp batteries out of action. 2.2.10.

(P.) 31/10. Peter Miller. Improvements in drills or bits applicable for machine or hand drilling. 2.2.10.

(P.) 32/10. Elisha Tippett. Improvements in or relating to rock drilling machines. 2.2.10.

(P.) 33/10. James Higham. Improved automatic and continuous sample dividing apparatus. 3.2.10.

(P.) 35/10. John Whitford (1), Joseph Amos Mills (2). Improvements in mine ventilation. 3.2.10.

(P.) 36/10. Frederick George Fell. Stamp safety clamp. 4.2.10.

(P.) 38/10. Frederick Wesley Thomas. Clean up machine. 5.2.10.

(P.) 41/10. Duderick Cornelius Johannes Enzlin. Improvements in safety lamps. 9.2.10.

(P.) 42/10. Allan Ewen Victor Fraser. Automatic petrolene gas generator. 9.2.10.

(C.) 43/10. John Lowther. Improvements relating to bearers for mine shafts.

(C). 45/10. Frederick Benjamin Adolphe Kirsten. Improvements in apparatus for recovering diamonds. 10.2.10.

(P.) 46/10. Edward Jameson (1), Victor Frolich (2). Improvements in means for operating gravity stamps. 11.2.10.

(C.) 47/10. Henry William Muhleisen. Drilling apparatus. 11 2.10.

(C.) 48/10. William Philip. Improvements in safety devices for hoisting apparatus. 11.2.10.

(P.) 49/10. Charles McGregor. Improvements in means for igniting the fuses or the like of explosive charges in blasting. 12.2.10.

(P.) 50/10. Emmanuel Kohlmeyer (1), Wilhelmus Hendrikus Visser. Improvements in apparatus for concentrating pulp. 12.2.10.

(P.) 51/10 M. S. Beaton (1), J. Eason (2), J. H. Wilson (3). Improvements in automatic lubricating apparatus. 16.2.10.

(C.) 52/10. H. Stephan. Improvements in conduits for granular matter. 18.2.10.

(C.) 53/10. R. Macpherson (1), W. E. Heys (2). Improvements in the manufacture of soaps. 18.2.10.

(C.) 54/10. H. H. Benn (1), P. A. Ransom (2), G. F. Buck (3), The Unbreakable Pulley & Mill Gearing Co., Ltd. (4). Improvements in connection with friction clutches. 18 2.10.

(P.) 56/10. F. Rautenback. The mono-rail car. 18.2.10.

(C.) 57/10. C. C. Hansen. Improvements in mountings for drills. 18.2.10.

(C.) 58/10. C. C. Hansen. Improvements in mining machine mountings. 18.2.10.

(P.) 60/10. J. R. Thurlow. Improvements in or relating to apparatus for classifying crushed ore products. 21.2.10.

(C.) 61/10. J. Cornell. A combined cock and lubricator. 21.2.10.

(P.) 62/10. J. Holman. Improvements in means for allaying the dust or collecting the detritus in rock drilling. 21.2.10.

(P.) 63/10. H. C. Behr. Improvements in mine ventilation. 21.2.10.

(P.) 64/10. H. C. Behr. Improvements in means for sub-dividing streams of pulp. 21.2.10.

(C.) 65/10. J. N. Cook. A combined pipe attachment. 21.2.10.

(C.) 67/10. H. Donner. Improved process and apparatus for obtaining an increase in the pressure of a liquid. 22.2.10.

(P.) 68/10. J. Nicholls. Toxic torch. 22.2.10.

(P.) 69/10. C. F. E. O. Schmitt. Improvements in tube mill feeding means, also applicable in part to spiral pumps. 24 2.10.

(P.) 70/10. T. M. Smith. Improvements in means for obtaining better combustion of the fuel in the furnaces of steam generators and the like. 25.2.10.

(P.) 71/10. E. J. Way (1), A. J. Arbuckle (2). Improvements in apparatus for the treatment of crushed ore products. 25.2.10.

(C.) 72/10. A. M. Robeson. An improved method of refrigeration. 25.2.10.

(P.) 73/10. W. O. Purves. Automatic coupling for buffers. 26.2.10.

(P.) 74/10. F. V. W. Swanton. Improved apparatus to produce power by weighted epicycles acting in a certain epicyloidal manner. 28.2.10.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Adams, E. L., *l/o* Rhodesia ; Village Deep, Ltd., P.O. Box 1064, Johannesburg.

Adler, H. W., *l/o* Cleveland ; Rose Deep Ltd., P.O. Box 6, Germiston.

Brown, J., *l/o* Luipaardsvlei ; P.O. Box 172, Fordsburg.

Chapple, A. J., *l/o* Cobar, N.S.W.; Oriental St., Stawell Victoria, Australia.

Comrie, A., *l/o* Essexvale ; Susanna Mine, Gwanda, Rhodesia.

Evans, Lewis, *l/o* Johannesburg ; Gwelo, Rhodesia.

Jameson, C. G., *l/o* Cleveland ; P. O. Box 272, Salisbury, Rhodesia.

Johnson, Tom, *to* P. O. Box 2986, Johannesburg.

Johnston, W. H., *l/o* Doornfontein, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.

Jupp, H. L., *l/o* Luipaardsvlei ; Boksburg.

Linton, Robt., *l/o* Los Angeles ; Sierra Mining Co., S.A., Ocampo, Chih., Mexico.

Macaulay, M., *l/o* Barberton ; Battlefields Mine, Rhodesia.

Mackenzie, A. C., *l/o* Germiston ; P. O. Box 1597, Johannesburg.

Mather, T. W., *l/o* Fordsburg ; Gold Coast Amalgamated Mines, Ltd., Tarkwa, West Africa.

Matthews, E. L., *l/o* Denver ; Warren Hill Syndicate, Klerksdorp.

Pam, E., *to* Village Deep, Ltd., P. O. Box 1145, Johannesburg.

Reece, F. B., *l/o* Magollon, New Mex ; Blaisdell Coscotitlan Syndicate, Apartado 92, Pachuca Hid., Mexico.

Rhodes, C.E., *l/o* Guanajuata ; El Oro Mining and Railway Co., Ltd., Aparado 30, El Oro, Estado de Mexico, Mexico.

Simpson, F. L., *l/o* Johannesburg ; Bushtick Mine, per private bag, Bulawayo.

Southwell, B., *l/o* Krugersdorp ; Knights Central, Ltd., P. O. Box 91, Germiston.

Stockett, A. W., *to* Simmer and Jack Proprietary Mines, Ltd., P.O. Box 192, Germiston.

Strange, E. H., *l/o* Canada ; 3931 Temple St., Calumet, Mich. U.S.A.

Terry, C. W., *l/o* Gadzema ; Shamva Mine, private bag, Abercorn, Rhodesia.

Thorpe, W. E , *l/o* East Rand ; Princess Estate and G.M. Co., Ltd., P. O. Box 112. Roodepoort.

Womble, L. A., *l/o* Boksburg ; Princess Estate and G.M. Co., Ltd., P. O. Box 112, Roodepoort.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. MARCH, 1910. No. 9.

Proceedings
AT
Ordinary General Meeting, March 19, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, March 19th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

44 Members: Dr. J. Moir, Messrs. W. R. Dowling, F. F. Alexander, K. L. Graham, J. E. Thomas, H. A. White, Prof. J. A. Wilkinson, A. F. Crosse, W. A. Caldecott, R. Allen, A. J. Bowness, W. Broom, P. Carter, F. W. Cindel, F. C. Ford, J. Gaze, J. Gray, C. E. Hall, J. H. Johnson, T. Johnson, T. W. Jordan, G. A. Lawson, J. Lea, F. G. Macdonald, S. Morison, S. Newton, E. A. Osterloh, D. J. Pepler, F. D. Phillips, J. Pyles, S. C. Quin, O. D. Ross, H. A. Scarf, W. Sharp, G. H. Smith, A. L. Spoor, S. H. Steels, C. T. Sydenham, A. Thomas, J. M. Thorburn, F. W. Watson, F. Wells and L. J. Wilmoth.

17 Associates and Students: Messrs. C. F. Bayly, C. J. Crocker, J. Cronin, W. J. R. Hunter, A. King, A. Olsen, A. Plinke, F. J. Pooler, H. B. Powter, J. M. Robinson, M. Schaap, A. M. Thomas, W. E. Thorpe, H. Ward, W. Westwood, and E. J. Wiseman.

12 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the February *Journal*, were confirmed.

NEW MEMBERS.

Messrs. H. A. White and Prof. J. A. Wilkinson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Lloyd, James John, c/o British Consular Agent, Diego Suarez, Madagascar. Assayer and Amalgamator.
McPhee, James, Vereeniging Estates, Ltd., Vereeniging. General Manager.
Mills, Joseph Amos, City Deep, Ltd., P. O. Box 1411, Johannesburg. Mechanical Engineer.
Nichols, Horace G., M.Inst.M.M., "Downside," Ashtead, Surrey, England. Mining Engineer.
Pollard, Benjamin Edward, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Assayer.
Spence, Arthur Godfrey Benham, New West Bonanza, Ltd., P. O. Box 137, Klerksdorp. Surveyor and Assayer.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Brookes, William Horace, Bantjes Consolidated Mines, Ltd., P. O. Box 2, Florida. Assayer.
Dyason, Edward Clarence, M.Aus.I.M.E., 60, Queen Street, Melbourne, Australia. Mining Engineer.
Holden, William Clifford, A.Inst.M.M., Cinderella Deep, Ltd., P. O. Box 75, Boksburg.
Simpson, John Robert, Roodepoort United Main Reef G. M. Co., Ltd., P. O. Box 115, Roodepoort. Cyanider.
Smith, Thomas Francis, Crown Mines, Ltd., P. O. Box 158, Fordsburg. Cyanider.
Waterhouse, Lionel Lawry, B.E.(Sydney), Mt. Conqueror Gold and Arsenic Recovery Co., Ltd., Peel, via Bathurst, New South Wales. Metallurgist.

As Students.—

Ninow, Wilhelm, Simmer Deep, Ltd., P. O. Box 178, Germiston. Tube-mill Learner.
Parnell, Frederick Leopold, French Bobs Mines, Ltd., Louw's Creek, Barberton. Mining Assistant.
Schaap, Maurice, Jumpers G. M. Co., Ltd., P. O. Box 2631, Johannesburg. Cyanide Learner.
Taylor, Norman Sargant, French Bobs Mines, Ltd., Louw's Creek, Barberton. Mining Assistant.
Thorlund, Johannes, Transvaal University College, P. O. Box 1176, Johannesburg. Mining Student.

General Business.

Mr. A. F. Crosse (*Past-President*): There is a rather important matter which has been before the mining public for a good many months now, which ought primarily to interest this Society and that is the question of returning sands to the mines. I do not want to express any opinion on it this evening, but the question ought to be carefully thought out and be considered by a special sub-committee. We all know that the sands, though we may remove as much cyanide as possible, still contain a certain amount of cyanide. Now the addition of lime has no permanent effect in retaining that cyanide. I have no opinion on the subject whatever, but it is a very important matter. Some of you have, no doubt, collected butterflies, and may remember a quick way of killing them without injuring the specimen. A wide-mouthed glass-stoppered bottle is taken, into which a mixture of plaster of Paris and water is poured until it covers the bottom for an inch or so. After the plaster has set, a little solution of cyanide of potassium is poured in : the plaster of Paris, which is sulphate of calcium, a neutral salt, decomposes the cyanide of potassium so that the air in the bottle contains for a long time a certain amount of hydrocyanic acid. If there is any cyanide left in the tailings which it is proposed to return underground, this might possibly be decomposed slowly in the mine with the consequent liberation of hydrocyanic acid. It would be expensive to instal plants for dealing with the sands on the various mines, and afterwards find out that their use could not be continued. I think the Society should pay some attention to it and ascertain whether these sands would evolve hydrocyanic acid.

The President: I am sure we are indebted to Mr. Crosse for introducing this matter, but at the same time to take any action at the present time is perhaps a little beyond the scope of the Society. It will be brought up before the next Council meeting, and should the Council decide that anything should be done, the matter will be gone into. Should any gentleman care to write a paper on the subject, it would probably be well discussed, and it would then be brought forward in a public manner. We thank Mr. Crosse for bringing the matter before us.

DECENNIAL INDEX TO THE SOCIETY'S PROCEEDINGS.

The President: Under General Business I would like to draw attention to paragraph 7 of the Agenda. Some of the members of the Council thought it would be advisable that at this time an index should be prepared which should be applicable to all the volumes that have been published by the Society. A sub-committee of the Council went into the matter and found that it would require about £400 guaranteed before anything could be done, and the committee does not recommend the Council to take any action unless this guarantee is forthcoming. Therefore the Council would like all those, who would care to have the index, to send in their names and subscriptions, so that it might decide whether any action shall be taken in the matter.

BULLION AND ASSAY VALUES.

The President: On the same page of the Agenda we have a paragraph relating to bullion and assay values. The Council of the Institution of Mining and Metallurgy has issued circulars asking that those who have any opinions on the matters set forth here, should place those opinions before them. This Society has received a copy of the circular and it has been considered advisable to give the members an opportunity of stating their views. It seems to me that to-night a few minutes can be devoted to discussing some of the items tabulated. The first of these says: "Assay reports shall state the exact condition of the sample as to dryness when assayed." So far as I know in these parts there are no report forms which state definitely how the sample is treated after entering the assay office, and I think it is usually taken for granted that the sample is assayed as received. The Institute of Mining and Metallurgy want us to give an opinion. If any gentleman would care to speak on the matter we should be pleased to hear him. The circular reads as follows :—

THE INSTITUTION OF MINING AND METALLURGY. STANDARDISATION.

Copy of the Report of Sectional Committee "B"—Bullion and Assay values, as submitted to the Council, December 15th, 1909. (Note.—Before finally adopting the recommendations of the Committee, the Council will be glad to consider any suggested alterations or additions, which should be received by the Secretary not later than June 30th, 1910.)

BULLION AND ASSAY VALUES.

The Committee for the Standardisation of Bullion and Assay Values was formed for the purpose of recommending rules which shall, as far as possible, be followed by assayers, metallurgists and mining men, in reporting the results of assays and of metallurgical, dressing and other tests of which assays form an essential part, so that ambiguity shall be avoided and the possibility of misinterpretation or misuse of reports shall be minimised.

Reports of assay results, extraction tests, metallurgical and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the Committee urges the necessity for definite statements as to the nature and condition of the sample as assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead, antimony, etc., where both "wet" and "dry" methods of assay are in use, results should be reported as obtained by "fire" assay or by "wet" assay, with further details as to method, if judged advisable. Similarly, it should be stated whether the sample was assayed in the condition in which it was received or "air-dried" or dried at —°.

In reporting that any assay for such a constituent as tin, arsenic, tungstic acid, etc., corresponds to so much "black tin," "white arsenic," "wolfram" or other recognised saleable product, the report should give the result for such constituent, together with a definite statement as to the figures used in calculating such saleable product. Thus, an assay showing 1% of metallic tin might be reported as 1% of metallic tin, equivalent, if all could be extracted as such, to 1·5% of "black-tin" containing 66·6% (or two-thirds) of its weight of metallic tin. Similarly, in reporting results, where assay values might be mistaken as representing available produce, it should be stated that such are merely "gross" assay values, a statement or estimate of the actual "commercial" values being added whenever possible.

In such matters as ore-dressing, ore-treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and its relation to other samples, the Committee emphasises the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labelling, sealing and otherwise ensuring the correctness and authenticity of samples submitted for tests or assays.

The Bullion and Assay Values Committee submit the following recommendations for adoption by the Institution :—

(1) Assay reports shall state the exact condition of the sample as to dryness when assayed.

(2) Assay values of gold and silver ores and products shall be represented in pennyweights and decimals or in ounces and decimals, and not in oz., dwts. and grains. They shall be expressed in terms of fine gold and fine silver respectively, not as "bullion."

(3) Assay values of alluvials shall be reported in grains and decimals of a grain of "fine" gold, or in pence (at 2d. per grain of "fine" gold) or cents per cubic yard.

It is recommended that in the absence of specific information, one cubic yard of ordinary alluvial, excluding boulders, be taken is equivalent to 3,000 lb. (1½ short tons).

(4) In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution. (In the case of cyanide solutions, the use of the "fluid ton of 32 cubic feet" is recommended. It closely approximates to 2,000 lb. and is in common use.)

(5) When it is necessary to state or estimate the money value of an ore, etc. (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

(6) Laboratory sieving tests shall be made with the I.M.M. Standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

Mr. A. F. Crosse (*Past-President*) : If we dry slime at the temperature of boiling water a certain result is obtained. If we dry it at 200° C. it makes a considerable difference. I have had slimes assaying 7 and 8 dwt. of gold per ton, and a difference of 10% of moisture would be important.

The President : That was a point I was also considering. I do not know whether it has come under your notice, but I have occasionally received samples extremely wet and in a very fine state of division which have been taken from some old drive where the rock has been almost weathered away. In a case like that I should say it ought to be stated whether the sample was assayed after being air dried, or otherwise.

Prof. J. A. Wilkinson (*Member of Council*) : I understand that the recommendations are to be made regarding samples received from all parts of the world. In our case we must therefore remember, that the air here is much dryer than in most other places, and in specifying for conditions of dryness I think it would be best to state that any given sample or samples should be dried for a given period of time at a definite temperature. The case of coal naturally occurs to one, where serious discrepancies and discordant results have been obtained, owing to the fact that different temperatures for drying and determination of the volatile matter have been used by different chemists.

Mr. A. F. Crosse : That reminds me of an interesting case which occured to me about 14 years ago. A Past-President of our Society analysed some coal, and a similar sample was also sent to me for analysis. The results were very different as regards the volatile matter. He dried the coal at the ordinary temperature of boiling water. I dried it at a higher temperature. I found about 15% moisture, and he reported about 4%. He thought my result impossible but I proved its correctness to him. The coal was somewhat exceptional, but it shows what an enormous difference it may make whether coal is analysed when dried at 100° C. or when dried at 115° C. In the case I quote it made a difference of about 11% in moisture.

The President : These remarks about coal, though interesting, are scarcely to the point. What we are asking is with regard to assay reports of gold. Will Prof. Wilkinson make a proposition that we should definitely state the temperature at which samples have been dried ?

Prof. J. A. Wilkinson : Yes, as I think uniformity in procedure so as to get definitely comparable results is an eminently desirable object. Concerning the behavour of slimes when drying I have only a little experience, but judging from their chemical composition, I see no reason to place them in a different category.

Mr. H. A. White (*Member of Council*) : I should like to point out that a fixed temperature will not give the same results everywhere. If the boiling point of, say, toluene is prescribed at the place of assay, more consistent results can be expected.

The President: The next paragraph is in general acceptation here and we should agree with it, and I take it that the following paragraph with regard to assay values of alluvials should be adopted also. With reference to paragraph 4 dealing with assay values of cyanide and other solutions, this brings up the question Mr. Wilmoth raised in his paper with regard to acid washes. I should like to know whether any gentlemen has anything to say with regard to the treatment of acid washes, that is whether we should report such samples by volume or by weight.

Mr. H. A. White: There is one point in paragraph 4 "in parts by weight." That would mean percentages, whereas the usual practice is to report in dwts. and decimals per fluid ton. In the case of cyanide solutions the man who is responsible for handling the solution has, as a rule, no means of weighing it, and can only measure it. This would certainly be more convenient for him, and save a great deal of calculation if the results were reported in "fluid tons of 32 cubic feet."

Mr. F. W. Watson (*Member*): I think assay values ought to be uniform, and reported by weight. I do not see that there need be much trouble in cyanide plants on this score.

Mr. H. A. White: There are other fluids to be assayed besides gold bearing solution, as for example, alcohol, which is frequently reported in volumes per cent., and if it is more convenient to give results with reference to volume I think this should certainly be considered.

Prof. J. A. Wilkinson (*Member of Council*): I am of the opinion that we should exercise a certain degree of care, since you will notice that both cyanide solutions "and others" are mentioned. Here, on the Rand, we are concerned, practically speaking, with no solutions other than cyanide. The specific gravity of these approximates almost exactly to that of ordinary water. In other metallurgical industries this would not necessarily be the case.

The President: You have heard the opinions expressed, and these will now be at the disposal of the sub-committee of the Institution of Mining and Metallurgy. The next paragraph does not interest us very much, and there is nothing of moment in the last paragraph.

THE ASSAY OF TIN ORES.

By Jas. Gray, F.I.C. (Member).

The increasing importance of the tin industry in the Transvaal has made the necessity for a quick and, at the same time, accurate method of analysing tin ores a matter of considerable importance.

The examination of tin ores may be considered under two headings, viz., the dry assay and the wet analysis, and I will first deal briefly with the former, which has been the method chiefly used in the past but which, under the best circumstances, cannot be considered to give any more than a rough guide to the tin contents of an ore.

The Dry Assay.—When the tin contents of an ore are low it is necessary to concentrate its value by vanning or some equivalent process, and this procedure opens the way to serious error. Pearce (*J.S.C.I.*, 1904, Abs., p. 273) found the losses on vanning an artificial sample to be 21·553%, this figure being the average loss obtained by three experienced vanners. Hancock (this *Journal*, Vol. viii., Abs., p. 155), however, gives the loss on artificial samples at 4 – 5·65%, and on natural samples from 6 – 9·1%. It is obvious, therefore, that a serious error results whenever vanning or similar operations are performed. The next procedure is to clean the above concentrate by boiling with acids. This operation is condemned by, among others, Collins (Trans. Inst. Min. and Met., May, 1904), who states that cassiterite is not altogether insoluble under such treatment. The concentrate is then dried, and a portion or all is fluxed and the mass fused in the assay furnace. Many fluxes have been suggested, but potassium cyanide is mostly favoured. When cyanide is used the mass is fused at a low red heat for about 15 minutes when the crucible is removed from the furnace, tapped to collect the tin at the bottom and allowed to cool. It is then broken and the metallic tin separated from the slag and weighed.

Those interested in the dry assay of tin will find a very interesting paper on the subject by Hoffmann in the *Chemical News*, Vol. 62, and further work on the same subject by Rennie & Derrick, in the Proceedings of the Society of Chemical Industry for 1892, pp. 662-7.

Fulton: "A Manual of Fire Assaying," p. 155, gives six reasons for inaccuracies in the fire assay of tin. The reasons are:

"1. Some of the tin, reduced in the assay from the oxide, is apt to be volatilised at the temperatures necessarily employed.

2. Metallic tin may be slagged by alkaline carbonates, used in some of the methods of assay, forming stannates.
3. Foreign metals present in the ore are apt to be reduced and enter the button
4. Sulphides present carry tin into the slag. If sulphates are present they are reduced to sulphides.
5. Silica and silicates, always present in the ore, even after very careful concentration, carry tin into the slag as silicate, while the SnO_2 passes* through the lower state of oxidation in being reduced to metallic tin.
6. The cassiterite before reduction is apt to combine with basic fluxes present in the assay, and be carried into the slag as stannates."

It is obvious from the above that serious objections can be raised to every single step in the procedure adopted for the dry assay of tin in ores and, *even should the result obtained happen to be correct, it is due to a balancing of errors.*

The Wet Analysis.—The chief difficulty in carrying out an analysis by the wet method is in effecting solution of the cassiterite. On reference to the literature available on the subject, I find that eight more or less distinct methods have been suggested. These are :—

1. Passage of a current of hydrogen over the finely powdered ore heated to redness, by which means the cassiterite is reduced to metallic tin, whereafter its solution is effected in hydrochloric acid.
2. Heating the finely powdered ore with zinc dust in a porcelain crucible and extraction of the reduced tin with hydrochloric acid.
3. Treating a mixture of the finely powdered ore and zinc dust with dilute hydrochloric acid.
4. Heating the finely powdered ore with sodium carbonate and sulphur, extraction with water and precipitation of tin sulphide after with acid (method originally due to Rose).
5. Fusion of the ore with potassium cyanide, extraction of soluble portion with water and solution of residue with hydrochloric acid.
6. Fusion with caustic alkalies and acidifying the dissolved melt with hydrochloric acid.
7. Fusion with sodium peroxide, extraction with water and solution in hydrochloric acid.
8. Hallet's method : fusion of the ore with KHF_2, extraction with sulphuric acid and precipitation as metastannic acid by dilution and boiling.

A modification of method 1 has been proposed by Muller (*Chemical News*,, Vol. 85, 1902, p. 147), who determines the loss of oxygen on passing hydrogen over the cleaned ore.

Fusion with metallic sodium has also been suggested as a satisfactory method of effecting solution. (Anon., this *Journal*, Vol. iii., Abs., p. 451).

I have experimented with nearly all the above methods, and my conclusions regarding them are as follows :—

1. The uncertainty which exists as to whether all the oxide has been reduced to metallic tin necessitates a retreatment of the residue in a current of hydrogen or a fusion with fusion mixture and borax, thus increasing the time necessary for the estimation. Parry states this method to be the most satisfactory but, in my opinion, the fusion or reduction of the residue which is necessary is a serious factor against it adoption.
2. Very satisfactory results were obtained by this method, and it is also very quick.
3. This method cannot be considered seriously as some varieties of cassiterite are under these conditions more soluble than others. Collins in his experiments found the percentages of cassiterite soluble to vary from 0·6 – 96·2%, depending on the source of the cassiterite. Hancock states that he found reduction to be assisted by allowing the hydrogen to accumulate under pressure, and suggests experiments along this line.
4. A refusion of the residue is also necessary in this case, and the method is likewise tedious and dirty.
5. The loss of tin as alkali stannate owing to the presence of alkalies in the cyanide is one objection in this case, and results obtained cannot by any means be considered satisfactory.
6. This method is very satisfactory, and my only objections are, the tendency of alkalies to froth over and the time necessary for solution of the cassiterite—a period of which Simpson (*Chemical News*, Vol. 99, p. 243) gives as 45 – 60 minutes.
7. Of all the methods tried this was the most satisfactory, 10 minutes being the maximum time necessary to effect a complete solution of the ore.
8. I have not experimented with this method, but it does not appear to show any advantages over those previously mentioned.

Either gravimetric or volumetric methods can be used for estimating the tin when obtained in solution, but there is no doubt that the gravimetric method is unsatisfactory owing to the precautions which have to be taken to prevent any other metals which may be present from scoring as tin. Parry (this *Journal*, Vol. X., Abs., p. 258) says "that the SnO_2 gravimetric assay is probably the worst analytical method in common use in inorganic analysis."

---

* *i.e.*, during the time that the tin is in the stannous condition. Ed. Comm.

There is no doubt that volumetric processes are much more reliable and accurate, and of the many known only the ferric chloride and iodine solution methods will be discussed. Though, no doubt, satisfactory in many cases, I find the ferric chloride fails, to a large extent, with Transvaal ores on account of arsenic being frequently present in them and the difficulty of removing it. I first endeavoured to obtain results by treating the ore with aqua regia, and although no signs of arsenic were observed on reducing the solution yet inaccurate results were obtained on titrating. Parry ("Assay of Tin and Antimony," p. 38) apparently found this to be the case also with the ferric chloride titration as after treating slags with acids, his procedure is to precipitate the tin as sulphide and redissolve before titrating with ferric chloride. Another drawback to the titration with ferric chloride is due to the presence of titanium in many of the ores (especially those from Swazieland), and I find that unless it be removed prior to the titration it scores as tin, but it can easily be removed by evaporating the ore with hydrofluoric acid. Personally, I have found the iodine titration to be very satisfactory, and although it is said that the weak point of the process is the assumption made that, after reduction with iron nails for 20 to 30 minutes, all the tin is present as stannous chloride, yet, when consistent results are constantly obtained, this objection cannot be fatal.

The influence of impurities on the results obtained by titration with iodine solution was also determined with the following results:—

With SnO_2 alone the quantity of iodine solution used = 36 c.c.

Arsenic.	Iodine Solution.	Remarks.
1·26%	36·15 c.c.	The arsenic was precipitated as a dark-brown or black precipitate which, however, did not obscure the end point.
6·31	36·2	
37·87	36·3	

Antimony.	Iodine Solution.	
11·9%	36·1 c.c.	The antimony was precipitated in lumps.
35·71	36·3	

Bismuth.	Iodine Solution.	
7·17%	35·9 c.c.	The metal was precipitated similarly to antimony.
21·31	36·35	

Molybdenum.	Iodine Solution.	
11·1%	36·2 c.c.	A chocolate coloured precipitate was formed.
33·3	36·2	

Tungsten.	Iodine Solution.	
10·4%	36·1 c.c.	A light blue coloration took place which did not interfere with the starch blue reaction.
41·6	35·9	

Titanium.	Iodine Solution.	
15% approx.	36·1 c.c.	The starch blue colour was quite discernible.

It will thus be seen that the effect of the above impurities was negligible even when present in considerable quantities.

With regard to arsenic, Ibbotson and Brearley (Thibault's Metallurgy of Tin), who reduce with powdered antimony instead of iron found that arsenic affected the iodine titration, they also state that reduction with iron has the same effect. In view of my results I am unable to confirm their statement.

Low recommends reduction with metallic nickel, but I cannot see that any advantage is gained by its use. Parry (this *Journal*, Vol. x., p. 258) also deprecates its use, especially if copper be present.

The method I adopt in carrying out tin analysis is as follows:—

From 0·5 gm. to 2 gm. of the finely powdered ore is fused with from 4-6 times its weight of sodium peroxide in an iron crucible and, after cooling, the melt is extracted with water. The solution thus obtained is transferred to a conical flask and excess of HCl added. The solution is then heated to about 90° C., reduced with iron nails and the reduction continued for 45-60 minutes after the solution has become colourless. The flask and contents are then rapidly cooled in a stream of running water, a piece of marble added to the solution to maintain a neutral atmosphere, and when cold a little starch paste added and the solution titrated with iodine solution, which has previously been standardised against pure tin, until the starch blue colour appears.

There is little or nothing new in these notes, but I hope they will lead to a vigorous discussion, which will result in a standard method being adopted for tin analysis in this country.

I have to thank Mr. Heymann for permission to carry out the necessary experimental work in his laboratories, and also Mr. Thorburn for his assistance during the progress of the experiments.

REFERENCES TO THE METHODS OF EFFECTING SOLUTION OF CASSITERITE IN TIN ORES.

Method No. 1.

McIvor, *C.N.*, Vol. 85, 1902, p. 179.

Hampe, *Chem. Zeit.*, 1887, p. 19.

Post, *Traité Complet d'analyse Chimique*, 1884, pp. 375.

Mitchell, *A Manual of Practical Assaying*, 1888.

Hiorns, *Practical Metallurgy and Assaying*, 1892, p. 349.
Rennie and Derrick, Proc. *Soc. Chem. Ind.*, 1892, p. 662-7.
Arnold, *C.N.*, Vol. 36, p. 238.
Beringer, *Assaying*, 1901.
McKenzie, Trans. *Inst. Min. and Met.*, Vol. xiii.. Nov., 1903.
Hancock, this *Journal*, Vol. viii., Abs., p. 155.
Parry, "Assay of Tin and Antimony," 1906.

Method No. 2.

E. Donath and F. Müllner, *J.C.S.*, 1888, Abs., p. 531.
T. Moore, *J.C.S.*, 1893, Abs., p. 501.
E. Donath, this *Journal*, Vol. viii., Abs., 1907, p. 56.

Method No. 3.

J. S. C. Wells, *Chem. News*, Vol. 64, p. 64.
Collins, Trans. *Inst. Min. and Met.*, May, 1904.
Hancock (see above).
Koomans, this *Journal*, Vol. ix., Abs., p. 30.
Parry (see above).

Method No. 4.

Post (see above).
Fresenius, *Chemical Analysis*, Vol. 11, p. 427, 1900.
Rennie and Derrick (see above).
Prost, *Chemical Analysis*, 1904.
Campbell and Champion, *J.C.S.*, Abs., Vol. 76, 1899, p. 62.
Mitchell (see above).
Abelspies, Trans. *Inst. Min. and Met.*, Vol. xiii., 1903, pp. 99-100.
Furman, "A Manual of Practical Assaying," 1903.
Parry (see above).

Method No. 5.

Crookes, "Select Methods," p. 378, 1905, due to Hart.
Thorpe, "Quantitative Analysis," 1900.
Abelspies (see above).
Parry (see above)

Method No. 6.

Burghardt, *J.C.S.*, Abs., p. 1,027, 1890.
Rennie and Derrick (see above).
Hiorns (see above).
Mitchell (see above).
Anon., this *Journal*, Vol. iii., 1902/3, Abs., p. 451.
Parry (see above).
Thibault "Metallurgy of Tin," 1908, p. 37.
Low, "Technical Methods of Ore Analysis," 1909.
Simpson, *C.N.*, Vol. 99, 1909, p. 243.

Method No. 7.

Anon. (see above).
Darroch & Meiklejohn, this *Journal*, Vol. vii., Abs., pp. 52-3.
Angenot, *J.C.S.*, Vol. 86, 1904, Abs., p. 784.

Method No. 8.

Crookes (see above).
Parry (see above).
Hiorns (see above).
Mitchell (see above).

Mr. A. F. Crosse (*Past-President*): I should like to propose a hearty vote of thanks to Mr. Gray for his interesting paper. I may say that these analytical papers are always of great value to our Society.

The President: We are indebted to Mr. Gray for bringing forward these notes. They are extremely interesting, and as he has brought forward his own results, they are all the more valuable. I hope they will be the means of educating the general public as to which is right and which is wrong in tin assays. We all know very well the discrepancies in tin assays which were reported some years ago. I trust that these methods will be tested by analysts here, so that we may be guided in judging which method should be used in assaying tin ores.

The vote of thanks was unanimously agreed to.

MINING AT THE REDJANG-LEBONG GOLD-SILVER MINE, SUMATRA.

By H. Philp (Member).

The Redjang-Lebong Gold-Silver mine is situated in the S.W. part of Sumatra, and transport to the mine is effected by means of native carts drawn by oxen and water buffaloes from the port of Benkoelen, 158 kilometers distant from the mine. Two ranges of hills are crossed en route, the highest point attained being 926 metres, the elevation of the mine above sea level being 400 metres.

The mine was located on information given by the "Redjangs" to a coffee planter; old galleries were fairly extensive in the outcrop of the reef, and extended to a depth of 10 metres from surface level. The native miners appear to have followed the lines of least resistance and greatest value, these two conditions frequently coinciding; the galleries generally followed vughs in the reef, or rich leaders which had become kaolinised.

The precise age of these old workings is not known, the local Redjangs expressing ignorance of their own past history, but from the number of graves around it is evident that a fairly large population formerly lived on the mine. Round stone hammers were frequently come across on the surface and in the old workings, also stone mortars, some round and others oval in shape, and measuring some 12 in. and 8 in. across the major and minor axes respectively. The metal was apparently recovered by bruising the ore down to a flour in these mortars with rubbing stones and then panning; some of the ore was amalgamated, a little quicksilver having been found near a creek just north of the mine. It would appear that the natives exploited the mine by building fires against the working face, drenching the heated rock with water and so splitting it, and then further breaking up and removing the rock by means of wooden wedges and stone hammers.

The geology of the district is as yet but imperfectly known, the exceedingly dense jungle and the lack of exposed faces, except in swiftly flowing mountain creeks, making examinations a trying and expensive business. From the little known, it would appear that the fundamental rocks of the district are schists and granite, these rocks showing as a mountain range a few kilometers east of a lesser range composed of rhyolites and andesites; it is on one of the hills of the latter that the mine is located. Overlying the schists and granite is a bed of shale, and above the shale there are conglomerates, tuffs and earths. The shales are of interrupted or local occurrence and would appear to have experienced considerable erosion. The deposition of the above beds has been followed by extensive earth movements and the intrusion of igneous rock, rhyolite, followed at a subsequent period by further disturbances which fissured the rhyolite and made passages for the inflow of andesite which, together with some of the over-burden of shale and conglomerate which has fallen into the fissures, constitute the reef. As the result of the igneous intrusions the shale has been altered to a hard black rock and considerably shattered, and it is found in the reef in all sizes, from huge boulders to minute grains cemented together by reticulated veins of quartz or a white mineral, calcium silicate, locally called Truscottite.

Waters rich in silica and minerals generally have come from below and cemented the breccia of andesite, etc., forming the reef body and building up definite quartz veins in the mass whilst at the same time depositing gold and silver. Calcium carbonate and siliciate have also been deposited throughout the reef as veins and masses. The reef outcrops on the eastern slope of a hill and strikes NNW.-SSE; the dip at surface is about 70° WSW, this diminishing at depth.

The ore body extended some 45 metres above the surface tramming level, below which it was worked as an opencut to a depth of 11 metres the ore being transported along the first adit level.

Three adit levels in all have been driven to the reef, the first is 185 metres long and 11 metres below surface level. The second 380 metres long and 35 metres deep, and the third 800 metres long and 64 metres deep. Through the third adit is discharged all the mine water, a fine Cornish pump with 20 in. plunger and 8 ft. stroke being installed at this level, the pump being driven by a 72 in. diameter Pelton wheel, with a 6 in. pin nozzle, capable of developing 120 h.p. at 220 revolutions per minute under a 300 ft. head. The adit as driven was framed with ellipital H steel frames of 120 mm. × 58 mm. section and measuring over the major and minor axes 2·4000 and 1·380 metres respectively, the average pitch of these frames was 90 cm., but in loose ground this was reduced to 25 cm. Behind the frames a complete lagging of 2 in. deals was driven. In places the ground proved very treacherous, twisting the frames into weird shapes and necessitating the substitution of rivetted steel piping 600 mm. diam. and $12\frac{1}{2}$ mm. thick.

The mine is divided into nine sections along the strike, each 50 metres long, the numbers run from − 1 at the south end to + 7 at the north end.

There are two shafts sunk in the footwall of the reef, the "Middle" shaft in No. 2 S section, and the "South" shaft in − 1 N section. The "Middle" shaft measures 2 metres × 1·50 metres in the clear, having two compartments each 1 metre × 1·50 metre; this shaft goes to the third level, and will probably be carried to the fourth level but not further; the hauling is done by a small steam hoist of about 20 h.p.

The "South" shaft measures 3 metres × 3 metres and has six compartments each 1·500 metre × 1·000 metre; there are two hauling ways, two pump ways, one timber and one ladder way. The weight of the cage and loaded lorry is 1,736 lb. and a 20 h.p. electric hoist with 1·5 metre drums is used; this will later on give place to a larger hoist of the same model. The footwall of the reef is all that could be desired, but the clay hanging wall and the great size and fractured nature of the reef body, which in certain sections is also very soft, has led to the devising of special methods for stoping it. A description of the third level working is typical of the whole. Parallel to the reef strike and a few metres back in the hard footwall rock a permanent main gallery, 2 metres × 2 metres, is driven at each level. From this gallery, at the third level, there is set out at every 25 metres along the strike a main cross-cut, 2 metre × 1·80 metre, which is carried through the hanging wall. The reef in the south end of the mine, subsections Nos. 1S, 1N, 2S, 2N, is hard quartz, but so full of "heads" and "slips" that at times it settles for from 20 to 40 meters of its length and for the full width of the reef, which is here 16 to 20 metres. The full width of the face is worked, and two 2 metre drives, running north and south, are left through the filling for the transport of the ore and the packing. The drives are kept open by putting in stacks, two metres square, of 20 cm. diam. timber on both sides of the road and by logs spanning the roadway. According to the nature of the ground these stacks are built close together, or from 1 metre to 1·50 metre apart; each stack costs 15s. and the road capping 5s. 4½d. Inside the stacks and between them is all tightly packed with waste rock, and these stacks are always built in as soon as there is soom enough for them.

The stopes are worked two metres high to enable the natives to bar down the rock freely and also to prevent accidents from falls of rock. As the "level" stope is worked out a new stope is started from a footwall winze beside the main cross-cuts, and stoping carried on as above described. There are two chutes and one travelling way in each cross-cut, the chutes being so arranged that the quartz can be dumped directly into them from the tram in the drives. Filling is obtained by cutting into the hill crossing the reef at the south end of the mine and also from a winze put up in the footwall in N3 section, to the top of the hill.

In sections 3, 4, 5 and 6 of the mine another method of stoping is employed. In this part of the mine much of the reef is soft enough to be worked with a pick, but it is broken up in places by masses and veins of hard quartz.

From the footwall gallery the main cross-cuts, 25 metres apart, are set off as in the south end and carried to the reef, a footwall drive in the reef is then put in connecting up the cross-cuts and winzes. The reef is now attacked by taking out a cut two-and-a-half metres wide across the reef and next to the cross-cut, next the cut a pillar of corresponding width is left, and then another cut put in, the filling of the first cut proceeding with the breaking of the second; as soon as the second cut is filled in the pillar of ore is taken out.

The cross-cuts in these sections measure 2·500 metres wide over the timbers, and are built in with these leg sets never more than a metre apart, thus if a slip knocks out a leg the remaining sets will support the set until a new leg is put in, The sets are two metres high so the cross-cuts are made 2·250 metres to get in the cap piece and 50 mm. planks; 150 mm. to 200 mm. round timber is used, a set costing 4s. 2d., and the planks 8s. 4d., allowing a few of the latter for sollars for the boys to shovel on.

In this part of the mine five boys will bore, pick and shovel ore for sixteen wagons, each of 0·37 tons capacity, per shift of eight hours. Very little drilling is necessary, as half the ground can be broken out by the pick, the other half is moved by drilling holes twelve inches deep in the quartz, and charging each with one-and-a-half small sticks of gelignite.

As the level stope is worked out we go up the winzes and drive north and south to get in the reef footwall drive again, after which cuts across the reef are stoped out as above mentioned.

A couple of boys will drill six feet in a shift, usually three two foot holes, one back and two water holes.

In general, three-and-a-half small plugs of gelignite are used in a hole, and four hammers give about 11·10 tons. In the drives and cross-cuts the Siemens and Halske electric drill with independent motor is used. Provided a sufficient number of the machines are run to keep a specially trained repair hand fully occupied, this drill does fair work and is well liked here. It takes very little power—about 1 h.p.—the safety fuses used to protect the motor burn when the load passes 1 k.w. or 1·34 h.p. Three boys operate each machine. The estimated cost of a drive 1·800 × 2·000 metres worked out as follows:—

	Guilders.
Explosives	22·42
Fuse and Caps	1·34
Power	1·66
Labour	8·00
Supervision	8·00
Removing Rocks	6·23
Repairs	10·00
Spare parts	4·00
Amortization	8·38
Drilling and Sharpening, Fuel and Steel	8·00
Oil, Waste and Sundries	1·00

Total cost per metre driven (1·800 metres × 2·000 metres × 1·000 metres) = 81·03 guilders, or 41s. 2d. per foot.

In drilling over a drive or cross-cut with machines fifteen holes two feet deep are put in, breaking down about three tons of rock; each hole receives 4½ plugs of gelignite.

A small air drill, made by the Dinsburger-Maschinenbau-Actien-Gesellschaft, Disnberg, at Rheim, is being tried with fairly satisfactory results; the machine is noticeable for its freedom from nuts and bolts. The cylinder = 55 mm. stroke 150 mm., weight of machine 85 kilos. On a seven days run, working twenty-one shifts of eight hours, twelve holes two feet deep were put down each shift; the air pressure at surface registered 60 lb., and at the drill 40 lb. The machine advanced four metres in a 2 metre × 2 metre drive, the ground was hard rhyolite with quartz and full of joints, bad ground to work in. The machine had previously been working in stopes for nine days and was put on this trial without any preparation; subsequently it ran for thirty days before being sent up in consequence of the small valve air feed pipe having choked. Two natives, receiving 8d. a day as pay, handle the machine.

There is a point in connection with the underground administration worth calling attention to, as it has been found of great value here both in helping to control the native labour and economising stores. Each European miner is given a private store underground, and on going on duty he rolls up all his boys, generally about 50, as experience shows this to be as many as a man can handle to advantage, at this store: the number on shift and the number of absentees is at once reported by messenger to the underground manager and thence to the compound manager. He then hands out the lamps and tools, etc., and later, should any extra supplies be required, the boys come to the store for them, a boy being kept as caretaker who reports to his boss. Here all old drills, picks, etc., are brought and exchanged, the old ones going to the surface at the end of the shift and a new supply brought down the next time that particular shift gang goes on duty. Thus there are no missing drills, picks, hammers, shovels and lamps as everything is checked at the beginning and end of each shift, and nothing new is given out except on production of what remains of the old. Lamp oil and machine oil is stored here, and explosives, fuse, etc., are safely locked away until actually required; here also the miner keeps a change of clothing and oilskins in case he should require them.

The European labour employed in the mine is very cosmopolitan, Australian, Scotch, German, Swedish and Dutch, the last predominating. The work is divided over three shifts of eight hours each.

The native labour is all imported from Java and is generally divided into Sundanese from the highlands of the north of the island and Javanese from central Java and the coast towns. Of the two types the Sundanese are of slightly bigger build and are more muscular and energetic. The average native is about the same height and build as an English schoolboy of 14 or 15 years of age, and their boyishness is accentuated by their favouring short trousers of linen and fancy print; a crowd of these boys in their wonderfully coloured suitings (a loose jacket and trousers of cheap linen or print) makes a very striking picture.

The boys are engaged on a three years' contract and receive 8d. a day as pay, and to this may be added 8d. more as the cost to the company for transport, food, housing, hospital, etc. A fair number enter on a second contract for a further two years, for which they receive 10d. a day and a cash present of 25s.

They are well housed and well fed; three days a week they receive fresh meat and on the remaining four good dried fish; they are fed three times a day, 5.30 a.m., 11 a.m. and 5 p.m., and on each occasion may take as much boiled rice as they care for; at the midday meal in addition to the rice and meat or fish they receive a good supply of excellent meat soup well thickened with fresh vegetables.

The nature of the work done by the boys and the intelligence displayed is much the same as that of the Kafir labour on the Rand.

The reef frequently attains a width of 50 feet, and averages about 1 oz. gold and 10 oz. silver per ton. The average working costs during 1908 were:—Mining, 5s. 11d.; development, 1s. 8d; mine maintenance, 10d.; surface tramming, 2d.; on an average monthly output of 7,481 tons.

In closing I wish to thank Mr. K. Kriekhaus, General Manager of the Redjang-Lebong Mine, for his kind permission to publish these notes, also Mr. Arthur Yates, General Manager of the Lebong Soelit Mine, to whom I am greatly indebted for advice and help in their preparation.

Mr. Palmer Carter (*Member*): I should like to move a hearty vote of thanks to Mr. Philp for his very interesting paper. It is such papers as these we are looking for, papers which give us the details of how things are done in other parts of the world. The author alludes to the native miners and the manner in which they recognise the value of "fracture planes." With regard to the native labour mentioned, it makes us all on the Rand envious when we find they can get boys on a three years' contract at 8d. a day, with only an additional 8d. a day for their cost in food. I should say there is a splendid opening on the Rand for a lot of those boys. We all appreciate this paper very much.

Mr. Tom Johnson (*Member*): I have great pleasure in seconding the vote of thanks.

The vote of thanks was agreed to.

TREATMENT OF ORE SLIME.

(*Read at November Meeting, 1909.*)

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

REPLY TO DISCUSSION.

Mr. A. F. Crosse (*Past-President*): The only remark made on my slime treatment process was a question by Mr. C. O. Schmitt regarding the cost of my plant.

The capacity of the cone for treating slime varies very much according to the nature of the material, but I have found that there is no slime more easy to treat than the tube mill slime produced on the Rand. I find, also, that I can treat a much larger amount per cubic foot of cone capacity of this slime than I could if I were treating an argillaceous slime.

From experiments on tube mill slimes, I can confidently maintain that I can treat three charges per 24 hours in one of my cones, but, of course, I must allow the pulp to settle in ordinary vats. So that, taking the cost of two cones large enough to treat 85 tons of dry slime per charge at about £750 each, this portion of the plant would cost £1,500. This is certainly much cheaper than anything that can be done by the decantation process, and will give a higher extraction of the gold contents.

ENDLESS ROPE HAULAGE.

(*Read at December Meeting, 1909.*)

By H. G. Kay (Associate).

DISCUSSION.

Mr. Tom Johnson (*Member*): Mr. Webb, in his contribution to the discussion of this paper, mentions that the side rope system adopted on the Premier Diamond Mine has the disadvantage, that if anything happens to the full side the empty side has to stop; this is true and is general in endless rope work. The only case I know of where this is otherwise is where two ropes are used on a dump, one for the full cars and a separate one for the empty cars. Considering that the Premier Mine handles, say, 1,000 cars per hour per rope (about 10 times the amount handled per rope by any company on the Rand) they do not appear to worry about this so-called disadvantage. As to the flanging, if any, it would be opposite to what Mr. Webb says, as a little thought will show. Further consideration would induce one to keep the rope a little wide, and to raise the near rail to counteract flanging, but in practice there is no more flanging with the side rope than with the top rope. Mr. Webb is wrong in saying a fixed type of jockey is necessary with the runners shown by Mr. Kay in his Sketch I.; a feather on the stalk will prevent the rope lifting the jockey out and still allow it to be lifted out by hand.

As to saving money on jockeys, I put in jockeys at the Rose Deep five years ago, which were fitted for the pins to be turned any amount found necessary, but the engineer assures me that they have not started to turn them yet, and the cost of the worn pins for five years has been nil. If jockeys are properly made, nearly any of the forked type will do for our sand dumps. Any failure in their use comes from making them of the wrong material and not properly understanding the principle of the rope drive. It is the twist given to the rope which should give the grip.

Mr. F. F. Alexander (*Member of Council*): I understand Mr. Johnson to say that the jockey pins on the Rose Deep have lasted for five years. I may say that that is not my experience. They last for about three months.

Mr. Tom Johnson: I am only going by what the engineer on that property told me.

Mr. F. F. Alexander: The author of the paper was correct in saying that these jockeys do flange the wheels considerably. In order to counteract that I have shifted the front wheels forward 4½ in., and by doing so the flanging is overcome. The wearing sockets described by Mr. Kay are a decided advantage.

Mr. J. E. Thomas (*Member of Council*): Practically the same device is in use at the Simmer Deep, but instead of the jockey being fixed on the side of the truck carriage it swings on a pin, forming part of the jockey arm, stepped into a hole in the end of the carriage. As the jockey jams on the rope it makes practically a rigid arm, thus pulling the truck parallel with the rope and not towards it, so that "flanging" does not occur to any greater extent than when the truck is running free.

FURTHER NOTES ON RAND MINING.

(*Read at February Meeting, 1910.*)

By Tom Johnson (Member).

DISCUSSION.

The Secretary read the following contribution to this discussion:—

Mr. H. G. Roake (*Associate*): What Mr. Tom Johnson says about the right and wrong way of building packs and pigsties is very good, particularly as regards the latter, but I do not agree with him as to the practicability of the former. I prefer relying on pigsties in a case of bad hanging, for the following reasons:—

Boys can soon be trained to build them, the green timber suitable is cheap and easily handled, small waste is more easily obtained than the large waste required for packs, and the building of packs in inclined stopes is slow and expensive. If a rock is built wrongly into the wall of a pack the wall will bulge and burst when the weight comes on, and the pack will degenerate into a heap of waste, and although the author has seen miles of walling built on the top side of drives I feel sure the building was done with white labour and not with black. To teach natives to do this work well requires time, and their contracts generally expire just as they are becoming efficient. Now, with pigsties it is necessary that the timbers should be square over one another at the corners and well wedged, but inefficient boys weeded out of the shovelling and drilling gangs can do the rest. If the roof is very broken and the pigsties fairly numerous those 6 ft. × 6 ft. in size will be found to give the best results, and there would be no objection to building packs all along and resting on the rise side of these. With a tender roof the author advocates putting on more machines, and, by pushing on the face faster, give no time for the roof to break. I doubt very much whether managers would agree to such a policy, as the number of machines available is usually limited, and, therefore, if one face is forced others must lie idle, a condition of things apt to upset the tramming and the grade of the ore sent to the mill. Again, with a tender roof a large number of machines, even small ones, working on the face would bring about conditions highly favourable to the collapse of such roof. In a case of this kind I would prefer putting in hammer boys till the necessary number of pigsties have been built, when the machines can return, or, better still, running it permanently as a hammer stope.

The driving to the boundary and working home system might be a sound one if the ore was payable throughout, but here and there blocks, at present considered unpayable, will have to be left behind to become gradually buried up, so that when a further reduction of cost renders these blocks payable the connecting levels will have to be reopened to render them accessible again, or, in other words, they will have to be developed twice over. To avoid such a contingency the mine should be kept open wherever possible.

THE TUBE-MILL CIRCUIT AND CLASSIFICATION.

(*Read at February Meeting, 1909.*)

By G. O. SMART (Member of Council).

DISCUSSION.

Mr. C. F. Bayly (*Associate*): Mr. Smart has given us a most interesting and instructive paper on the tube mill circuit and classification, and he has brought it forward at a most opportune time since day by day tube mills are playing a more and more prominent part in all our reduction plants.

In the clearest manner he has described the many advantages which cones possess for classifying and dewatering purposes when combined with the use of the diaphragm. As he points out the cones are operated with a considerable depth of sand. A cone 6 ft. in diameter and 9 ft. deep has a capacity of 85 cub. ft. To avoid the loss of time and disorganisation of classification entailed by emptying such a cone of its contents, and the difficulty and discomfort of affixing any device depending on a screw, the importance of some device to overcome these troubles is obvious. A device which we have found most useful, and which is in use on the dewatering cones at the Simmer and Jack East tube mill plant is a sliding gate for regulating or cutting off the underflow of the cones. A model is on the table for the inspection of the members. A short description of this device may be of interest. It merely consists of a slide made of iron suitably arranged, so that it can slide easily, in a cast iron casing which is attached to the bottom of the cone. In this slide there are one or more holes in which nozzles of different sizes may be placed, and by movement of the slide a different sized nozzle can, when necessary, be brought immediately into use. There is obviously no necessity to empty the cone, nor to change the flow of pulp into another cone, nor to remove the feed pipe to the tube mill, when a change of nozzle is desired. In the same way a blank in the slide completely cuts off the underflow. The nozzles merely consist of flat iron or steel rings which fit into holes in the slide, and are held in place by a shoulder. For ordinary purposes only two nozzles are necessary in the slide, and the better arrangement is to have one at each end and the blank in the centre.

The proceedings then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

FUSIONS FOR TRUE SILICA.—"Trouble is sometimes experienced in removing the melt from the crucible. In addition to this, considerable time is required for solution and the crucible is often injured. To obviate these difficulties I have adopted the practice of taking the melt from the furnace, and while it is at its maximum heat I pour it into about 50 c.c. of water in a beaker. This granulates the material and renders it quickly soluble. The thin skim of material left on the crucible is easily dissolved without injury to the platinum. There is absolutely no spattering or trouble of any kind in this operation, which is simply an application of the practice of granulating the sample of smelter slags used for analysis."—JOHN HERMAN.—*Western Chemist and Metallurgist*, Dec., 1909, p. 476. (H. A. W.)

Estimation of Molybdenum in Molybdenum Glance.—"The usual fusion with alkali and nitrate seldom effects complete decomposition of the mineral, however finely this be divided. Another possible source of error in the hands of all but the most experienced workers is the final ignition of the precipitated sulphide to trioxide, since the latter is volatile at a temperature but slightly above that necessary to convert the lower oxides into this form. Accurate results may be obtained if the analysis be conducted as follows:

The sample is reduced to the finest possible state of division, and about 0·3 grm. digested for two to three hours with fuming nitric acid. The nitric acid is then evaporated and the residue moistened with 1 or 2 c.c. of sulphuric acid and again taken practically to dryness, but without driving off all the acid. The residue is taken up with 50 c.c. of water, and the solution, which contains most of the molybdenum, filtered. To extract the last traces of molybdenum, the residue on the filter is washed successively with water, dilute ammonia, and again with water, and the filtrates united. Iron and aluminium are removed by treatment with excess of ammonia, and the filtrate concentrated to a volume not exceeding 200 c.c. Copper is removed by ammonium sulphide, and, on acidifying and boiling the filtrate, the bulk of the molybdenum is precipitated as sulphide. The filtrate from the latter is evaporated to dryness, the bulk of the ammonium salts expelled, the residue taken up with water and treated with ammonium sulphide, and then with hydrochloric acid, to recover the last of the molybdenum. The sulphide is dried, the filter carefully incinerated, and ash and precipitate heated with sulphur in a Rose crucible in a current of hydrogen, and weighed as MoS_2."—E. Collett and M. Eckardt.—(*Chem. Zeit.*, 1909, *33*, 968-969.)—*The Analyst*, Dec., 1909, pp. 540-541. (E. H. C.)

Rapid Method for the Estimation of Vanadium in Ores and Technical Products.—"From 0·5 to 1 grm. of the material is fused with seven or eight times its weight of sodium peroxide for 15 to 20 minutes, the cooled mass extracted with boiling water, and the solution filtered and acidified with sulphuric acid. Alcohol is next added, and then, disregarding any precipitate the alcohol may produce, sulphur dioxide is passed through the liquid until it smells strongly. The liquid is, if necessary, filtered, and then heated to expel alcohol and sulphur dioxide. Arsenic is separated as sulphide, the filtrate boiled to expel hydrogen sulphide, and the vanadium determined by titration with permanganate. The author always takes such quantities for titration that the titration liquid contains approximately 1 per cent. of vanadium, and he standardises his permanganate on a similar solution, made by boiling a solution of ammonium vanadate with soda till all ammonia is expelled, acidifying with sulphuric acid and reduction with sulphurous acid. The acidity and temperature, as well as the concentration of the titration liquid, should also be kept as nearly as possible the same as when standardising."—Piñerua y Alvarez.—(*Chem. Zeit.*, 1909, *33*, 1149.)—*The Analyst*, Dec., 1909., pp. 546-547. (E. H. C.)

Use of Dicyandiamidine for the Estimation of Nickel and its Separation from Cobalt, Iron, Chromium, Zinc, Manganese, and Magnesium.*—"Various modifications have been made in the methods of using the dicyandiamidine reagent since the various papers were published (*Analyst*, 1907, *32*, 273, 394). The weighing of the nickel as anhydrous sulphate has been abandoned in favour of the weighing of the anhydrous salt, $Ni(C_2H_5N_4O)_2$, obtained by drying the precipitate at 115-160° C. The precipitate is washed with a 3 per cent. solution of ammonia, as 100 c.c. of distilled water dissolve 0·0148 grm. at 15° C. In small volumes of solution (60 c.c.) precipitation is complete in fifteen minutes, and the solution need not be allowed to stand for six to twelve hours, as previously stated.

Nickel can be precipitated free from *cobalt* if 15 to 20 c.c. of 10 per cent. solution of cane-sugar are added before the solution is made alkaline. From the filtrate, which shows a reddish-yellow to red colour, cobalt is precipitated as sulphide, as electrolysis yields a metal containing carbon. *Iron* is best held in solution by the addition of Rochelle salt, 10 to 15 grm. of the salt being used for each of iron. Small quantities of iron are conveniently separated as basic acetate before precipitating nickel; *Chromium* if present as chromate, does not interfere with the precipitation of nickel; if present as a chromic salt, it may be oxidised as chromic acid with ammonium persulphate in dilute sulphuric acid solution; or Rochelle salt may be added as for iron; or else acetic acid (or formic acid) may be added, and the solution boiled for an instant, the resulting chromic salt being stable in the cold towards alkalies. If the last procedure is used, chromium can be precipitated in the filtrate by simply diluting it and boiling for some time. *Manganese* is best kept in solution by the addition of citric acid and a small quantity (0·5 grm.) of hydrazine sulphate; it can be recovered from the filtrate by adding hydrogen peroxide, or, if cobalt is also present, by treatment with hydrogen sulphide at 100° C. The methods for separating *zinc* and *aluminium* from nickel have not been altered. Small quantities of *magnesium* can be kept in solution by the addition of Rochelle salt and ammonium chloride while the nickel is being precipitated; in the presence of larger quantities of magnesium, however, it is necessary to precipitate the nickel first as sulphide, which is then dissolved and converted into the dicyandiamidine salt. Rochelle salt may also be used if *cadmium* is present to hold the latter in solution during the precipitation of the nickel. *Copper*, *mercury*, and *silver*, on the other hand, give insoluble salts with the dicyandiamidine.

The above methods of separation were successfully applied in the analysis of German silver, commercial nickel, nickel steel, and garnierite. In the analysis of the latter, iron, nickel, cobalt, and manganese are first precipated together as sulphides, leaving magnesium in solution. Iron is next separated as basic acetate, and nickel precipitated in the filtrate as above."—H. Grossman and W. Heilborn.—(*Paper communicated to the Seventh International Congress of Applied Chemistry; Chem. Zeit.*, 1909, *33*, 841-842, 851-852.)—*The Analyst*, Oct., 1909, pp. 455-456. (E. H. C.)

Heat of Combustion and Explosion Temperature of Explosives.—"In order to obtain the true explosion temperature and heat of combustion, it is necessary to know the composition of the explosion gases at the moment of completion of the explosion in the closed vessel. This differs considerably from that found on analysis, on account of the reactions taking place during cooling. The authors show that if A = CO_2, B = the hydrogen, C = the CO plus three times the methane, D = the water less the methane, all expressed from the analysis as gram-molecules per grm. of explosive fired, then—

---

* See this *Journal*, Vol. vii., p. 221.

$$\frac{(C+x)\,(D+x)}{(A-x)\,(B-x)} = K.$$

For temperatures about 2,000°, $K = 6{\cdot}6$ (Langen), or 5·5 (Mallard and Le Chatelier). The composition of the gases before cooling is thus obtained by deducting x from the gram-molecules of CO_2 and H_2, determined analytically, per grm. of explosive, and the addition of x to those of CO and H_2.

The true heat of combustion is calculated from that determined experimentally by deducting 50,600 calories for each gram-molecule of methane found, and 10,000 (Ostwald) or 10,100 (Berthelot) calories for each gram-molecule of CO_2 formed during cooling.

In order to obtain more accurate results, gravimetric methods are substituted for the usual gas analysis by absorption and measurement of volume. The sample is placed in the bomb described in the preceding abstract, usually 1 to 1·5 grm. of dry powder in a bomb of 10·1 c.c. content, or 10 grm. of other explosive in a bomb of 750 c.c. After sweeping out with dry CO_2-free air and closing the valves, the charge is fired, the bomb allowed to cool, and the gases—consisting of H_2O, CO_2, CO, CH_4, H_2, and N_2—passed slowly through a weighed $CaCl_2$ tube and potash bulbs. The mixture of CO, CH_4, H_2, and N_2 is then led over hot copper oxide in a hard glass tube 25 to 50 cm. long, connected to a $CaCl_2$ tube and potash bulbs. The CH_4 and (by difference) the CO are determined in a fresh analysis by leading the gases issuing from the first potash bulbs into an absorption apparatus containing cuprous chloride solution, which is renewed several times during an analysis from a vessel at a higher level, through towers containing dilute sulphuric acid (to absorb ammonia), concentrated sulphuric acid (to keep back moisture), and thence to the combustion tube. The CO_2, weighed in the second potash bulbs, will now be due to the CH_4 alone. If the two firings are carried out under exactly similar conditions, the results of the two analyses will be directly comparable. The gases are finally swept out of the bomb, $CaCl_2$ tube, and potash bulbs by a stream of dry CO_2-free air, which, to avoid oxidation of the CO absorbed in the cuprous chloride, has been passed through alkaline pyrogallol, and the rubber connection leading to the cuprous chloride appartus closed with a pinch-cock. By raising the level of the cuprous chloride solution the gases are driven over into the sulphuric acid towers, and thence, with a stream of dry CO_2-free air, into the combustion tube and bulbs. By means of a pump with a drying tube interposed, the $CaCl_2$ tube and bomb, which is warmed to 50° C., are evacuated, and the water remaining in the bomb thus transferred. The nitrogen is determined as described in the preceding abstract."—O. Poppenberg and E. Stephan.—(*Zeit. f. d. ges. Schiess- und Sprengstoffwesen*, 1909, *4*, 281 and 305.)—*The Analyst*, Nov., 1909, pp. 497-498. (E. H. C.)

Estimation of Free Lime in Cement.—"A solution of ammonium bromide in absolute alcohol is distilled in presence of the cement, the ammonia evolved being regarded as a measure of the free lime in the cement. The alcohol used must be carefully freed from water, by distillation over lime and subsequent standing over 'activated' aluminium, until evolution of gas can no longer be observed. Test analyses are cited showing that the method is accurate to within 0·1 per cent. of CaO. Ammonium bromide is not decomposed on boiling its alcoholic solution either alone or in the presence of calcium carbonate."—R. Brandenburg.—(*Chem. Zeit.*, 1909, *33*, 880.)—*The Analyst*, Oct., 1909, (E. H. C.)

Decomposition of Sulphates in an Air Current.—The results were obtained by heating in a porcelain combuston tube in a porcelain boat and measuring the temperatures with a Standardised Le Chatelier thermo couple. Air was passed in and the SO_2 and SO_3 collected in NaOHAq carefully neutralised with H_2SO_4Ag using methyl orange as indicator.

$FeSO_4$.—First traces of decomposition were shown at 550° C. Up to 580° C. there was slight increase which became rapid up to 600° C. when there was a sudden increase. On further heating to 650° C. only very slight traces appeared and none were noted between 650° C. and 960° C.

$CuSO_4$.—First traces of acid were shown at 400° C. On slowly increasing to 690° C. only slight amounts were noted but at 690° C. rapid evolution occurred, but up to 900° C. no further traces were evolved.

$ZnSO_4$.—The same phenomenon occurred, traces being evolved up to 730° C. and a rapid increase at 760° C., and none on further heating to 980° C.—W. S. Landis.—*Metallurgical and Chemical Engineering*, Vol. viii., No. 1, Jan., 1910, p. 22. (J. A. W.)

METALLURGY.

Cyanide Treatment of Slime.—"I was much interested in reading in the first number of this Society's proceedings a paper by Mr. Ferdinand McCann relating to what he has termed the 'Dilution' system of slime treatment, for he has practically described the treatment which has for some years been advocated by Mr. W. Leonard Holms and myself, though in the apparatus we have designed for the purpose we carry the system rather further than Mr. McCann suggests.

In 1903 we first patented our separator, which had for its object the satisfactory separation of cyanide solution from slime after agitation, by the continuous removal of the lower layer of settling slime from a vessel into which slime pulp was continuously running, clear liquor being allowed to overflow from an annular ring around the top of the vat. The thick slime removed from the first separator was diluted by mixing with weaker barren solution and then run into another separator, where the operation of removing a thick pulp, etc., was repeated. Following this another or several more separators could be used if desired, the process being a continuous one.

Since then several improvements have been made in the machine and three separators are now being installed at the 'El Bote' Mining Co.'s plant in Zacatecas.

As the process has been carried rather farther than Mr. McCann suggested, perhaps a short description of the separator plant might be of interest.

The separator consists of an ordinary cylindrical vat, around the central axis of which revolves a hollow cylindrical shaft, which carries at its lower extremity two or more strongly built arms, to the under sides of which are affixed blades, in such a manner that when the shaft is revolved the blades continually push the thick bottom layer of slime towards the centre.

Passing down the centre of the vat inside the hollow shaft is an iron casing containing two screw conveyors whose flights are intermeshed, the screws revolving in opposite directions. The lower flights of these screws are exposed, since the casing does not extend quite to the bottom of the vat. Consequently the thick slime pushed in by the blades is forced against the lower flights of the conveyor screws, and by them conveyed up the casing and discharged at the top of the vat. The slime as it leaves the top of the casing appears somewhat like a large sausage, and it contains from 26% to 30% moisture.

The strain on the arms is of course great, and they are therefore strongly built to resist the torsional strain and at the same time to cut through the dense slime as far as possible. The speed of these arms, however, is so slow (about 1 revolution per minute) that the actual HP required is very small, being in the neighbourhood of 1/50 HP per ton of slime per day.

The extraction of values will of course be very much greater than in the case given by Mr. McCann, assuming that the pulp is diluted to the same extent as he dilutes it, since the amount of liquid left in the slimes after removal from each separator is only about one-third of what would be present when removing a 55% moisture slime.

In our separator, however, we still further reduce the amount of value removed with the slime by allowing replacing solution to enter the vat a little above the bottom, so as to form a zone of liquor through which the settling slime particles have to pass before reaching the bottom of the vat. If a dense salt solution were used a very complete washing of the settling particles could in this way be obtained, but as in practice weak cyanide solution or water will generally be employed, a certain amount of diffusion takes place, and the amount of original solution displaced is not so great as it otherwise would be. At the same time, as a considerable portion of the removed moisture consists of replacing solution, quite a remarkable saving is thus effected.

By a simple automatic device the amount of replacing solution which enters the zone is equivalent to the amount of moisture which passes out with the removed slime, so that if the latter increases the amount of replacing solution is increased and vice versa.

As Mr. McCann suggests, there is no need to precipitate completely all the solutions, and there are several ways of arranging the flow of solution so as to obviate this necessity.

In our separator plant water, to replace that which goes away as moisture in the finally removed slime, and that which is lost by evaporation, is only added to the replacement zone in the last separator.

There is one point in connection with this method of treatment which I think Mr. McCann has not mentioned. This is the fact that in the treatment of the majority of ores, agitation with only one solution usually produces an inferior extraction of values compared to that obtained when successive fresh solutions are used. It is true that there is but little agitation of the slimes between each separator, but still it is mixed with fresh solution and in passing through the separators receives a certain amount of treatment, the beneficial effect of which should not be overlooked.

L. Mennell: The use of the proposed dilution system alone however, that is without conjunction with vacuum filter methods to complete the washing, as proposed by Mr. McCann, and using the Dorr Thickener diminishing the moisture in the pulp to only 55% I do not consider any improvement upon the best vacuum filter practice. The frequently imperfect washing of the cakes in the latter process is a source of undoubted loss, sometimes too considerable to overlook. Any new process which will diminish or eliminate this loss is cordially welcomed. The economy and despatch however with which the vacuum filter process is operated as also the small bulk of 'wash solutions' required to be sent to the precipitation boxes, make it a very practicable and cheap system for extracting the valuable solutions from the slime. This is indeed so much so that, notwithstanding the small losses in washing the cakes of some very 'slimey' ores, any new process devised to displace the vacuum filter method must also involve the handling of small amounts of solution for washing, storage and precipitation, or the advantages reaped from a more perfect extraction of the valuable solutions will be more than offset by the cost of manipulation, precipitation and loss in cyanide itself.

The 'principle' involved in the dilution system, that of fractional separations of the 'mother liquid' by repeated operations of thickening and dilution on the slime associated with it, each operation leaving less valuable solution in the pulp, is undoubtedly sound. Unfortunately, however, its application for the thorough and final washing of the exhausted slime, using even the best thickener *yet tried in practice*—the *Dorr*—necessitates the handling of enormous amounts of cyanide solution in the dilution operations; and it is precisely the difficulties and cost of manipulation of these heavy tonnages of solution, the precipitation of their low grade values, the storage of the various grades of cyanide wash solutions and the loss of cyanide by discharge of nearly an equal weight of weak cyanide solution to each ton of slime, which constitutes the draw-backs to the use of the dilution method and inferiority to the vacuum filter plant.

The scheme of operations favoured by Mr. McCann, which however in my opinion does not do full justice to the process he advocates as I will show below, requires an *enormous* tonnage of wash solutions to be constantly as above-mentioned manipulated, and a most careful supervision. After discharge from the agitation tanks and thickening by passage through the first Dorr Thickener, the pulp containing 55% cyanide solution is then diluted and thickened again *twice*, using the following tonnages of diluent solution; 1st dilution—549 tons cyanide solution with each 50 tons of dry slime amounting to 11 tons of solution per ton of dry ore; 2nd dilution—549 tons of weak wash solution, to the same tonnage of dry slime, equal to a 11 tons more of diluent per ton ore.

There is thus used in the scheme by Mr. McCann roughly some 22 tons (! !) of diluent cyanide solution per ton of dry slime in the dilution washing system, and this of course in addition to the tonnages required to be handled in the milling and cyanide agitation operations. This 22 tons of diluent amounts in a plant treating 300 tons per day to some 6,600 tons of solution to be pumped about, stored and approximately one half of it daily passed through the zinc boxes for precipitation (! !) Supposing the mill were crushing with 10 of weak solution per ton of ore—(not considering the latter's moisture), the cyanide operations performed with 3 of strong solution to one of slime, Dorr Thickeners used both before cyaniding and after, and then the above scheme of dilution separation for extracting the values employed: the total tonnage of solution to be handled per ton ore would amount to about 35 tons—(10 tons to mill, 1·78 tons to cyanide agitation tanks, 10·98 tons to 1st Dorr discharged pulp, 10·98 tons to 2nd Dorr discharged pulp and 1·22 of water into the system, equal in total to 34·96 tons)—This means upon a 300 tons per day plant that about 10,500 tons (! !) require to be pumped and in the neighbourhood of 4,500 tons of solution passed through the precipitation boxes! This will be seen to be an excessive amount, just about in fact twice the tonnage required where a vacuum filter plant is used. It is absolutely *necessary* to pass the above large amount of solution through the precipitation boxes also, otherwise its efficiency as a diluent ceases and the object of the process is defeated. The greater part of this solution would be strong, and in its passage over the zinc cause an excessive consumption of this metal.

One of the disadvantages of Mr. McCann's system is that the pulp as finally discharged from the last Dorr Thickener contains 55% moisture as cyanide solution. This solution would doubtless in practice be of a strength of from ·05% to ·12%; and the loss one ton of this per ton of dry slime means a cost of from 500 gm. to 1·2 k. of cyanide, worth from 35 cents to 84 cents (Mex.) per ton of slime. This is a considerable loss and is really equivalent to leaving in the cakes some 18 gm. to 25 gm. of soluble silver, per ton of dry cake, using the vacuum filter process. The latter value would be very bad work for any modern efficiently equipped vacuum filter plant, working upon ores of ordinary value, but the loss of equivalent value in cyanide of potassium by the dilution process *as outlined by Mr. McCann* would be *very* possible. Of course when first operations began, with water as the last diluent, this would not be the case; but the percentage of cyanide in the diluent would rapidly creep up and be only arrested at the cyanide strength of the mill solutions. The addition of one ton of water per ton of slime daily—(the difference between 55% moisture as discharged and the, say, 5% contained in the original ore)—to keep up the volume of the solutions, would have *no* washing effect when added to so much diluent to be even considered. It is no argument for this dilution system, when the above losses of cyanide are incurred, to compare it favourably with those resulting in the decantation system still unfortunately in use on some plants. The losses however in the latter—now out-of-date system—are a very excellent reason for the immediate adoption of a good vacuum filter plant.

As has been however above said, Mr. McCann does not in my opinion do justice to the process he advocates. He favours in his dilution an entirely excessive proportion of 10 of solution to 1 of the liquor associated with the exhausted slime. This amounts in his two operations of dilution and thickening to the use of 22 tons of diluent per ton of dry slime as above shown. If the value of the cyanide solutions before commencing these two dilutions were 100 gm. of silver: then the value of the solution after the second dilution or as discharged would be only 1 gm. Equally good 'practical' results might have been obtained by the use of dilutions of *3 of diluent solution* to *one of dry slime*—supposing the proportion were sufficiently thin, in the case of any particular ore, to enable 'easy settling' in the Thickener), but with the operations of dilution and thickening repeated *four times instead of twice only*. If the value of the solution were as before: 100 gm. silver per ton before commencing the dilutions, then after these *four* dilutions of *three* to *one of slime* the value of the solution as discharged would be 2·7 gm. per ton, and the tonnage of diluent used very considerably less, amounting to only *7·1 tons* of diluent solution as compared with Mr. McCann's scheme using 22 tons. If a thickness of pulp of 3 tons of solution to one of ore were unsuitable for good settling with some ores, then in those cases more diluent would have to be used, in no case exceeding 4 tons to one of ore. Pulp of a thickness of 3 to one of ore however, will usually settle all right, and consequently, 4 dilutions to pulp already containing 1·22 tons of solution—(or 55% moisture) needs the adding of only four times 1·78 tons or 7·1 tons in total for the whole four dilution operations. Such a considerably lessened tonnage of diluent solution to handle makes the system much more economical. Moreover the addition of one ton of wash water to the last (1·78 tons) dilution will be of real help in keeping the cyanide of potassium loss small and as a more efficient wash to the slime for its silver values also. Indeed fractional separation or washing is always more efficacious when performed a greater number of times with small dilutions—(greater however than the liquid to be diluted), than by using large dilutions once or twice only.

The process is by no means new. Messrs. Leonard Green and Leonard Holms some years ago invented a thickener with the precise object of separating the valuable solutions from the slime by this dilution system. Their thickener has never been tried on a large scale however, but tests which I made 3 years ago upon a 100 kilo model, in conjunction with Mr. Green, showed the pulp thickened to about 28% moisture or having about the consistency of sausage. I believe that *this* thickener would give much better results for the operation of this dilution system of extracting the values from the pulp, for the reason that each dilution given would be much more effective, indeed about twice as effective as when diluting the thinner pulp from the Dorr apparatus. Using this thickener, should it give a pulp in practice of only 28% moisture as above mentioned in experiments on a small scale, one ton of dry slime would thus only contain some *0·4 tons* of cyanide solution. It would thus be only necessary to give *two dilutions* of *three of solution to one of dry slime*, using the Green and Holms' thickener, to get the value of the solution assaying 100 gm. of silver as before, down to a value of 1·7 gm. (!!) This would be a *very good* practical result and the solution required to be handled as diluent would only amount to 2·6 tons in each dilution or 5·2 tons in all per ton of slime. The results which this thickener will give in practice are therefore to be awaited before the adoption of this dilution system can be recommended. If it will really give in practice a pulp with only 28% or 30% 'moisture,' then it will make very economical and efficient this dilution system for the extraction of the values from the exhausted pulp. Meanwhile, the Dorr thickener it has been shown, necessitates the handling of such large quantities of solution, that it would *not* be economical to apply the process completely but could only be done partly, for instance with one dilution of 3 of solution to one of ore, and the use of vacuum filters to complete the washing."—L. M. GREEN and L. MENNELL.—*Institutio Mexicano de Minas y Metalurgia*, Oct., 1909, p. 50. (H. A. W.)

MODERN CYANIDE PRACTICE AND MACHINERY.—Notwithstanding the widespread development the cyanide process has undergone since its successful *debut* at the Robinson Mines, the chemistry of the process has undergone little change. The Robinson plant consisted of a circular wooden solution tank, containing a 0·25 per cent. solution of potassium cyanide, which was connected by piping with a wooden percolation tank, into which the tailings were dumped for treatment, and a sump tank, where the auriferous solutions were stored before being treated in the zinc precipitation boxes.

The cyanide practice at the most modern mill on the Transvaal or Nevada goldfields does not differ in principle from that of the early Robinson plant treatment. Weaker cyanide solutions are now used, and in a few instances, other chemicals are added to the cyanide solution to counteract the bad effect of harmful minerals dissolved with the gold from the ores by the cyanide solution. But while the chemistry of the process has not changed, the detailed treatment has been revolutionized during the past few years.

Cyanide vats are now constructed on a large scale of steel and concrete, in place of the small wooden tanks used in former years. They are elaborately furnished with automatic machinery and appliances for charging and discharging the tailings, and for handling the various solutions. These developments have resulted in reducing the time occupied in getting the gold into solution to a minimum, and, consequently, in lowering treatment costs to very moderate figures.

Fine Grinding.—It is essential for rapid treatment that the cyanide solution be brought into direct contact with the gold. This can only be done by crushing the auriferous material as finely as possible. This was formerly done in stamper batteries and grinding pans. The coarse gold was extracted by amalgamation on copper tables, and by re-grinding the sands in fine crushing pans. Fine grinding is to-day done in a variety of mills of large capacity.

Agitation and Aeration.—In order that the gold in a finely ground product from tube, or other fine-grinding mills, may be quickly brought into contact with the cyanide solution, it is necessary to agitate the solution tanks. This is done in a variety of ways. The simplest method is to have the tanks equipped with a revolving spindle furnished with stirrers or paddle bars. The Traylor agitator, made by an American firm, consists of a tank containing a suspended pipe connected with the discharging orifice of a centrifugal pump. The bottom of the pipe contains a double T connection fitted with four smaller pipes arranged at right angles like the spokes of a wheel. These small pipes contain a number of small jet pipes which are inclined toward the bottom of the tank. When the pump is set in motion it draws the cyanide solution from near the top of the tank, through a small valve, and forces it back again into the tank through the suspended pipe and out through the jets in the smaller pipes at the bottom of the tank. The pressure of the pulp as it is discharged from the jets causes the agitator pipe arms to revolve slowly. The pulp flowing from the jets stirs up the pulp clinging to the tank bottom. This is a simple and effective mechanism for agitating, aerating, and mixing cyanide solutions.

For dealing with large quantities of cyanide solutions an appliance known in Mexico as a Pachuca tank, and in Australia as a 'tall tank agitator,' is largely in use. It consists of a cylindrical iron plate tank about 8 ft. in diameter and 35 to 40 ft. in height with conical bottom. A pipe, having a diameter of 1¼ in. for each foot of diameter of the tank, is suspended in the centre of each tank. The top of this pipe is open and reaches barely to the top of the tank. A small air pipe is run down the centre of the suspended pipe as far as the bottom. The tank is filled with the ground ore and cyanide solution. Air is pumped into the small central air pipe and allowed to escape at the bottom of the larger suspended pipe. This air lightens the water column inside the larger pipe and causes the solution to over-flow the pipe at the top of the tank while fresh pulp is sucked in at the bottom of the suspended pipe. This fresh pulp is carried up the pipe and overflows it. In this way there is a steady circulation kept up throughout the tank, and every particle of ore is constantly turned in contact with cyanide solution and bubbles of air. The tanks are automatically discharged at the bottom by flushing through a discharge cock.

Filtration.—In the early days of the cyanide process filtration was carried out almost solely by simple percolation in wooden tanks. This was always a slow and expensive method of separating the gold solution from the treated material; but it is only recently that better methods have become available. Filter-pressing was first profitably tried, and later numerous vacuum filters were introduced into cyanide practice. There are a variety of such filters now on the market. They are all simple, and are mostly of large capacity. The Butters and Moore filters are the most popular ones to-day. Continuous suction filters of the Ridgeway type are rapidly coming into favour.

Several modifications of the Ridgway filter are now on the market. A modification made by W. A. Caldecott is in very successful operation at the Simmer Deep mine at Johannesburg, South Africa.

The Imlay Process.—An important improvement in cyaniding has lately been patented by W. L. Imlay, of Philadelphia, Pa. In the Imlay process the sands are crushed to pass a 150-mesh screen and carried to a launder which feeds the top of an inclined lixiviating trough. The trough is from 150 to 200 ft. in length and contains a number of rolls and iron plates. When the pulp is fed at the top of the trough it falls by gravitation, and passes through the rolls and over the iron plates. There are about 50 plates and the same number of rolls. Each alternate plate is heated by steam. The rolls are revolved by sprocket wheel flanges and driving chain. The rolls not only agitate the pulp as it passes through them, but develop numerous bubbles in the cyanide solution which contain air with sufficient oxygen to re-supply continuously the excess of oxygen required to keep the cyanide in its most active condition.

The heating of the alternate iron plates is an important feature of this process. Every other plate has a water back on the under side so that steam can be admitted to heat a section across the width of the plate. The admission of steam is controlled by a valve. The steam enters on one side and discharges on the opposite side, and can be regulated to suit requirements.

By the time the cyanide and pulp have passed down the 150 ft. of the lixiviator trough all the values have been dissolved in the cyanide solution. The solution and the pulp leave the lixiviator at the lower end and enter a distributing launder. This launder can be tilted so as to deliver into either one of two or more sand tanks. The cyanide and pulp flow from the lixiviator to the launder, then to a sand tank where all the sand settles. As much more solution is used than there is sand, this tank soon overflows through a pipe into a slime tank. Both of these tanks are built on the lateral filtering plan, the slime tank having a filtering capacity equal to the discharge over the lixiviator. From the time the cyanide solution starts at the top of the lixiviator and dissolves the values out of the ore, is filtered, and the values are deposited in the zinc box, and the pump has returned the solution to the stock tank barely 10 minutes elapse.

This being a one-solution process but little water is required. Storage tanks for water, weak, medium, and strong solutions, and wash water are not needed. A standard strength of solution, and full quantity in stock tank is all that is employed.

The sand tanks are emptied by any of the usual methods." — A. Selwyn-Brown. — *Engineering Magazine*, Sept., 1909, p. 960. (J. A. W.)

A Simple Identification Test for Iron and Steel.—The Spark Test.—" The phenomenon of of the production of sparks in the grinding of iron materials by means of emery wheels is a familiar one. Depending on the size and shape of the metal and the pressure between the metal and the wheel, the sharp edges of the emery crystals detach chips of different sizes at a very high velocity, which are raised to a red heat by the transformation of mechanical into heat energy. The path described by the glowing chip from its origin to its extinction forms a line of light, which may be called the spark ray, of variable length depending on the mass of the spark. The size of the spark in turn depends on the grain of the emery, the cutting speed, the nature of the metal, and the pressure between metal and wheel. The initial temperature of the spark is influenced principally by the cutting power and cutting speed of the emery wheel, and in a minor degree by the nature of the metal to be ground.

It has remained for Herr Max Bermann. a Buda-Pest engineer, to observe that the sparks produced in grinding iron materials possess uniformly characteristic differences, which depend upon the percentage of carbon present and the inclusion of alloy materials in the metal. He suggests the spark test as a simple means of classifying iron and steel products of various compositions, and has evolved a theory of formation which seems to raise his suggestion above the plane of mere speculation. From a paper read at the recent Copenhagen congress of the International Association for Testing Materials, reprinted in *The Iron Age* for September 23, we take a few notes of the characteristics of sparks of various steels, an outline of Herr Bermann's theory, and his suggestions of possible applications of the test.

Examination of one of the longer spark rays of a soft steel, or wrought iron, shows a quite fair line of light, at the end of which is an extended drop, pointed in the direction of flight. The heat of this drop formation varies between a bright red and a white glow ; close observation shows that this pointed dark red end is extended to form a second much smaller drop. At the broadest point of the first drop formation, where it is of a white heat, pointed, prickle-like lines run out of it in all directions with almost explosive violence. The pointed branching lines, have, in the lateral projection, a forked appearance, while in the direction of their flight they look as if rushing from a very brightly glancing knot shooting forward in an explosion-like manner and disappearing in a moment.

This explosion-like appearance and articulation of the spark ray, to which the name 'spark picture' is given, is different for different kinds of iron, and is, in fact, characteristic for each. For all carbon steels the spark picture is a tuft of prickle-like lines. Steel with 0·07% to 0·08% carbon shows two or three lines which appear to start from different points of the shining drop. The number of branching lines increases with the percentage of carbon. At 0·25% to 0·28%, a crowd of lines appear to spring from a common point of the drop, and with still larger percentages the lines are proportionately more closely crowded over a larger part of the drop. The spark picture of crucible cast steel resembles the branch of a blossom and the individual branching lines have a lilac-like form.

From the number of prickle lines in the spark formation, therefore, the percentage of carbon in steel can be estimated and the different kinds of steel distinguished. The addition of other elements to the steel introduces variations in the spark from which the various alloy steels can be no less easily recognised. The presence of manganese is especially easy of detection. In the spark picture of manganese steel the individual branching lines are no longer pointed but in each case show a second explosion-like phenomenon, shorter lines collecting, like leaves around a central point. The extent and shape of the spreading ends of the primary branching lines appear to stand in connection with the percentage of manganese in the steel, the number of primary lines still indicating the carbon content. In the case of tungsten steel, the spark rays are dark-red, hatch-stroked lines which show little tendency to produce a spark picture until the steel is pressed hard against the emery. When the pressure is sufficient, the drop formation appears at the end of the spark rays and out of it spring the branching lines ; in this case the ends are neither pointed nor leafy, but take the forms of little shining balls which seem to be projected out of the drop, the branch lines being scarcely visible. The combination of spark rays and spark picture of chrome-tungsten steels is characterised by two kinds of ray, very thin dark red and thicker brick red, which are absent in the case of simple tungsten steel. The spark picture of these high-speed steels consists solely of short curved drop formations ; the long needle-shaped lines with clubbed ends which sometimes appear may be taken to indicate the presence of a subsidiary alloy metal. Nickel steels shows a spark picture similar to that of carbon steel, but without the regularity of the latter. The sporadic manner in which the spark picture appears is the distinguishing characteristic of the spark test for nickel steel. The spark pictures of the various kinds of cast iron are as easily distinguished as those of the various steels.

Herr Bermann's theory of the spark phenomena is based on two facts of which his observations have given ample proof. These are : '(1) at a certain point in its line of flight the red hot spark assumes a yellow heat ; it then reaches a white heat, thereby transforming itself in an explosion-like manner into the spark picture ; (2) at the moment of its explosion-like transformation the spark is in a fluid state. The increase of heat of the spark (where an extinction due to rapid cooling would more reasonably have been expected) is caused by an internal source of heat, represented on the one hand by the transformation heat of the amalgamated carbon at the critical temperature produced by cooling, and on the other by the combustion heat of the transformed carbon which in the *status nascens* suddenly burns. The heat of oxidation of the exterior surface of the mass of sparks acts in opposition to the too rapid cooling of the spark. The quantity of oxygen necessary for the combustion of the carbon is now supplied by the crust of black oxide of iron which, up to the point of critical temperature, has been formed.

'The heat of combustion of the carbon constituent provides the quantity of heat necessary for the fusion of the mass of sparks. As a matter of fact, however, this heat is not sufficient to melt the whole mass, because the amount of carbon is too small. A melting of the spark takes place when the mass of the unoxidised core within the oxidised iron crust is small enough to be melted by the quantity of heat available. In this case the combustion gases of the amalgamated carbon burst the outer crust of the spark mass and thrust out the fluid contents in the direction of the primary branching lines. The silicon and phosphorus therein contained also burn at the melting heat of the iron and raise the temperature of the fluid mass. From this explanation it follows that the size of the fusible spark increases with the percentage of amalgamated carbon, while the quantity of the fluid core and the tension of the combustion gases are correspondingly greater and the branching lines more numerous.

'The practical applicability of the spark test is manifold. The most important instances of it may be shortly given as follows: —

'1. The classification of the different kinds of iron in accordance with 'percentage of carbon' and 'principal alloy metal.' The ends of rods which may eventually have been wrongly arranged in the storing rooms of the iron working industries are made to bear against the revolving emery wheel. The shape of the spark picture gives unerring indication of the kind of iron to which the rod in question belongs. Pointed branching lines in the spark picture denote carbon steel—*e.g.*, Siemens-Martin steel: leafy ends of the branching lines a manganic steel (Siemens-Martin steel with larger percentage of carbon). Spark pictures with a blossom branchlike arrangement indicate ordinary tool steel; dark red stroked spark rays and shining points and little balls thrown out of the drop formation, tungsten steel; two different kinds of spark rays of dark red and brick red colour, respectively, with short unarticulated drop formation, high speed steel with chrome and wolfram as alloy elements; when explosion-like articulation is present, a secondary alloy metal (molybdenum, vanadium or titanium) is in question.

'2. The spark test is so sensitive that it gives strikingly clear indication of a difference of 0·01% of carbon, and it accordingly offers a very simple and rapid check on the chemical analysis in regard to the amalgamated carbon in steel, and also forms a suitable check on the Siemens-Martin process (in place of the forge test), particularly when analysed standards are made use of for comparison. The assay sample is chilled and the sparks compared with those of the standard. The carbon in the charge is then increased or lessened, according to the differences which show themselves.

'3. In the purchase of different kinds of iron the spark test supplies a sure means of excluding material which does not comply with the conditions. When a suitable sample is available, it is only necessary for the testing official to satisfy himself of its identity with the material offered by comparison of the spark pictures. The material for quality tests is chosen from charges which give spark pictures identical with those of the samples, and from those of which the spark pictures show considerable difference. At the test these latter will show themselves as not in accordance with the conditions.

'4. The spark test supplies a very sensitive means of establishing with certainty any differences there may be in the chemical composition at different points in the iron product, and also in finished objects and structural parts.

'5. The spark sheaf and the spark pictures provide the possibility of at once determining whether a cast iron is gray or white, and also of distinguishing shades of difference in the individual varieties. Fine dark red spark rays, spark pictures here and there, and lines collecting round the drop formation like a net, formed by the leaf like articulation lines, denote black or very dark gray cast iron; spark pictures corresponding with manganic steel, gray cast iron. The netlike lines retreat more and more with the increase of the assimilated carbon, and with light gray cast iron they disappear altogether.

'6. In shop work the spark test, when once introduced, is not easily done without. In the hardening room counsel is taken with the spark prior to the hardening of the tools (which are to-day made of steel of the most varied kinds), to avoid the occurrence of gross mistakes in the hardening temperature applied. The wire for the manufacture of different spiral springs is likewise examined with the emery wheel, to see whether it has the degree of hardness and other properties suitable to the purpose in view.

"In the forge the spark is applied to the distinction of good malleable wrought iron from such as is difficult to weld.'"—MAX BERMANN—International Association for Testing Materials.—*The Engineering Magazine*, Dec., 1909, pp. 262-265. (J. A. W.)

FERRO-MANGANESE IN THE BLAST FURNACE.—"In an article in *Stahl und Eisen* for July 21, O. Höhl communicates some results of experiments made by Josef Jakobi at Olchowaja, South Russia, on the production of ferro-manganese in the blast furnace. This is a branch of iron-works practice of which the literature is exceedingly scant; in addition to the interest which attaches to the record of remarkably successful results, the infrequency of such detailed communications as Herr Höhl's lends a special interest to his report of the Olchowaja experiments.

Two ores were available for the experiments, of the following compositions:

	I. Per cent.	II. Per cent.
Manganese ...	51·60	53·50
Iron	1·45	0·90
Phosphorus ...	0·173	0·25
Silica	7·80	6·20
Alumina	1·12	2·80
Lime	1·80	0·15
Magnesia	2·18	1·80
Baryta	2·20	2·60

The ores were washed almost entirely free of dust, in order to prevent loss of manganese in the form of loose dust. Being rich in manganese and low in silica, they were of first-class quality for the purpose, although the percentage of phosphorus was rather high. The maximum phosphorus content of the ferro-manganese had to be kept down to 0·40%, so that only a small amount of ore No. II could be used.

The desired percentage of manganese in the product was set at not less than 77%. It was calculated that 70% of the manganese in the charge would be recovered, and that the coke consumption would be 2·5 tons per ton of ferro-manganese. The manganese content of the charge was fixed at 83%; the charge itself was made up as follows:—ore No. I, 12,960 lbs.; ore No. II, 3,240 lbs.; hematite, 360 lbs.; limestone, 5,400 lbs.; and coke, 18,000 lbs. Of the manganese, 70% was to go into the product, and 10% into the slag; 20% was allowed for loss in dust and by volatilisation. The desired proportion of iron in the ferro-manganese was set at about 10·5%; 10% was allowed for loss of iron in slag and dust. With reference to the other elements, silicon, phosphorus and carbon, the charge was calculated so that the final product should contain: manganese, 83%; iron, 10·5%; silicon, 1·0%; phosphorus, 0·4%; and carbon, 5·5%. The yield expected from the charge was 7,200 lbs. of ferro-manganese, and 7,600 lbs. of slag (about 106% of the yield of alloy), of the following composition: silica, 26·8%; ferrous oxide, 1·4%; manganese oxide, 14·3%; alumina, 9·0%; lime, 36·7%; magnesia, 5.7%; and calcium sulphide, 12·6%.

The runs were made in an old furnace with four 6-in. tuyeres. The blast was heated in four Cowper stoves to a uniform temperature of 950° C. Previous to making the runs for ferro-manganese, the furnace was used for several days to smelt hematite and to produce 20% spiegeleisen. Thirty-six hours after charging, an alloy with 77·8% manganese was obtained. When the manganese had reached 81%, the intermediate iron, containing on an average 58·7% manganese, was remelted with the charge. The manganese content of the final product varied between 78 and 83%.

When uniform working had been reached, the blast pressure was raised to 16 in. gauge. In order to keep down the temperature of the furnace gases, the ore and coke were moistened and the upper part of the furnace was constantly sprayed with water. These precautions, combined with the rapid descent of the charge under high blast pressure, resulted in a reduction of the temperature of the gases to below 200° C. The usual ignition of the gases in the production of ferro-manganese in the blast furnace was entirely prevented.

Over the whole working period of nine days, the manganese content of the slag varied between 2·41 and 10·87%, corresponding to 3·1 and 14% of manganous oxide. An average sample of the slag made over the whole period had the following composition:

	Per cent.
Silica	30·32
Alumina	10·88
Lime	41·34
Magnesia	2·96
Ferrous oxide	1·41
Manganous oxide ...	8·52
Phosphorus pentoxide ...	0·01
Calcium sulphide ...	3·94
Baryta	0·48

The loss of manganese in the slag was about 6% less than had been anticipated. The total amount of alloy produced during the nine days was 904 tons. The composition averaged:

	Per cent.
Manganese	80·20
Silicon	1·16
Phosphorus	0·38
Carbon	6·46
Iron (by difference) ..	11·80

The total coke consumption was, roughly, 1,965 tons, or less than 2·25 tons per ton of alloy.

A manganese balance shows that 76·9% of the manganese charged was recovered, and 23·1% lost. The proportion of total manganese passing into the slag was 6·7%. The remaining 16·4% was lost in the dust and by volatilisation."—O. Hohl, *Stahl und Eisen.—The Engineering Magazine*, Dec., 1909, p. 276-277. (J. A. W.)

International Congress of Mining, Metallurgy, Applied Mechanics and Practical Geology, Dusseldorf, Germany, 1910.—This Congress, which was last held at Liege in 1905, will take place this year by invitation at Dusseldorf during the last week in June, 1910. The programme is an extensive one, embracing sections in (1) Mining, (2) Metallurgy (chiefly iron), (3) Applied Mechanics including Steam and Electrical Mining and Metallurgical Equipment, (4) Practical Geology, including Stratigraphy, Ore Deposits, Seismology, and Hydrology. An influential Council and Committee of organisation has been formed, composed of the leading mining, metallurgical and engineering men in North Germany, which almost anticipates a successful issue, more especially as the Congress is officially convened by the Rhenish Westphalian Mining Industry. The work of the Congress will be carried out, (*a*) in general meetings, (*b*) in meetings of the sections above specified, (*c*) by making visits to scientific institutions, industrial undertakings and excursions to districts of geological interest. The general Secretaries to the Congress have kindly forwarded a letter of invitation to any of the Members, etc., of this Society, who may wish to take part and are interested in its work. Membership is divided thus: (1) Honorary Members, by election, (2) Patrons by payment of £5 to the funds of the Congress, (3) ordinary Members by payment of £1. The condition of membership under (3) is the scientific or practical occupation in one of the four working branches. Forms of application may be obtained from the Secretary, Mr. F. Rowland, or from the General Secretaries, Dr. Schrodler, or Herr von Loewenstein, 3-5 Jacobistrasse, Dusseldorf, Germany. (J. A. W.)

American Cyanide Practice.—"The following is a summary of the principal details of cyanide practice in nineteen of the largest and most modern mills in the United States and Mexico.

The mills chosen by the author are:—

1. A Colorado mill, the name of which he is not permitted to give.
2. Combination, Goldfield, Nevada.
3. Desert, Millers, Nevada.
4. Dos Estrellas, El Oro, Mexico.
5. El Oro, El Oro, Mexico.
6. El Rayo, Santa Barbara, Chihuahua, Mexico.
7. Goldfield Consolidated, Goldfield, Nevada.
8. Guanajuato Consolidated, Guanajuato, Mexico
9. Guanajuato Development Co. (Pinguico mill), Guanajuato, Mexico.
10. Guanajuato Reduction and Mines Co., Guanajuato, Mexico.
11. Homestake, Lead, South Dakota.
12. Loreto (Cia Real del Monte y Pachuca), Pachuca, Mexico.
13. Montana-Tonopah, Tonopah, Nevada.
14. North Star, Grass Valley, California.
15. Palmarejo, Chihuahua, Mexico.
16. San Prospero, Guanajuato, Mexico.
17. San Francisco, Pachuca, Mexico.
18. Standard, Bodie, Cal.
19. Veta Colorado, Parral, Chihuahua, Mexico.

The tables following gives the details of the stamp mill equipment, tube mills, tables and vanners, collecting and settling tanks, leaching tanks, agitation tanks and filter presses, and a summary of the consumption of materials per ton of ore treated. Mr. Shaw's paper contains, of course, in addition to the tabular data here presented, a vast amount of information regarding minor points of practice at the various mills which we have not space to reproduce. In conclusion, however, he gives some details of extraction and costs which we may summarise briefly.

At the Colorado mill mentioned above the extraction is 94%. At the Combination mill the total extraction is 94·8%, divided as follows: plates, 42·5%; concentrates, 22·3%; leaching, 14%, and slimes, 16%. The total cost (given in detail by Mr. Shaw) are $4.04 per ton. Detailed costs are given for the Desert and Dos Estrellas mills, the totals being, respectively, $3.367 and $1.350 per ton. At the

Table I.—Details of Stamps in Modern Mills.

Name of Mill.	No. of Stamps.	Weight of Stamps, Pounds.	Height of Drop, Inches.	Drops, Per Min.	Duty per 24 Hours, Tons.	Screen Mesh.	Life of Die, Days.	Life of Shoe, Days.	Life of Screen. Days.
Colorado	60	1,050	6 to 8	100	3·8	26	50	112	3
Combination	20	1,200	6	108	4·5	...	74	96	10
Desert	100	1,050	6	104	4·79	12-14	59	76	30
Dos Estrellas No. 2 ...	120	1,250	6·5	102	4·2	16 & 26	65	65	2 to 5
El Oro	100	1,000	7·5	104	3·75	35			
	100	1,150	6	102	4·00				
Goldfield Consolidated ...	100	1,050	...	108	...	16			
Guanajuato Consolidated ...	80	1,050	7·5	104	3·6	50	...	...	30-35
Guanajuato Development (Pinguico)	40	1,050	6·5	104	6·25	2, 4 & 8			
Guanajuato Reduction ...	160	1,050	7·5	100	3·1	26			
Homestake	1,000	900	10·5	88	4·0	No. 8 slot			
Loreto	40	1,050	...	106	3·0	16			
Montana-Tonopah	40	1,050	7	100	3·5	20			
North Star	80	1,050	8	96	3·1	20			25
San Francisco	30	1,050	6·5	104	...	20			
Standard	20	1,000	4 to 6	96-106	2·3	30	57	122	55
Veta Colorado	100	1,050	7	...	...	8-10			

Table II.—Details of Modern Tube-Mills.

			Size.					Pebbles.		Lining.	
Name of Mill.	Make.	Number.	Diameter.	Length.	Rev. per Min.	Capacity per 24 hrs., Tons.	Power, h.p.	Kind.	Consumptn per ton Ore, lb.	Kind.	Consumption, Pounds.
Combination ..	Abbé	1	4 ft.	16 ft.	26	30	16	Danish	2·4	Silex	1·2
	Abbé	1	4 ft.	12 ft.	26	24	14				
Dos Estrellas ..	A-C.	5	5 ft.	24 ft.	26	121	55	Quartz	...	El Oro	0·84
El Oro	Abbé	2	4 ft. 6 in.	19 ft. 6 in.	31	100	50	...	8·4	El Oro	1·0
	Krupp	1	3 ft. 11 in.	19 ft. 6 in.	31	110	48				
	Krupp	1	4 ft. 11 in.	23 ft.	25	190	80				
	Krupp	1	4 ft. 11 in.	26 ft.	27	275	87				
Goldfield Consolidated	Gates	6	5 ft.	22 ft.							
Guanajuato Reduction	Abbé	2	4 ft. 6 in.	20 ft.		80	43	Danish	0·75	Silex	
Loreto	Abbé	2									
	Krupp	1									
Montana-Tonopah ...	Gates	2	5 ft.	22 ft.	27	52	42·5	..	2·22	Silex	
North Star	Abbé	1	4 ft. 6 in.	20 ft.	20	30	16	Quartz	40	Chl'd Ir	
San Francisco ...	Krupp	—	4 ft. 3 in.	13 ft. 1 in.	29	...	...	Danish	...	El Oro	
Standard	Gates	1	5 ft.	22 ft.	24	125	50	Danish	3·0	S. Steel	0·5
Veta Colorado ...	...	5	5 ft.	14 ft.	...	.	...	...	...	Ch. Stl.	

Guanajuato Consolidated, the crushing and concentrating costs are $0.55, and the cyaniding costs, $1.615 per ton, a total of $2 165. The total milling and cyaniding costs for this mill for the whole year 1908 are given as $2.30 per ton. The cost of cyaniding slimes at the Homestake is given as $0.2453. At the Guanajuato Reduction, the gold extraction is 88%, and the silver extraction 86%, the cost being $1.575 per ton. At the Montana Tonopah the total cost of treatment is approximately $3.50 per ton; the gold extraction is 94%. and the silver extraction 87%. The costs for crushing, concentrating and cyaniding at the North Star are $0.85 per ton.

Table III.—Details of Tables and Vanners.

				Recovery, P.C.	
Name of Mill.	Tables.	Vanners.	Concentration, Ratio.	Silver.	Gold.
Colorado	11	30	13-18 to 1		
Combination ...	.	7	40 to 1	22·3	
Desert	25	9	90 to 1	15·63	30·32
El Rayo	..	16			
Goldfield Consolidated	70				
Guanajuato Consolid.	16	13	50 to 1	50	
Guanajuato Reduction	32	18	...	31·0	28·0
Loreto	8	30			
Montana-Tonopah ...	8	16			
North Star	9				
San Francisco ...	6				
Veta Colorado ...	24				
San Prospero ...	8	...	230 to 1	24	

Table IV.—Details of Collecting and Settling Tanks.

Name of Mill.	Number.	Size, ft.	Total Capacity, cub. ft.	Capacity per 24 hr. cub. ft. per ton.
Sands—				
Colorado	3	40 × 6	38,247	165
	4	23 × 6		
	1	30 × 8		
Desert	4	33 × 7·5	25,659	80
Dos Estrellas ...	8	22 × 7	21,288	110
Guanajuato Reduction	3	40 × 8	36,159	139
North Star	6	14 × 10	8,640	170
Palmarejo	1	100 × 80	32,000	
Slimes, Settling—				
Combination ...	4	12 × 6	6,333	97
	2	16 × 9		
Desert	4	36 × 20	81,430	480
Dos Estrellas ...	cone	34 × 35	10,592	28
El Oro	4	36 × 20	81,430	215
Goldfield Consolidated	16	20·5 × 12	131,230	230
Guanajuato Consolidat.	24	9 × 9 × 8	18,432	125
	3	8 × 8 × 30		
Guanajuato Develop.	5	30 × 11·5	40,644	225
Guanajuato Reduction	3	18 × 22	16,797	70
Homestake	9	18 × 20	141,372	90
	9	26 × 20		
Montana-Tonopah ...	3	30 × 10	21,206	150

Table V.—Details of Leaching Tanks.

	No. of Tanks.	Size of Tanks, ft.	Total Capacity, cub. ft.	Capacity per 24 hr., tons.	Time of Treatment, hrs.
Colorado	3 4 1	40 × 6 23 × 6 30 × 8	38,247	165	
Combination ...	16	16 × 5	16,085	180	288
Desert	18	33 × 8	123,161	335	305
Dos Estrellas ...	12	36 × 5·5	67,176	385	184
El Oro	12	40 × 6·5	98,018	165	360
El Rayo	..	25 × 6	...	..	210
Guanajuato Consolidat.	7* 15	26 × 5 26 × 6	66,367	490	336
Guanajuato Develop.	14	29·5 × 5	47,845	190	
Guanajuato Reduction	15	40 × 8	150,795	580	480
Homestake	14	44 × 9	191,587	80	120
North Star	6	23 × 8	17,500	290	168

Table VI.—Details of Agitation Tanks.

Name of Mill.	No. of Tanks.	Size, feet.	Total Capacity, cub. ft.	Capacity per ton slime, cub. feet.	Time of Agitation, hours.
Combination ...	1 2	12 × 18 10 × 14	4,234	65	16
Desert	7	36 × 20	142,500	838	30
Dos Estrellas ...	12 6	36 × 10 36 × 20	243,296	750	13 6
El Oro	15	34 × 12	163,440	270	56
El Rayo	...	25 × 16	...	...	24
Goldfield Consolidat.	10	15 × 45	62,500	110	
Guanajuato Consolid.	14	30 × 10	98,960	660	300
Guanajuato Reduct.	13	36 × 12	158,789	660	72
Loreto	27	30 × 10	190.852	1,270	54
Montana-Tonopah ...	6	17 × —	...	...	32
North Star ...	2	8 × 7	1,500	30	
San Francisco ..	5	15 × 45	31,250	...	24
Standard	5 5	20 × 11 18 × 15	36,264	290	18
Veta Colorado ...	14	15 × 45	87,500	175(est)	

Table VII.—Details of Filter Presses.

Name of Mill.	Make of Press.	No.	Size of Leaf, ft.	Total Area, sq. ft.	Area per ton, sq. ft.	Time of Cycle, hrs.
Combination ...	Butters	54	5 × 10	4,708	72	227
Desert ...	Butters	172	4·3 × 9·75	14,448	84	180
Dos Estrellas...	Burt	70	28·3sq.ft	1,984	...	37
El Rayo ...	Butters	60	5 × 10	5,100	146	240
Goldfield Con.	Butters	336	5 × 10	28,560	50(est)	
Guanajuato D.	Butters	80	5 × 10	6,800	97	180
Guanajuato R.	Butters	75	5 × 10	6,375	40	90
Homestake ...	Merrill	2208	4 × 6	52,992	32	600
Liberty Bell ...	Moore	268	6 × 8	21,869	62	135
Loreto ...	Butters	104	5 × 10	8,840	74	120
Montana-Tono.	Butters	144	5 × 10	12,240	85	
North Star ..	Oliver	...	10 × 7 ft. revolv. drum	220	4·4	5·5
San Francisco	Butters	...	...	...	...	180
Standard ...	Moore	90	5 × 16	12,240	111	360
Veta Colorado	Ridgw'y	130	4 sq. ft.	520	1(est)	

Table VIII.—Consumption of Materials per ton of Ore Treated.

Name of Mill.	Die, lb.	Shoe, lb.	Tube-Mill Pebbles, lb.	Tube-Mill Lining, lb.	KCN, lb.	Zn, lb.	CaO, lb.	$Pb(C_2H_3O_2)_2$, lb.
Colorado ...	...	...	...	...	0·93	0·53	2·1	0·005
Combination	...	...	2·4	1·2	2·10	0·98	8·0	0·5
Desert ...	0·53	4·8	...	...	3·41	1·82	14·92	0·62
Dos Estrellas	..	...	...	...	1·17*	1·07	9·78	0·13
El Oro (sand)	...	...	8·4	1·0	2·8	1·2	12·0	
El Oro (slime)	...	...	...	...	0·75			
El Rayo ...	...	...	...	...	5·0	1·2	16·0	
Guanajuato C.	...	...	...	...	5·15	1·3	6·0	
Guanajuato R.	. .	...	...	...	1·70*			
Homestake (slime) ...	. .	...	..	..	0·31	0·13	4·5	
Loreto ...	..	...	...	...	2·2*	...	8·8	
Montana-Tono	...	..	2·22	...	2·8	0 97	9·2	0·28
North Star ...	...	...	40 0	...	0·5	0·33	4·0	
Standard ...	1·16	0·63	3·0	0·5	1·16*	0·6	7·5	

* NaCN.

—S. F. SHAW.—*American Insiitute of Mining Engineers, Engineering Magazine*, Sept., 1909, p. 987. (J. A. W.)

TEMPERATURES DURING CUPELLATION.—" R. H. Bradford (*Journ. Ind. and Eng. Chem.* I. p.p. 181-184) has made some investigations of the temperatures during cupellation. A Le Chatelier thermo-electric pyrometer with a couple of a platinum wire and a 90% platinum 10% rhodium wire was used in the experiments. The cold ends were kept at a known temperature by immersing in water along with a thermometer and were connected to a galvanometer of high resistance by copper leads. The couple was carefully calibrated and the calibrations plotted. To prevent the destruction of the wires when the junction was immersed in the molten lead, a thin coating of fire-clay was applied. The coating was made on the spot from moistened raw material and applied in a thin enough coat to offer but little interference with the surface oxidation of the lead button.

Lead buttons were introduced into cupels placed at varying distances from the front of the muffle and hence at varying temperatures to determine the minimum temperature of the lead button requisite for starting cupellation. Pyrometer readings of the heating metal and the cupel interior were taken in each instance. It was determined that the molten metal must attain a temperature of 900° C. or above before driving commences, or approximately the temperature of melting litharge.

Horizontal holes $\frac{1}{8}$ in. in diameter, were drilled to points beyond the centers of the cupels and at different levels in the different cupels, from directly beneath the bottom of the cup-shaped depression, to the bottom of the cupel directly above the muffle floor. The temperatures of the interiors of the cupels were then readily ascertained by inserting the junction of the couple. The temperature directly under the button, in every case in which the cupel had been properly pre-heated although lowering as the lead melted, rose to 900° C. or above before cupellation commenced. The temperature of the lower layers was usually a few degrees higher when the button opened. the melting metal apparently having taken its heat from the upper layers.

The temperature within the cupel and near the molten lead remained at 906° C. or above so long as the button continued to drive. A rapid lowering of the temperature of the interior was noticed if the button froze, showing that its heat was mostly derived from the heat of the oxidation of the lead. If a button was introduced into a cupel whose interior was well below 906° C. and the cupel and its contents pushed back into the muffle, the button began to drive when the heat radiated from the muffle roof, raising the litharge coating of the lead to a temperature of 906° C. The temperature of the button rose rapidly if allowed to remain in the same position, which temperature of course would cause excessive silver and gold losses. The formation temperature of 'feather' litharge was found to be below 906° Centigrade.

---

* Used for collecting tanks also.

The experiments established that when the driving is well started, the lead attained a higher temperature due to the heat of combustion, enough heat being developed to keep the button driving even though the cupel and its contents was drawn forward to a cooler position in the muffle. A temperature of 625 to 650° C. may be maintained a ¼ in. above and near the front of the muffle, with a gentle draught through the muffle, without danger of freezing, although the danger of freezing is lessened if the temperature is from 650 to 750°.

Cupellation will continue as long as the heat of combustion maintains the lead above the temperature of melting the litharge. Further litharge will form as long as the surrounding air is kept well below the temperature of melting litharge and will not disappear unless the temperature is raised above that point, althought a slight volatilisation takes place." —*The Engineering and Mining Journal*, Nov. 6, 1909. (W. A. C.)

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.*—" MR. EDWARD T. MCCARTHY: Though I have no special knowledge that the heavier is more efficient than the lighter stamp, yet assuming, for the sake of argument, that this is the case, there arises the question whether, in the case of gold ores where copper-plate amalgamation plays a considerable part in the economic treatment, the increased crushing capacity is not at the expense of a loss in the amalgamation. If this is so, then in working these free-milling gold ores, while there might be increased crushing efficiency, there would probably be a more than corresponding decrease in economic efficiency.

To explain what I mean: with the increased weight of the stamps there is not a proportionate increased area of screen discharge, and consequently no increase in the area of the plates where the discharge directly impinges—it is in this limited area that the plates are most efficient in catching the gold or silver.

If then, as must be the case, the velocity of the discharge is increased, there is a point, as has often been proved, where that efficiency of the plate rapidly falls away, and so far as my experience goes, no amount of plate-area added below this point seems to compensate for the loss of that impingement zone of amalgamation. If the above statement is correct, as I believe it to be, then, in the crushing of free-milling ores the economic efficiency of the heavier stamp is the moot point.

On the other hand, where tube-mills are employed, it may be that this loss can be compensated for by placing plates below their discharge. Such was the case in the British and Korean Corporation's Mill, put up in Korea to treat a very heavy pyritic (pyrrhotite) ore, and where the amalgamation was so good on the copper-plates below the tube-mills that I would not have erected the cyanide plant, had I been able to foresee this great saving of gold, especially in view of the fact that the mill was erected to take out the gold from a mine whose life could then be gauged as having practically come to an end.

Looked at then from a point of view where the ore is free-milling (or carries coarse gold) and does not call for tube-mills or cyaniding, there is much to be said in favour of the lighter stamp in conjunction with finer preliminary crushing.

As a matter of interest, about 25 years ago I ran the old Husband stamps with good results, my experience being, that provided careful injection of the oil into the cylinder was carried out every four hours there was neither heating or cutting of the cylinder. The objection to them was, that as this needed very careful manipulation too much depended upon the skill and care of the men in charge.

With regard to double as against single discharge—in my younger days it was a much controverted question, my strong impression is that those who advocated double discharge then did so more with the idea that it increased the amalgamating impingement zone than that it increased the crushing capacity of the stamps. But, in practice, as the back discharge was difficult of access, and the light bad under the ore-bins, the advantage of the heavier stamp in this respect could never have been thoroughly tested.

I would also like to point out that Mr. Bland (who was for some 30 years the successful manager of the old Port Phillips Mines, in Victoria) maintained that the old square head was equal to the round one, if not its superior. As he was a man who was remarkably in advance of his day, in keeping and tabulating exact data of all experimental work carried out, there may possibly be something in his contention: as, too, he had the old Cornish square-head, Australian and Californian batteries, running alongside each other, he was in a peculiarly good position to make comparisons.

MR. GILMOUR E. BROWN: One point, to which attention should be called, is that of the running weight of stamps.

Assuming that no compensating weights are used, what definite percentage of the original weight is meant by the running weight of a stamp?

Referring to p. 3 of Mr. Caldecott's paper, the running weight of a 1,350 lb. stamp, new, is given as 1,343 lb., 1,342 lb., and 1,356 lb., while again, on p. 7, the running weight of a 1,629 lb. stamp, new, is given as 1,520 lb., or a reduction of 6·7%, and stamps weighing 1,550 lb. and 1,350 lb., new, are given as 1,450 lb. and 1,250 lb., reductions in weight of 6·4% and 7·4% respectively.

It would appear that, in the examples from p. 7, Mr. Caldecott has allowed a maximum reduction in weight of 200 lb. in each of these cases, probably put down to wear of the stamp shoes.

The figures for stamp No. 1 in the following table, gathered by the writer with some care from a Western Australian mill, will serve to show that, in practice, the percentage reduction in weight may be greater than that which Mr. Caldecott's paper would lead us to believe.

Taking the extreme case of a worn-out shoe, the minimum length of stem allowable, that is irrespective of the state of wear of the shoe and die, and a well-scoured head, the reduction in weight is over 26%.

Assuming the running weight to refer to the mean falling weight, it would be, in this case, 1,033 lb.,

---

*See this *Journal*, vol. x., Sept., 1909, pp. 108-115.

PROPORTIONS BETWEEN WEIGHTS OF PARTS OF HEAVY STAMPS (NEW).

			New.		
No.	Weight of Stamp, New	Part.	Dimensions.	Weight.	Percentage of Stamp Weight, new
	lb.		in.	lb.	
1	1,190	Shoe	12 × 9 diam.	236	19·8
		Head	18 × 9 diam.	256	21·5
		Stem	204 × 3·5 diam. 3·87 for 2ft. above head	564	47·4
		Tappet	13·1 × 9·25 diam.	134	11·3
				1,190	100·0
2	1,537	Shoe	12 × 9·25 diam.	254	16·5
		Head	24 × 9·25 ,,	412	26·8
		Stem	192 × 4 ,,	681	44·3
		Tappet	14 × 10·12 ,,	190	12·4
				1,537	100·0
3	1,823	Shoe	14 × 9·25 diam.	292	16 0
		Head	32 × 9 25 ,,	566	31·0
		Stem	204 × 4 ,,	723	39·7
		Tappet	18 × 10·12 ,,	242	13·3
				1,823	100·0
4	1,998	Shoe	14 × 9 diam.	276	13·8
		Head	50 × 9·25 diam.	918	46·0
		Stem	156 × 4 ,,	548	27·4
		Tappet	20 × 9 94 ,,	256	12 8
				1,998	100·0

and the writer would suggest that this stamp, instead of being referred to as 1,190 lb. new, be described in literature relating to milling practice as 1,033 ÷ (plus or minus) 157 lb., the figure 1,033 being *the* important figure once the mill is in regular running order.

In the event of this stamp being provided with compensating weights, the designation would be different, and would depend entirely at what stage of the wear the first compensating weight was used.

The compensating weights might be used to keep the total weight up to the stamp weight, new, in which case the top hammer on the stem would be excessive, or they might be added to keep the weight constant at the mean falling weight, uncompensated.

The practice no doubt varies from mill to mill, depending on the ore and the millman, but there is a minimum falling weight for each class of stamp, which can be calculated, with some knowledge of milling, from the detail drawings of a stamp battery. Messrs. Fraser & Chalmers, Ltd., have kindly supplied the writer with details of their makes of heavy stamps (2, 3, and 4 in table), and, assuming wear on the same lines as in the case of stamp 1, the minimum falling weight of each has been calculated, and is given in the third last column of the table.

The average falling weight in stamps 2 and 3 is about 90% of the stamp weight, new, but stamp 4 is somewhat higher, and shows incidentally the advantage accruing from having the weight concentrated in the head, where it is most desired.

PROPORTIONS BETWEEN WEIGHTS OF PARTS OF HEAVY STAMPS (WORN).

Worn.				
Dimensions.	Weight.	Percentage of Original Weight.	Percentage of Stamp Weight, new	
in.	lb.			
1·5 × 9 diam.	44	18·6	3·7	Wear in Actual Practice.
*18 × 9 dia. from top to 12 in. down, 12 in. down to bottom, taper 7·5 in. diam.	238	93·0	20·0	
†175 × diam.	460	81·5	38·6	
as new	134	100·0	11·3	
	876		73·6	
1·75 × 9·25 diam.	61	24·0	3·9	Calculated Wear.
24 × 9·25 dia. and taper to 7·75 as in (1)	395	95·8	25·7	
†168 × 4 diam.	595	87·3	38·7	
as new	190	100·0	12·4	
	1,241		80·7	
2 × 9·25 diam.	66	22·6	3·6	
32 × 9·25 dia. and taper as in (2)	549	96·9	30·2	
†163 × 4 diam.	577	79·8	31·6	
as new	242	100·0	13·3	
	1,434		78·7	
2 × 9 diam.	62	22·4	3·1	
50 × 9·25 dia. and taper as in (2)	900	98·0	45·0	
†140 × 4 diam.	495	90·3	24 8	
as new	256	100·0	12 8	
	1,713		85·7	

These stamps would be classed as before, as 1,389—148 lb, 1,629—194 lb., and 1,856—142 lb. respectively, and then the mechanical engineer on the mine would have a poorer opinion of the friction losses and the metallurgist be more confident of his data in arriving at some relation between the weight of stamp and tons of ore crushed.

Discussion on paper by W. A. Caldecott.—*Institution of Mining and Metallurgy, Bulletin No. 63*, Dec. 9, 1909, p. 21. (H. A. W.)

MINING.

Development Work.—"If a mining venture is likely to prove payable, then the objective should be to win the asset as soon as possible, so that it can earn still more profit. If it is found not worth going beyond the prospecting stage, then the sooner that knowledge is acquired the better, for it is cheaper to do this quickly than to drag on indefinitely. At first, when opening up a mine, a limited number of men can be employed, until the main entrance, whether it be by shaft or adit, is sufficiently far advanced to warrant levels or cross-cuts being put in. The called-up capital is not earning interest while such development is going on. Therefore, a good organiser will push on with the dead work, even if he has to pay a bonus over and above a fair rate, so as to encourage the miners to work their best. This fact is recognised by the more up-to-date directors, and recently the managers of an optional mine were instructed to 'crowd on development work as much as possible, in order that we may not only obtain the most economical results (there is nothing so extravagant as development on a small scale), but also in order that we may obtain knowledge of the mine, and its possibilities, at as early a date as possible.' When a mine has once been opened up it is naturally cheaper to carry on development at the same time as productive work, than to work the mine to a temporary standstill, and then start developing again. In the former case development shares the standing charges, such as management, pumping, engine drivers, with the productive work. In the latter case development has to stand the whole cost. Not only that, but as a rule, fewer men can be engaged on development than on productive work, as there are more faces to be worked in stopes, than in drifts, winzes and shafts. The consequence is that the services of certain men have to be dispensed with, and labour conditions become disorganised. Then, again, if we look at it from a market point of view, this irregular way of working a mine gives to those who desire it every opportunity to run the shares up or down, irrespective of the real value of the mine. It is more advisable to push on with development, even if one has to borrow to do so; later on, ore can be got out quickly, or the output curtailed, according to the market price of metals or other conditions that may affect the management.' —(*Australian Mining Standard*), *Science and Art of Mining*, Sept. 18, 1909, p. 55. (A. R.)

---

* From measurements of many discarded heads.

† The distance from top of new die to top of upper guide is as follows:—(1) 15′ 2″ (2) 15′ 5″ (3) 15′ 5″ (4) 15′ 1″.

Electric Signalling in Mines.—" An electric signalling system has been installed in the Main and Edwards' Shafts of the Great Boulder Proprietary Gold Mine to a depth of 2,500 ft., with 30 polarised bells (29 levels) in the Main Shaft and 16 in the Edwards' Shaft; these have been working for the past 18 and 10 months respectively without causing a moment's delay, and have given entire satisfaction to the management. Subjoined is a brief description of the system, which is claimed to have many advantages over any previous system hitherto in use: the object is to provide by means of electric bells a simple and efficient means of signalling for the underground work. Alternating current is used instead of continuous current, and the system is as follows:

(*a*) For localities which supply such current a transformer, four insulated wires in armoured cable, watertight polarised bells, junction boxes, pulls or pushes, consisting of platman's pull and interplat pull, for platman to communicate with engine-driver and miners stationed at various levels, and *vice versa*.

(*b*) For localities without supply current, the same as the above, with the exception that, instead of a transformer a generator must be used. The working of the circuit wired to illustrate the actual procedure is as follows: Assuming a platman hauling at the 400 ft. level, and a shift boss at the 1,400 ft. level requires the cage, the shift boss simply rings the platman his number of rings by using the interplat pull corresponding to the level from which he rings. By so doing the bells are ringing at every level, excepting of course the engine-driver's bell; the platman replies to the shift boss by using the interplat pull with the same number of rings, and calls engine-driver by using the platman's pull for the level at which the cage is required. The engine-driver by using his pull replies to the platman's signal and lowers the cage to the signalled level. By this system perfect communication and understanding between levels will be obtained, and the engine-driver can secure the highest degree of safety. With the use of polarised bells the interrupter in the circuit, which causes so much trouble with the continuous current bells, is eliminated. By means of the transformer the high voltage current from the main supply is reduced to 25 volts, which is not dangerous to life. The saving of time in signalling allows more time for hoisting. The maintenance and working expenses are much lower than in any other bell system, and the possibilities of breakdown are practically nil. This system is undoubtedly far in advance of any other method of signalling.

The apparatus installed on the Great Boulder Mine is of the very best material, and every care was taken in fixing it. The main cable taken down the shaft is a submarine armoured one, made up as follows: From the outside jute, then steel armoured bonded, jute, prepared paper, lead sheathing, tape, and vulcanized rubber surround four insulated wires. Watertight cast iron junction boxes, measuring 15 × 11 × 5 in., are placed at every level on the centre leg of the plat-set. There are two watertight "Mix and Genest" pulls and one "Siemens-Halske" polarised bell placed in every level, a pull being placed on each of the legs about 7 ft. above the plat, and the bell on the cap-piece of the plat. A two-core No. 16 wire armoured cable (similar to the main cable) is used for wiring the plats from the pulls and bells to the junction boxes. The engine-driver's bell and pull are fixed on a board on the right-hand side, where the engine-driver is standing. The main conductors are tapped off from the supply main (550 volts), and are taken to a transformer (placed in the engine-house) which reduces the voltage to 25 volts. This voltage is quite sufficient to work these bells at any practicable working depth. A separate circuit connected with the same transformer has been installed for signalling from the braceman to the engine-driver, and *vice versa*, thus making the whole system complete.

The following are the working and maintenance costs: Transformer—No load losses per day = 0·24 units, or 87·6 units per year at 2d per unit = 14s. 7d. Assuming these bells work continuously for two hours per day, then 30 bells would consume 0·069 units, or 25·185 units per year at 2d. per unit = 4s. 2·3d. Maintenance cost = 10s. per month or £6 per annum. Total annual cost = £6 18s. 9·3d."—C. E. Grayson.—*Journal of Chamber of Mines of Western Australia*, Oct. 30, 1909, p. 293. (H. A. W.)

Stopping Leaks under High Pressure.—"At the 20th annual meeting of the Institution of Mining Engineers Mr. W. O. Wood described the method adopted at the East Pit, Murton Colliery, of stopping a leak in the tubbing of a working shaft. The point of leakage was 420 feet from the surface, and the quantity of water escaping 150 gallons per minute, under a pressure of 105 lb. per sq. in. The volume of water was not of importance as it was used as feed water for boilers underground, but it had been decided to replace steam by electrical haulage, which meant that unless the leak was stopped the water would in future have to be pumped at considerable expense. To accomplish this without stopping the pit in a coal drawing shaft working day and night was a difficult problem. It was decided to adopt the "Portier" method, which consisted of forcing Portland cement behind the tubbing, a method which had successfully been employed at French and Belgian collieries. The first attempt was unsuccessful as the leakage increased from 150 to 300 gallons per minute, an increase attributed to the erosive action of the sand, gravel, and cement forced through the sheeting composing the joint. Finally brown paper cartridges 4 in. in length by 1 in. in diameter were prepared, a leaden bullet 1 in. diameter forming the nose, while the body was filled with bullets, $\frac{3}{4}$ in. in diameter. The cartridges were injected to fill up the cracks and openings, and afterwards $\frac{1}{8}$ in. to $\frac{1}{4}$ in. gravel was passed in. This greatly diminished the flow, and finally an injection of sand and cement had the effect of entirely stopping the leakage."—*Science and Art of Mining*, Oct. 2, 1909, p. 73. (A. R.)

MINING TERMS.

Term.	Location.	Meaning.
Boxhole		Opening from level to stope above.
Bull ring		Collar of drill hole left unbroken by blast.
Back stope		Overhand stope.
Boy	South Africa	Native worker.
Cheesa stick	South Africa	Spiral strip of blasting gelatine wrapped round small stick, for lighting fuses.
Cap		Detonator.
Coca pan	South Africa	Mine Truck.
Change house		House for miners to wash and change in.
Dry ,,	Cornish	House for miners to wash and change in.
Floating dyke	South Africa	Dyke with same strike as reef and with varying dip, appearing and disappearing in workings.
Kraust	Cornish	Underground lunch.
Lash	South Africa	To shovel.
Lashers	South Africa	Shovellers.
Mine Captain		Underground manager.
Machine	South Africa	Rock drill.
Machine man	South Africa	Rock driller.
Piccanin	South Africa	Young native boy.
Raise or rise		Narrow development working towards the rise.
Stick	South Africa	Explosive cartridge.
Skoff	South Africa	Food.
Swallow	South Africa	Aerial conveyor in stopes.
Shift boss		Foreman of mine shift.
Vamping		Fine dirt under rails in levels, etc.

—*Mines and Minerals*, Oct., 1909, p. 138. (A. R.)

ELECTRICITY IN THE COAL MINING INDUSTRY.—"Electricity is practically independent of distance. There is no other form of energy that is transmitted as efficiently and conveniently as electrical. The coal-cutting machine is no longer an experiment. A practical electrically-driven drill has been devised. Also a puncher, which can be operated either by direct or alternating current, has been manufactured, and is now operating on a commercial basis. Fans, which the mine owners were originally afraid to drive by electricity, are now replacing the steam-driven kind with their long and uneconomical and expensive steam line. We also have electrically-driven tipple machinery, and the electrically-driven pump in wet mines. Some of our most satisfactory box-car loaders are electrically driven, while at many plants the coke-drawing machines are also electrically operated.

The power house of to-day is rapidly evolving itself into an alternating current proposition, becoming the central source of power for a number of mines. The difficulties that attended the initial introduction of alternating current have been largely eliminated, with the result that the polyphase system, which means either a two-phase or three-phase system, of alternating current transmission—with its adaptability to large units and long distances—has proved itself a commercial success.

The modern switchboard will be equipped with its oil circuit breakers and oil switches, its various safety devices, together with meters, including voltmeters, ammeters, frequency meters, power-factor meters, etc. We must not forget our exciter set, consisting of a direct current compound-wound generator, usually wound for 125 volts and used for excitation of the generating units. This exciter set may be either engine or motor-driven. We also have transformers, which are used to raise the original generator voltage to the proper transmission voltage. In addition, there are installed lightning arresters, choke coils, disconnecting switches with their details, for protective apparatus, so that we now have our alternating current power house most complete.

Among the principal advantages of a central power station there is the great saving in copper, and also the maintenance of a steady-working voltage—especially where the workings are located considerable distances from the power house. The saving in copper more than balances in many instances the entire cost of the sub-station during the life of the equipment. In transmitting the power, alternating current is used. The main power house generally furnishes current at approximately 2,400 or 6,600 volts, although 3,300, 13,000, and other voltages are used. This current being for power service, should be a 25 cycle, 3,000 alternation, polyphase circuit, preferably three-phase. The current is transmitted from the power house to sub-stations, where it is transformed to a lower voltage, and then in turn by means of a rotary convertor or a motor generator is tranformed from alternating into direct current.

The lowering transformers are generally arranged to transform from 2,400 or 6,600 volts high tension to approximately 440 volts low tension. The approximate ratio of the alternating current to the direct current voltage is about 7 to 10. If the alternating current voltage on the rotary converter is approximately 160 volts, then the direct current voltage will be around 250 to 275 volts. Generally speaking, it is considered advisable to put in lowering transformers, even with motor generator sets, as that makes it safer for the operator. A further advantage is that the lowering transformers act as additional protective apparatus—having the high tension and low tension sides separated. This arrangement also has the additional feature that taps may be taken from the transformers to operate independent alternating current circuits. The substation equipment requires practically no attention, and is almost automatic, easily adjusting itself to the various loads and conditions. Sub-station equipment requires no special foundation and anchorings. It is also true that the floor space required is considerably less than an engine and generator unit.

It is feasible to operate two circuits at a mine. Both alternating and direct current are used. The alternating current will operate the fans, crushers, tipple machinery, and pumps. Alternating currents have the additional advantage that any number of alternating current voltages may be obtained from the same generator, it being only necessary to use the proper transformer taps. The direct current will operate the locomotive and the direct current cutting machine. By this means, the operations are independent of each other. The sub-station, therefore, feeds both kinds of current simultaneously to the same mine.

As to mine locomotives, it is a better practice to have two 10-ton or two 13-ton, or even two 15-ton locomotives, which can be operated either separately or together in what is known as a 'tandem arrangement,' than to have a locomotive of 20-ton capacity or more. Too heavy a locomotive means heavy iron for the track, and such an engine is often unwieldly and inefficient.

There are also the reel and the gathering locomotives This latter type, known as the 'traction reel' or 'crab type,' does not go into the room at all, but a wire cable of from three-eighths of an inch to five-eighths of an inch thick and about 500 ft. long, is easily pulled in the room and fastened to the loaded car, which is then pulled out by means of a separate motor with its equipment of controller, resistances, hose and wire and other details. Both of the gathering types of locomotives are now being built so as to use one of the main hauling motors of the locomotive for gathering also.

As regards the cost of operating with electric haulage, it is obviously difficult to give exact figures which will apply in all cases, but it has already been pointed out that electrical haulage is cheaper than animal haulage. The life of the locomotive in the first place is longer than that of the mule, there being locomotives to-day in actual service which were installed 10 or 12 years ago. Against this, two or three years is the average life of a mule. The lengths of the haul and speeds required are also in favour of the locomotive.

The motors used for the operation of fans must be perfectly reliable, as on the successful operation of these depend the lives and efficiency of the men underground.

As regards the form of drive for fans, that is, whether the fan should be geared or belted or chain-driven, that should be carefully studied in each instance. Where there is a great fluctuation in in voltage, either a belted or chain drive is desirable. If floor space is an item, then between these two the chain drive is the proper one. If the voltage is constant and good regulation secured, then a geared fan is proper. Coupled fans can also be seriously considered, and these are in actual operation. Variations in speed can be obtained by either variable-speed motors, or in the case of belted machines, different pulley combinations can be used.

As regards the direct-current voltage in the mine itself, there has been, and still is, considerable discussion. Many are of the opinion that the voltage should not exceed 275 or 300 volts. On the other hand, there are just as many, if not more, who consider 550 to 600 volts not excessive. Certainly there is no getting away from the fact that with the higher voltage, there is a great saving in copper, and also that in most cases a better working voltage will be maintained. Looking at the question from the standpoint of safety, it is true the higher voltage is more likely to kill a man than the lower. But the point is to keep away from the wire. Especially in high coal this should easily be arranged. In many instances the trolley wire can be placed from 5 to 7 ft. above the rail, and should always be placed from 4 to 8 in. to the outside of the rail. With ordinary precaution this should be entirely safe. Where the coal is low, it might then be considered proper to put in the lower voltage, although it is by no means considered absolutely necessary.

It has been pointed out at various times by some operators that they consider the higher voltage safer than the lower. Their argument is that with the lower voltage the men get familiar and careless. On the other hand, they have considerable respect for the higher voltage, and are more careful and have fewer accidents. Local conditions will largely determine what voltage should be installed.

No matter whether 250, 275 or 550 volts is used, the conductor should be bare, uninsulated wire. Insulated wires in a mine are treacherous. The insulation will not last in a mine more than six months or a year at most. It deteriorates rapidly, and becomes a menace instead of an additional safeguard.

A system of wiring that meets with much favour where conditions will permit is the three-wire system. By this means 500 volts can be taken from the generator or rotary converter and divided in half, using two circuits of 275 volts each.

The three-wire system was originally introduced in order that a saving in copper might be obtained. The system has become in general use for both lighting and power distribution. It permits of a saving of 25% in copper in the feeder, when figured on a basis of current capacity; or a saving of 62% when figured on a basis of drop of potential, when the neutral wire is considered as equal in size to one of the outside wires.

Three-wire generators may be operated in parallel with each other, or in parallel with two-wire generators if required. By means of the three-wire system, it is possible to use the higher voltage; that is, 550 volts, for the stationary motors, such as pump motors, and 275 volts for the locomotives, considering the track as the neutral and return."—F. C. ALBRECHT, West Virginia Coal Mining Institute, *Electrical Engineer*, Oct. 22, 1909, p. 575. (J. A. W.)

MISCELLANEOUS.

PROTECTING STEEL FROM CORROSION.—"The use of steel in coal mining operations has been objected to on the ground of the presence of acid found in the waters. This condition obtains also, and probably to a larger extent in the copper mines, and in iron mines carrying pyrites. Such steel as has already been installed in the mines of the United States has given satisfaction without any serious corrosion, and without any protective treatment of the steel other than the use of good preservative paint. In wet mines in England steel girders are frequently tarred before being put in, but the actual loss from corrosion is so small as to be a minor quantity, even when the steel is not painted. In dry mines there is not much danger of any serious corrosion, but in conditions subject to moisture, and to varying temperatures, attention must be paid to this point. In my opinion true economy will be found in the painting of all steel for underground mine operations with one shop-coat of good paint and with at least one field-coat. If these are well worked in, the steel should need no further attention for years.

Careful investigation has been made as to the kind of paint which may be depended on for service in contact with acid waters, and I have conducted laboratory experiments to demonstrate the fitness of different pigments for this purpose. I agree with the conclusion reached by A. S. Cushman, of the Scientific Section of the Paint Manufacturers' Association of America, that steel should not be painted with carbon paints in the manufacture of which sulphuric acid has been used, and the use of coal-tar products is therefore to be avoided. The natural carbons, such as graphite and hydro-carbons, such as asphalt and gilsonite, may be recommended for the second-coat work if properly ground and mixed with a good vehicle. For the first coat, pigments should be used of an inhibitive character. The oxides of iron, such as Venetian reds, are usually manufactured by chemical processes, and their use is to be avoided, though a good natural oxide of iron paint may be used under dry atmospheric conditions. For first-coat work zinc chromates should give good service, but probably the best to use is red lead and oil, which has been demonstrated to be a first-class pigment for preservative treatment in most situations. The red lead should be pure and mixed in the proportion of not less than 15 to 16 lb. of red lead per gallon of oil. The oil should be pure, and in my judgment raw oil is better than boiled. The matter of vehicle is of some importance, and care should be taken to see that the pigment is thoroughly mixed before application. Red lead is a heavy pigment and settles quickly, necessitating the mixing of a new lot if it has stood any length of time. The settling may be retarded by the use of barytes or 'asbestine,' and the use of the latter is to be recommended by reason of its permitting a firm hold on the pores of the steel. For first-coat work, therefore, a mixture of red lead, oil, and asbestine, in the proportion of at least 15 lb. of red lead and 2 lb. of asbestine per gallon of oil, with sufficient japan dryer to work well, may be recommended as probably the best which can be had under present conditions; and for the second coat either silica-graphite or Mexican graphite; both coats to be well worked in, and the first coat be thoroughly dry before the application of the second. The theory on which this preservative treatment is recommended is based on the use of a practically inhibitive pigment to prevent the inception of corrosion in the steel, and the use of a second coat to protect the first atmospheric and temperature conditions, and to fill up thoroughly any vacancies or voids which may occur in the first. Insomuch as the particles of the red lead and the graphite are extremely small, the combination of these two coats should give a film practically impervious to atmospheric moisture, or to acid-bearing waters.

We live in an age of steel. The large demands made on our timber resources and the increasing scarcity of timber suitable for the heavier constructions of underground mine-work, make the consideration of a substitute a matter of vital moment. Experience in varied lines of construction has shown steel to be the one material known to the engineering world which by its flexibility and convenience of use can replace wood fully and satisfactorily. It can be perfectly adapted to all phases of construction; it is easy to obtain and convenient to fabricate and erect, and its long life under all conditions of moisture temperature, and stress amply compensates for the increased cost of its first installation. Reduction in weight, economy in erection, the possibility of re-use, and its fire-proof character, are but a few elements of advantage. The application of steel to mine timbering is but in its infancy. Considerations of economical operation will necessitate larger use of steel in the future, and the mine owner will be wise who looks beyond present first cost to ultimate economical operation and steady uninterrupted returns on investment."—R. B. WOODWORTH.—*Mining and Scientific Press*, Oct. 23, 1909, p. 560. (W. R. D.)

"ALUMINIUM is used in iron and steel works for removing oxygen from the oxides of iron and other substances, the heat generated being so great as to raise the temperature of large bodies of iron. It also has the power of combining chemically with the gases imprisoned during the cooling of the metal, thus preventing porosity. For these purposes the metal is either used in the form of an alloy known as 'ferro-aluminium,' or as the pure metal, either in the granulated or bar form or in small pieces weighing uniformly one-eighth ounce or one-fourth ounce each. This property of keeping molten metal hot or of raising the temperature of a molten metallic bath has been utilized in the thermit welding process, invented by Goldschmidt. In this process aluminium and iron oxide are intimately mixed in a finely divided state and ignited by means of a fuse. The heat of combustion in the ensuing reaction raises the temperature of the casting to the welding point. The reaction takes place in a funnel-shaped crucible, from which the fluid metal resulting from the reaction is run into a suitably shaped mould formed around the area of the joint to be made, which is pre-heated by means of a blow-lamp to avoid chilling the first lot of metal coming through."—R. B. WOODWORTH.—*Mining and Scientific Press.*—Oct. 23, 1909, p. 560. (W. R. D.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

MODERN PRACTICE IN MINING.—By R. A. S. REDMOND REDMAYNE, M.Sc., M.Inst.C.E., M.I.M.M., F.G.S., His Majesty's Chief Inspector of Mines. Vol. II.—The Sinking of Shafts. 7s. 6d. (London: Longman's Green & Co.)

"Shaft sinking—as is well known—is one of, if not the most difficult problems of the mining engineer and the colliery proprietor. Manuals by recognised authorities have not been wanting upon various colliery and mining plants. Shaft sinking has had ample attention, but until this volume, 'The Sinking of Shafts,' there has been no work or reference without interpolation of kindred mining subjects, and the very important one of sinking has not hitherto received the full attention it deserves.

Professor Redmayne has recognised the want and admirably filled it by one of the most clearly written treatises yet put before the engineer. The work in question cannot from its clearness and ampleness but be of very material value to the student, the practical collier wishing to increase his knowledge, and to even skilled colliery managers. There is very little question that in rectangular shafts there can be shown mathematical and practical losses of strength, and it is somewhat singular that at present new rectangular shafts are being sunk to considerable depths. Weakness at the corners, indeed in every part, of rectangular shafts are not uncommonly shown; many such shafts twist, numbers are lost, where had they been circular, the latteral pressure would have been more distributed, and a collapse prevented.

The tables of costs of shafts, forms of contractors' agreements, specifications and descriptions, together with details of the varied systems are clearly set forth in the volume under notice, thus affording easy reference which, before deciding upon the special form of sinking to be employed and during the actual progess of the work, cannot fail to be of great use. As a reference book for students the work must command attention, and every college and colliery owner could, with benefit, place the work in the library or the office.—*London Mining Journal*, Oct., 30, 1909, p. 187. (A. R.)

A TEXTBOOK OF ORE DRESSING. By ROBERT H. RICHARDS. Pp. 702. Illustrated. $5. (New York: McGraw-Hill Book Co.)

"This work, by the well-known author of 'Ore Dressing' is the outgrowth of a demand for a textbook for college and other use that covers the subject of ore dressing in a single volume. The work is mainly a condensation of the author's previous work. Typical machines and processes only are described, and the data has been generalised, as far as possible. To broaden the sphere of the book a chapter has been added on coal washing. The subjects are treated from the standpoint of a teacher. The book is divided into six parts, with 18 chapters and an appendix.

Chapter I. —General Principles, gives a comparison of ore dressing and smelting, and the physical properties of minerals are applied to ore dressing.

Part 1, comprises a chapter on Preliminary Breaking, including blasting in the mine and coarse crushing.

Part 2, Final Crushing, comprises five chapters, as follows: Rolls, Steam Stamps; Gravity Stamps and Amalgamation; Grinders other than Gravity Stamps; Laws of Crushing.

Part 3, Separating, Concentrating, or Washing, comprises eight chapters, as follows: Preliminary Washing and Hand Sorting; Preparation of the Crushed Ore for Concentration; Principles of Screen Sizing and Classifying; Coarse-Sand Concentrating; Fine-Sand Concentrating; Slime Concentrating; Miscellaneous Processes of Separation.

Part 4, Accessory Apparatus, is covered by one chapter.

Part 5, Mill Processes and Management, consists of two chapters, as follows: Mill Principles and Processes; General Considerations.

Part 6, Coal Dressing, consists of one chapter, dealing with the practice of coal washing. In the Appendix are given some important tables.

The book is well adapted to the needs of students preparing to enter work in the mining or metallurgical field, and to those already engaged in milling work."—*The Mining World*, Nov. 27, 1909, p. 1084. (W. A. C.)

THE GEOLOGY OF CAPE COLONY, an Introduction to. By A. W. ROGERS and A. L. DU TOIT. Cloth, small 8vo. 500 pp. Illustrated. (London: Longmans, Green & Co.) Price 9s For sale by *The Mining Magazine* (London).

"This is a second edition of a work that appeared five years ago, and contains much new matter incorporating information obtained by recent examinations and investigations. The book belongs to the 'South African Science Series,' which was inaugurated to encourage the study of natural science in South Africa, and the authors are members of the Geological Survey of Cape Colony.

The ground covered consists of a description of the general geology and paleontotogy of the colony; ore deposits and economic geology do not receive much attention. From a mining point of view, Cape Colony does not present any features of great interest with the exception of the copper district in Namaqualand and the diamond pipes of Kimberley. These are not treated in this book. There are many occurrences of coal in the Karroo formation in Cape Colony, but they are of no great thickness, and most of them are spoilt for fuel purposes by the dolerite intrusions that have coked them and mixed them with ash. There are extensive deposits of iron ore in Griqualand, between Prieska and Kuruman; these beds have been formed by the solution of the limestone underlying ferruginous sandstones, the fracturing of the latter, and the replacement of silica by hematite by circulating waters. The formation varies from banded ferruginous sandstone to pure hematite, and there must be enormous quantities in existence, but the distance from centres of civilisation and absence of fuel make the ore practically valueless. As a contribution to the literature of the geology of South Africa, this book is valuable." —*The Mining Magazine* (London), Nov , 1909, p. 247. (W. A. C.)

VAN NOSTRAND'S CHEMICAL ANNUAL, 1909. Second Issue. A Handbook of Useful Data for Analytical, Manufacturing and Investigating Chemists and Chemical Students. Edited by John C. Olsen. 12s. 6d. London: Constable & Co., Limited. New York: D. Van Nostrand Co.)

"Those who possess the first (1906) edition of this work will welcome the appearance of this second issue, which is fully up-to-date and in which all factors dependent upon atomic weights have been recalculated from the International atomic weights of 1909. Many new tables have been added, and a useful list of books and of articles which have appeared since 1906 have been added in the form of appendices, totalling over 100 pages.

Valuable alike to the metallurgical chemist and assayer, to the organic chemist and to the physicist, its 576 pages contain all the usual tables for which one searches in a laboratory guide, and many others which are both useful and unexpected. A search through its pages will show that many of the factors, etc., which the worker has been in the habit of calculating for himself are there given with their logarithms, and that the tables of factors for calculating percentages, etc., from analytical results have been so amplified as to enormously enhance their value.

The molecular weights, etc., are commonly calculated out to the second place of decimals, so that those who pine for an accuracy in their calculations which is unattainable in their practical work, may obtain their wish; and although it is, perhaps, an unnecessary refinement to go beyond one decimal point, it is, at any rate, an error on the right side.

Valuable tables of the physical constants of both inorganic and organic substances, including the elements, their principal compounds, and the most important alloys, occupy about half the volume.

Important tables on the heats of combustion of various elements and combustible materials, and of the calorific values of typical fuels, are given with a concise and unusually clear preamble, stating what the various heat units in vogue really mean and how they are correlated.

A good index, a valuable adjunct even to a systematically arranged series of tables, has been added, and it may, perhaps be permissible to suggest that so useful a feature might well be amplified in the next edition."—*London Mining Journal*, Dec. 24, 1909, p. 505. (A. R.)

MODERN MINING PRACTICE.—By GEORGE MITCHESON BAILES. Five volumes each about 250 pages. Ill. (Sheffield: J. H. Bennett & Co.) Price: £2 12s. 6d. For sale by *The Mining Magazine.*

"This book deals in an encyclopedic way with the practical details of coal mining, and provides information on theoretical subjects requisite for the intelligent study of underlying principles. It forms an excellent course of instruction for students desirous of qualifying for responsible positions in coal mines. The practice quoted is chiefly British, but Continental and American methods also receive attention. The author for 15 years was lecturer on mining to the Warwickshire County Council and is now principal of the Mining Engineers Correspondence School at Sheffield.

The first chapter deals with such branches of chemistry and physics as are of use to the mining engineer. The second discusses coal-dust, its dangers and prevention, and the third gases encountered in coal mines, also explosions and their results. Chapter 4 deals with methods of ventilation and describes various sorts of fans, and Chapter 5 discusses the theory of friction of air-currents in mine-passages. Chapter 7 describes different forms of safety lamps. The above chapters are all contained in Vol. I.

Vol. II. includes chapters on conducting air to the working face, the removal of gas, mine-fires, breathing apparatus, first aid to the injured, diseases of miners, geology applied to mining, explosives and blasting, prospecting and boring. Vol. III. contains further chapters on boring, which is treated very fully and clearly; also chapters on modern methods of shaft-sinking under all sorts of circumstances and through all sorts of ground. Vol. IV. deals with methods of developing and working coal seams, timbering, coal-mining machines, and rock-drills, while Vol. V. treats of boilers, engines, electric machinery, winding, pumping and hauling. Coal washing, coking, and surveying are mentioned, but very little information is given, presumably because these subjects are fully treated in other publications and are not strictly mining. The chapters on boring and shaft-sinking constitute the best parts of the books for the general reader."—*The Mining Magazine* Oct. 1909, p. 161. (W. A. C.)

GOLD REFINING. By DONALD CLARK. 8vo, 127 pp. (Australia: Critchley Parker, *Australian Mining Standard.* London: Sir Isaac Pitman & Sons.)

Mr. Clark, in this volume, brings together a deal of information relative to this subject, as well as to parting. Apparently all known methods have received consideration, those of present commercial importance being described in detail. It is particularly interesting however, because it embodies, besides descriptions of standard practice, accounts of considerable experimental work, carried out by the author in his search for improved methods, quite a large proportion of the space available being devoted to this. The various methods of treating the gold precipitate obtained in cyaniding are illustrated by examples taken from the practice of representative mills and the use of sodium bisulphate or nitre cake, both as a source of acid in the wet treatment of cyanide precipitate, and as a refining agent during melting, is fully discussed. Much of the experimental data is relative to this point.

The style of writing might with advantage have been a little clearer in places, while the printing and proof-reading leave something to be desired. The illustrations, however, are good and, on the whole, the book can be depended upon to form a useful addition to a metallurgist's library. (G. H. S.)

THE ORE DEPOSITS OF SOUTH AFRICA. Part II. By J. P. JOHNSON. (Crosby, Lockwood & Son.) 5s. net.

This little handbook of 50 pages, including diagrams and statistics, is designed to cover 'The Witwatersrand and Pilgrims Rest Goldfields and similar occurrences,' whilst Part I. of Mr. Johnson's volumes covered the 'Base Metals.' In our criticism of the previous volume, we expressed the view that the author had worked upon a scientific plane above the heads of those best served by such a compilation, the prospector and investor. In the present instance, the criticism must be reversed, for it cannot be justly said that there is similar need for a handbook on Transvaal gold occurrences, unless they are described in detail or the heights of originality are reached. Of the Transvaal base metals, the general reader knows little, of its gold there is the substratum of common knowledge which need only have been discussed briefly as a basis; or a thorough and detailed examination of the deeper problems. The Witwatersrand system is treated with greater detail in Hatch & Corstorphine's book on the geology of four colonies than in Mr. Johnson's specialising production. Perhaps one is disappointed with Part II. of 'The Ore Deposits of South Africa' because one has learnt to expect better things from Mr. Johnson, but disappointment will assuredly be felt by the large number of readers that the author's reputation is likely to win for the production. If, however, it is received merely as a handbook, it will serve a good purpose. Sixteen pages are devoted to the 'main auriferous conglomerates' of the Witwatersrand Goldfield, including several pages of sections. In his typical Central Rand stratigraphical section the author suggests the name of 'Parktown Beds' for the strata lying between the Orange Grove and Hospital Hill quartzites. Whatever the need for grouping, alterations in the already confused nomenclature of Transvaal formations and series must be left to the Geological Survey. It is not so much the suitability of a name that matters as the standardisation of its use, and uniformity can now only be enforced by such authority as that cited. The thickness of the quartzites between the Doornfontein beds and Main Reef is noted at 3,501 ft., apparently a typographical error. Mr. Johnson gives an interesting resumé of the various theories profounded on the origin of the gold in the Main Reef, and closes his impartial analysis by bravely hoisting the tattered ensign of the alluvialists.

The book also describes briefly the Klerksdorp and Pilgrims Rest goldfields, and numerous banket or quartz gold deposits in other districts. The geological sections comprise the most useful feature of the production. (R. S.)

Armstrong, E. Frankland. Carbohydrates: Glucose, the Disaccharides and the Glucosides. Roy. 8vo. *Longmans.* Net 3s. 6d.

Bruce, James and Harper, Harry. Practical Chemistry. 12 mo., pp. 248. *Macmillan.* 2s. 6d.

Catalogue of Modern Works on Science and Technology. New Ed. 8vo. sd. *Chapman & Hall.* Net 1s.

Ennis, William D. Linseed Oil and Other Seed Oils. An Industrial Manual. Roy. 8vo., pp. 330. *Constable.* Net 16s.

Harcourt, Leveson, Francis Vernon. Civil Engineering as Applied in Construction. 2nd Ed., revised by Henry Fidler. 8vo., pp. 642. *Longmans.* Net 16s.

Johnson, J. P. Geological and Archæological Notes on Orangia. 4to. *Longmans.* 10s.

Jones, H. C. Introduction to Physical Chemistry. Cr. 8vo. *Macmillan.* Net 7s.

Kempe, H. R. The Engineers Year Book of Formulæ, Rules, Tables, Data and Memoranda, 1910. Cr. 8vo., leather. *C. Lockwood.* 8s.

McConnell, Primrose. Farm Equipment Buildings, and Machinery. A Practical Handbook. Illust. Cr. 8vo., sd., pp. 108. *Cassell.* Net 1s.

Morsch, Emil. Concrete-Steel Construction. Translated from the 3rd (1908) German ed. Revised and enlarged by E. P. Goodrich. Roy. 8vo., pp. 378. *Constable.* Net 21s.

Murray, J. Allan. Soils and Manures. 8vo. pp. 370. *Constable.* Net 6s.

Palæontologia Universalis. (Issued under the auspices of the International Geological Congress by D. P. Oehlert and K. V. Zittel). Series 2. In 4 Parts. Plates. 4to., pp. 182. *W. Wesley.* Net 32s.

Sang. A. The Corrosion of Iron and Steel. Cr. 8vo. *Hill Pub. Co.* Net 4s. 6d.

Abstracts of Patent Applications.

(C.) 187/09. John Pendale (1), Harry Ormes Mascall (2). Process for disintegrating diamondiferous ground. 17.4.09.

This application relates to means for the disintegration of diamondiferous ground, in which an aqueous solution of carbon dioxide is used as the disintegrating agent.

(C.) 569/09. Hugo Philip Crush. An improved method of an apparatus for filtering liquids. 10 12.09.

This is a vacuum filter of the fixed submerged basket type in which the frames employed are self-discharging, as described in Transvaal Patent 578/ Dec., 06, and where the necessity for employing large pumps, intermittently running, for filling and emptying the filter tank is obviated by installing a system of storage vats connected by a large main to the cone-shaped bottom of the filter tank, and having small continuously running pumps for returning the pulp and wash-water from the lower to the upper storages.

It is also claimed that the greater rapidity obtained in the operations of filling and discharging prevents the formation of air-cracks on the slime cakes.

(C.) 577/09. Henry Livingstone Sulman (1), Henry Howard Greenway (2), Arthur Howard Higging (3). Improvements in or relating to the concentration of ores. 15.12.09.

The above application refers to a modification of the "flotation" method of concentration, which consists in the substitution of certain specified organic substances for oil or oily liquids previously used in conjunction with slightly acidified water, as mineral frothing agents. The substances mentioned comprise certain organic acids, ethereal salts, aromatic hydroxy-compounds and ketones.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 75/10. W. B. Crotton. Improvements in internal combustion engines. 1.3 10.

(P.) 78/10. John Smith (1), Charles de Vertus Duff (2). Improvements relating to jockeys for rope haulage. 4.3.10.

(C.) 79/10. Duncan McLean. An improved draught inducer for chimneys and ventilating shafts. 4.3.10.

(P.) 80/10. William McVitty. Improvements in nut locks. 4.3.10.

(P.) 81/10. Charles Christiansen. Improvements in valve devices applicable to tools, rock drills and the like. 5.3.10.

(P.) 82/10. Tom Cecil Leonard Howard. An improved means for igniting the fuses of explosive charges in blasting operations. 5.3.10.

(P.) 83/10. Alfred Adair. Improvements in precipitating apparatus. 5.3.10.

(P.) 84/10. William Cochran Boyd. Liners for tube mills. 7.3.10.

(P.) 85/10. Joseph Henry Dale. Improvements in feeding means for tube and similar mills. 7.3.10.

(C.) 86/10. Carl Bosch (1), Wilhelm Wild (2), Otto Schonherr (3), Johannes Brode (4). Hermann Wolf (5). Improvements in and relating to the manufacture of nitrates. 8.3.10.

(P.) 87/10. Sidney Joseph Crowther (1), Wilkinson and James Nicholson (2). Improvements in rock drills with detachable cutters. 8.3.10.

(C.) 89/10. Alfred Walton Stockett (1), John Hawthorne Wilson (2), John Whitton Marshall (3). Improvements relating to doors for mine cages. 8.3.10.

(P.) 90/10. John Lowther (1), Thomas Thompson Nichol (2), Benjamin Daniel Bushell (3). Improvements in pipe couplings. 9.3.10.

(P.) 91/10. Pretro Antonio Chiappero. Improvements in the treatment of ores and apparatus therefor. 10.3.10.

(P.) 92/10. Charles Edward Meyer. Improvements in and relating to the treatment of zinc-gold slimes and in apparatus for use in connection therewith. 10.3.10.

(C.) 93/10. Oscar Birgen Carlson. Improvements in the method of providing ammonium perchlorate. 11.3.10.

(P.) 94/10. Edward John Way (1), Alexander John Arbuckle (2). Improvements in vats or vessels for the treatment of ores. 11.3.10.

(C.) 95/10. Albert Hibbs Taylor. Improvements in rotation devices for fluid pressure operated hammer tools. 11.3.10.

(C.) 96/10. John Dewrance (1), Herbert Ernest Williams (2). Improvements in the manufacture of alkaline cyanides. 11.3.10.

(P.) 97/10. Alexander Gemmell. Acetylene hand lamps. 12.3.10.

(P.) 99/10. Wallace Puckey (1), Eber Nice (2), William Joseph Carter (3), Harold Westmoreland Carter (4). Improvements in ore feeders. 14.3.10.

(P.) 100/10. Paul Jerrard. Improvements in and relating to flying machines, airships and the like and in apparatus therefor partly applicable, also to other motor actuated machines. 15.3.10.

(P.) 101/10. James Arthur Hart. An improved process for the reduction of tin from cassiterite ores and concentrates thereof and apparatus therefor. 15.3.10.

(P.) 102/10. James Drage. Improvements in filtering machines. 15.3.10.

(P.) 103/10. Roelof Petrus van der Merwe. Automatic gate for railway level crossings, roads, etc., to be put in action by trains, carts, etc., respectively. 16.3.10.

(P.) 104/10. Lars Pedersen. Improvements in respiratory devices for use in mining and the like. 16.3.10.

(P.) 105/10. John Henry Anton. Improvements in method of attaching pulleys or similar devices to revolving shafts. 17.3.10.

(P.) 106/10 William McFarlane (1), Thomas Harrison (2). Improvements in rollers for haulage ropes and the like. 17.3.10.

(P.) 108/10. Hermann Stadler. Improvements in pulp settling apparatus. 17.3.10.

(P.) 109/10. Junius Ford Cook. Improvements in tube mill feeding devices. 17.3.10.

(P.) 110/10. Junius Ford Cook. Improvements in means for feeding pebbles to tube mills. 17.3.10.

(P.) 111/10. Andrew Francis Brink. Improvements in respiratory devices. 17.3.10.

(C.) 112/10. James Hodgkinson. Improvements in or relating to mechanical stokers for boilers and other furnaces. 18.3.10.

(C.) 113/10. Louis Brennan. Improvements relating to methods of and means for imparting stability to, and maintaining stability of bodies such as single track vehicles or vessels. 18.3.10.

(P.) 114/10. Alexander Sonter (1), Thomas Pearson Smith (2). Improvements in shaft bearings. 18.3.10.

(P.) 115/10. William Charles Chergivin. Improvements in apparatus for use on sands and other dumps. 18.3.10.

(C.) 116/10. Carl Giesecke. Process and apparatus for dressing ores. 18.3.10.

(C.) 117/10. Hans Charles Behr. Improvements in stamp mill cams. 18.3.10.

(C.) 118/10. Paul Destefani. Improvements in devices for burning paraffine, wax, or the like, for illuminating or other purposes. 21.3.10.

(P.) 119/10. Henry Alford Walker. Improvements in drills or bits applicable for rock drilling and like operations. 21.3.10.

(P.) 120/10. James Inglis. Improvements in automatic tipping apparatus for mine trucks and the like. 21.3.10.

(P.) 121/10. James William Straughan. Mechanism for driving reciprocating pumps or the like, by means of a wind or other motor. 21.3.10.

(P.) 122/10. William Arthur Caldecott. Improvements in means for transporting and de-watering pulps. 23.3.10.

(P.) 123/10. Andrew Danks. Improvements appertaining to amalgamating plates or surfaces. 23.3.10.

(P.) 124/10. Alfred Adair. Improvements in lixiviating apparatus. 23.3.10.

(P.) 125/10. Samuel Pram Brook. Improvements in linings for tube mills. 23.3.10.

(C.) 127/10. Edwin Burt. Process of filtration. 24.3.10.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Ball, H. S., *l/o* Johannesburg; c/o Standard Bank of S.A., Ltd., Clements Lane, London, E.C.

Bosanquet, F. G., *l/o* Hereford; Van Ryn Deep, Ltd., Benoni.

Brunton, W. G., *l/o* Premier Mine; 101, Staib St., Doornfontein, Johannesburg.

Cindel, F. W., *to* D Section, Crown Mines, Ltd., P. O. Box 156, Fordsburg.

Gerlings, H., *to* 46, Hoofd St., Braamfontein, Johannesburg.

Hamilton, E. M., *l/o* San Francisco; c/o Dolores Mines Co., 112, Broadway, New York, U.S.A.

Heeley, A. W., *l/o* Essexvale; Mozambique Mines Office, P. O. Box 2, Macequece, Portuguese East Africa.

Hesom, W. W., *l/o* Eureka; 44, Gordon Road, Bertrams, Johannesburg.

Hill, J. Whitelaw, *l/o* Scotland; P. O. Box 142, Salisbury, Rhodesia.

Holden, W. C., *l/o* Boksburg; North Randfontein G. M. Co., Ltd., P. O. Box 26, Randfontein.

McEwen, M. T., *l/o* Selukwe; Asp Mine, P. O. Kimberley Reefs, Rhodesia.

Medlin, E. J. G., *l/o* Germiston; P. O. Box 275, Akron, U.S.A.

Morrison, S., *to* P. O. Box 102, Fordsburg.

Nicholson, A. S., *l/o* Germiston; Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg.

Roake, H. G., *l/o* Germiston; 108, Main Reef Road, Denver.

Roberts, S., *l/o* Randfontein; Krugersdorp Club, Krugersdorp.

Strout, E. A., *to* c/o Compañia de Santa Gertrudis, S.A., Pachuca, Hidalgo, Mexico.

Thomas, G. G., St.Inst.M.M., *l/o* Johannesburg; P. O. Box 43, Sabie.

Tonnesen, O. S., *l/o* London; East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. X. APRIL, 1910. No. 10.

Proceedings
AT
Ordinary General Meeting, April 16, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, April 16th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

65 Members: Dr. J. Moir, Messrs. W. R. Dowling, C. B. Saner, F. F. Alexander, E. H. Croghan, A. Richardson, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, A. Whitby, H. A. White, Prof. J. A. Wilkinson, A. F. Crosse, W. A. Caldecott, R. G. Bevington, H. A. Adams, R. Allen, S. Beaton, F. W. Bentley, B. V. Blundun, F. W. Cindel, A. A. Coaton, M. H. Coombe, W. M. Coulter, W. H. Cusworth, G. G. Ferris, J. Gaze, J. Gray, W. L. Hamilton, Dr. J. McC. Henderson, A. E. Irwin, J. H. Johnson, J. A. Jones, G. A. Lawson, H. Lea, J. Lea, R. MacGregor, C. E. Meyer, J. T. Mitchell, H. Morrell, P. T. Morrisby, M. T. Murray, S. Newton, H. G. Nichols, E. A. Osterloh, E. Pam, F. D. Phillips, J. F. Pyles, E. T. Rand, L. J. Robinson, W. H. Roe, H. Ross, O. D. Ross, A. Salkinson, C. B. Simpson, S. H. Steels, R. Stokes, A. Thomas, J. M. Thorburn, O. S. Tonnesen, C. Toombs, J. F. Walker, F. Wells and L. J. Wilmoth.

23 Associates and Students: Messrs. C. F. Bayly, C. Y. Bruce, G. J. V. Clarence, C. L. Dewar, C. E. Hawkins, W. J. R. Hunter, A. King, C. J. McCaffrey, G. Musson, N. N. Newland, J. G. Peebles, J. Pope, H. B. Powter, H. G. Roake, M. Schaap, T. F. Smith, H. Stadler, D. J. Thomas, J. Thorlund, W. Waters, F. Wartenweiler, W. Webb and L. A. Womble.

25 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the March *Journal*, were confirmed.

NEW MEMBERS.

Messrs. J. E. Thomas and R. Allen were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

COOK, JAMES, Battlefields Mine, Rhodesia. Mine Manager.

DOVETON, DOUGLAS HODSON, Groenfontein Tin Mines, Potgietersrust. Mining Engineer.

OLSEN, ARNOLD CARL LOUIS, Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Resident Mechanical Engineer. (*Transfer from Associate Roll.*)

PATERSON, WILLIAM JAMES, East Rand Proprietary Mines, Ltd., P. O. Box 56, East Rand. Engineer.

WEBB, CECIL FRANCIS, Geldenhuis Deep, Ltd., P. O. Box 1, Cleveland. Cyanider.

The Secretary announced that since the last meeting of the Society the Council had admitted as a student member:

CALDERWOOD, DOUGLAS YOUNG, Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Mining Apprentice.

GENERAL BUSINESS.

The President drew attention to the arrangements made for the Annual Dinner of the Society to be held on the 14th May, saying that it was hoped to make the function a success and more sociable than in the past, by the introduction of round tables, at which six or eight persons could sit down together.

PRESENTATION TO LORD SELBORNE.

I also want to draw your attention to the fact that last evening I had the honour of presenting to His Excellency, Lord Selborne, in your name and in the name of the other professional and technical societies of the Rand, a cabinet made of Rhodesian mahogany. Enclosed within this were samples of the divers processes which the reef has to undergo in its treatment here until there is obtained the gold, a little bar of which was in the cabinet. Along with this was an album containing about 140 views of the Transvaal. They were a fine collection, and His Excellency was greatly pleased with the presentation.

SEVENTEENTH ANNIVERSARY.

Another matter to which I wish to call attention is that we are now celebrating our 17th birthday, and I need not say how much the Society has gone ahead since it was formed. I have here the minutes of the first meeting held. These, you will be pleased to hear, are in the handwriting of our Vice-President, Mr. Dowling. From the records of this meeting it will be seen that Mr. A. F. Crosse was in the chair, and many other well-known names appear. They were :–

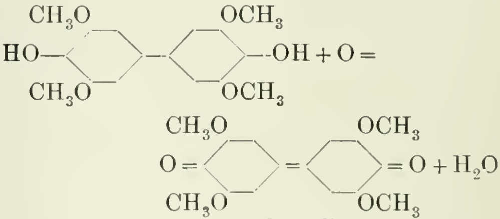
Messrs. W. Bettel, T. W. Wood, J. Littlejohn, A. Adair, S. H. Pearce, R. E. Hall, W. R. Feldtmann, F. B. Stephens, A. F. Stewart, T. W. Thomson, H. F. Julian, S. B. Asher and W. R. Dowling.

These were our pioneers. It is also interesting to read here the suggestions which were brought forward and adopted as the basis of our Society. The following suggestions were approved of :— "That members of the Society meet once a month, and the day of the month be the third Saturday of each month." For 16 years we have met on this third Saturday of each month. Secondly, "That the amount of the subscription to the Society be left in the hands of the Committee, with a recommendation that it be made as low as possible." That also we have tried to keep up. Third, "That membership be open to members of the assaying profession and of the chemical industry, and that membership be subject to ballot and be unlimited as to numbers." Gentlemen, we have followed this out as best we could. Starting with a dozen men who attended the first meeting, we have a total membership of just on 1,200 members. I think we may congratulate ourselves that we have done so well, and also on the fact that as regards the matter published in our *Journal*, equally excellent progress has been made.

TEST FOR PRUSSIC ACID.

Dr. J. Moir (*Vice-President*) : I have a little demonstration to make, of a new test for prussic acid which I have invented. The principle of the test is the oxidising action produced when a cupric salt is added to prussic acid (owing to its reduction to cuprous cyanide). An organic compound which oxidises to a coloured substance is added, and the most sensitive one I have found is hydrocoerulignone. The test is sensitive to something like one part in 5,000,000 of water, but unfortunately the reagent is rather unstable : however, other oxidisable substances of the same class will answer although not so sensitive, and some of these, like benzidine, keep much better. The reactions in the case of the very sensitive reagent (copper acetate, acetic acid and hydrocoerulignone) are :—

(1) $2Cu\bar{A}_2 + 2HCN + H_2O = 2CuCN + 4H\bar{A} + O.$



Coerulignone

In very dilute solutions an instantaneous orange colour is obtained, and in stronger solution a red precipitate of coerulignone which possesses a violet lustre. If benzidine be used, the colour obtained is indigo blue.

Mr. H. A. White (*Member of Council*) : I should like the Doctor to explain if this substance would not directly show the presence of H_2O_2.

Dr. J. Moir : Yes, it is an indirect test for HCN, and the reaction is also given by oxidising agents such as $FeCl_3$ or H_2O_2. It can be applied on a test paper in such a case, as oxidising agents are not volatile as a rule.

Mr. A. F. Crosse (*Past-President*) : Supposing there was a small amount of gold in the water, would it still cause a reaction ?

Dr. J. Moir : I think it would, because in that case there would always be an excess of cyanide, over and above that combined with the gold.

Mr. A. F. Crosse : No, there should not be. In return water there might be a grain or two of gold in solution. Outside that gold—there is perhaps only one or two grains per ton—there is probably no cyanide present. Still we could prove the presence of cyanide. Would this test show it ?

Dr. J. Moir : I think it would, but I should prefer to try the experiment.

THE TREATMENT OF ACCUMULATED SLIME, AND THE USE OF FILTER PRESSES FOR CLARIFYING SLIME SOLUTION AND BY-PRODUCTS.

By John D. O'Hara (*Member*).

As the treatment of current and accumulated slime by filter presses is now and again being brought forward, the writer thinks it may be of interest to members to give his experiences in treating accumulated slime by filter presses, compared with the decantation and the Adair-Usher processes.

The Meyer & Charlton and George Goch Companies also had filter presses, but an improved type, the Dehne press, and in those days 1904 to 1906, it was freely stated that the decantation process had seen its last days. I understand that one of the companies mentioned is now using the decantation or the Adair-Usher process.

On taking charge of these works (Nigel G. M. Co.) in October, 1904, I found that a filter press plant had just been erected, consisting of

Three (3) 4 ton Johnson presses, with the necessary plant.
One Monteju,
Three small circular iron solution vats.
Two air receivers, and
Two agitating vats.

The accumulated slime, averaging 11 dwt. and over, was treated as follows :—

The slime was hauled from the slime storage dam in trucks to a platform over the two agitating vats half full of water, with sufficient lime (caustic soda was first used, but found too expensive) to neutralise the acid in the slime, which was very acid. The slime was dumped into the agitating vats, thoroughly mixed, and the pulp thereafter run from these by gravitation to the Monteju, whence it was sent to the presses, one of which was filled at a time, by compressed air. Then the required cyanide solution washes from the circular solution vats were applied with a final water wash, after which air was blown through the press to dry the slime before discharging into trucks under the presses ; from there the slime cake was taken by mechanical haulage to the sand dump. All the solutions were passed through the presses by compressed air, and the moisture of the filter pressed slime averaged 25% to 30%.

The treatment was successful but the working cost was too high, and in treating 8,000 tons this was estimated at 12s. per ton. The residues were irregular owing to the high % of sand in the accumulated slime. As low as ·4 dwt. residues were obtained.

Average charge ...	11·340 dwt.	
Average residue ...	1·322 ,,	
Average extraction ...	10·018 ,,	=88%

It was naturally thought that the solution would be clear enough to gravitate to the extractor boxes, but we soon found that we had to erect filter vats, the same as are used for the decantation process. Filter cloths were an expensive item ; 50 in. cloths were used, specially imported, but they only lasted a few charges, whilst the small tonnage treated gave no opportunity to reduce the cost per ton.

The causes of the failure of the presses can be summed up as follows :—

1. Cost of cloths mentioned above.
2. Heavy consumption of cyanide and lime.
3. Cost of handling the slime from the dam to the dump.
4. Not being able to do more than 12 to 14 presses in 24 hours = from 48 to 56 tons.

Some members who have had experience with filter presses, may claim that more tonnage could have been treated, so I might mention now that the Nigel was visited by a good many practical and experienced filter press men, including Dr. Diehl and Johnson's representative, who after carefully examining the method of working, were convinced that a larger tonnage could not be done in the 24 hours.

In August, 1905, we decided to try the experiment of pumping the accumulated slime to the tailing wheel, with the idea of separating the sand and slime in the ordinary way. This proved successful, and the filter press plant was closed down in October, 1905. The slime was agitated and circulated, using a 5″ Morris pump with an air jet in the delivery of the pipe line to the tailing wheel.

We then tried to make use of the filter presses by pumping the slime from the decantation plant to the agitators, and from there into the presses, with the idea of recovering the gold in solution and possibly reducing the percentage of undissolved gold. Even this did not pay the cost of handling the slimes, though the gold in solution was recovered. The amount of undissolved gold remained unaltered. It was cheaper to discharge to the slime dam and pump back the solution, *i.e.*, by using a pump placed at the bottom end of the dam. Then we tried using the presses to clarify the slime decanted solution. This proved successful and we have used them for that purpose since that time.

The advantages gained are :—

(*a*) An ideal clear solution for precipitation of gold in the extractor boxes.
(*b*) No trouble by having to clean out filter tanks.
(*c*) Cleaner gold slime for smelting.

It was a relief after 12 months' hard work to make the filter presses pay and to eventually find them of some use.

The slime charges from November, 1904, to October, 1905, before we determined to pump the accumulated slimes to the wheel (except experimenting from August, 1905) were 5·038 dwt.

Residues	·720 dwt.
Extraction	4·318 dwt.

= 85·70%

From November, 1905, to June, 1909 :

Charges	4·156 dwt.
Residues	·688 dwt.
Extraction	3·468 dwt.

= 83·44%

The cost of handling accumulated slime to the wheel averaged 7s. to 8s. per ton, chiefly owing to there not being sufficient capacity in our slime plant. This has now been enlarged, with a reduction in cost to an average of about 5s. per ton.

After the stored accumulated slime had been finished, the General Manager decided to clean out the old water dam. Owing to the sand, gravel, ashes, weeds, and goodness knows what else, the sludge pump was continually blocking up. This difficulty was overcome by erecting a Chilian mill or puddler over the agitating tank. A cone was fixed in the tank, and the agitator done away with. Everything passed through the puddler, and since then the pump has given no trouble. The weeds were another source of trouble, so a launder was fixed over the tanks with small screens, and two boys employed to keep the screens clear. The same was done at the tailings wheel. The decayed vegetable matter forms an excellent manure, and this we use on the farm. Naturally, the bottom of the dam was the richest, and it paid us to take at least six inches of the black mud. The effluvia from stagnant water was not to be compared with the stench from the puddler while working the bottom of the dam.

Since July, 1909, all the current and accumulated slimes have been treated by the Adair-Usher process, with the following results to date:

Average Charge	3·812 dwt.
Average Residue	·473 dwt.
Extraction	3·339 dwt.

= 87·59%

which gives an average of 1·89% and 4·15% better extraction in favour of the Adair-Usher process.

The past three months, January, February, and March, are better still.

Average charge	3·635 dwt.
Average residue	0·351 dwt
Extraction	3·284 dwt.

= 90·37% = 5·80%

in favour of Adair-Usher, averaging 85·70% and 83·44%.

The residue samples taken are:—

(*a*) One washed sample for the undissolved gold.

(*b*) A dip sample before discharging.

(*c*) A dip sample just before washing out the cone.

(*d*) One taken every 10 minutes while discharging, at the delivery of the pipe into the slime dam.

The Adair-Usher process is, in our opinion, nothing more or less, than a good quick washing process. It is not effective for dissolving the gold, but only displaces the rich gold solution, so that the gold must be dissolved before transferring to the Adair-Usher vat.

Circulating in one vat for dissolving gold in either of the two processes, *i.e.*, decantation or Adair-Usher, is waste of time, for the cone slime, which naturally contains the most gold, is not got at. A transfer from one vat to another is worth all the time wasted on circulating, since the cone slime all passes through the pump. This is especially noticed when treating rich slime, and should be the same for poor slime. We have also noticed that the slime in the Adair-Usher vat after being shut off for settlement for discharge to dam, settles quicker than settlement by the decantation process. This is perhaps accounted for by the slime gradually settling while the solution is being slowly displaced. We can thus always rely on a complete hard settlement within 24 to 28 hours, after washing.

The percentage of accumulated slime treated daily as to current slime is 50 ton = 30%. The average value of the accumulated slime treated since November, 1904, is 10 dwt. It now averages about 6 to 7 dwt. The solution decanted per ton of dried slime = 5 to 6 ton.

For the past two years we have used what waste steam is available for heating the slime solution, and especially during the winter months it is a decided advantage for quick settlement, for in treating current and accumulated slime, settlement is not as good as when treating current slime alone, which is perhaps accounted for by the fact that oxidised slime is very light.

The writer used warm solutions in 1896 at the Rip G. M. C., Luipaards Vlei, when treating free milling ore by direct treatment.

By Products.—Our filter press plant being useless, apart from the presses, we decided to use the Monteju for black sand treatment. One of the agitators was placed in the Monteju, and iron scrapers fixed to the arms about half an inch from the bottom. About half a ton of black sand was agitated with cyanide solution, and decanted off by gravitation. The average extraction, by solution assays and ordinary residue sample amounted to 92% to 94%. One of the circular solution vats was lined out with old tube mill liners, and used for grinding purposes, imported pebbles being the grinding medium. The charges consisted of slag, old pots, liners, and ash from the smelting room with silver added. Good results were obtained, the residue averaging from 1 to 3 oz.

The batea was made by using the cover of the Monteju, and is driven from the counter shaft of the accumulated slimes plant. The rest of the plant the engineer has made use of, air receivers, etc., so the filter press plant is gradually paying for itself. It is much more satisfactory, especially on an outside mine, to treat by-products and recover the gold every month, than to bag and store them till a purchaser materialises.

My thanks are due to our General Manager, Mr. Curnow, for his permission to publish these figures, and to the Resident Engineer, Mr. W. H. Bodilly, for his valuable advice and assistance, without which we could not have succeeded in converting the plant for successful work.

The President: In proposing a vote of thanks to Mr. O'Hara for his interesting paper on his experiences at the Nigel, I feel that some excellent discussion should ensue, and that this should be the means of eliciting information as to work done not only in the outside districts but on the Rand. The treatment of accumulated slime has not been well discussed in this Society in the past, and Mr. O'Hara's results should certainly stimulate interest therein. The idea of sending the accumulated slimes to the wheel after washing and breaking up seems a good one, and the results obtained certainly guarantee its continuation. I would like to emphasise one point in this paper before it is published and gets to foreign readers and that is that the residues quoted, are not representative of those in present day Rand practice. Mr. O'Hara had a difficult proposition to handle. He was dealing with accumulated slimes with a considerable percentage of sand, and a large amount of organic matter, the deleterious effects of which we know, and his results therefore represent work done under quite different conditions than obtain on the Rand. In addition, Nigel reef results would not be comparable with those obtained on the Rand, since the ore there is not so amenable to treatment. I bring forward this matter to prevent misunderstanding for as some of you may have noticed in the February *Journal*,* there is an abstract wherein the writer deduces from some remarks in a report issued by one of our members that Rand slime practice must be at a low ebb. He fails to realise that what may be justifiably applicable to one small property is not in force on all the mines here. He, from want of knowledge, not to put too fine an edge on it, spreads himself out and proclaims at large the backward state of Rand metallurgy. From such procedure I wish to protect Mr. O'Hara and other members—hence these remarks.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

PART II.—FURNACE EXPERIMENTS.

By Prof. G. H. STANLEY (Member of Council).

The first small blast furnace devised for these tests consisted of a steel sulphuric acid drum, of 18 in. diameter, lined with 4½ in. firebrick, so that the internal diameter was 9 in., and height 42 in. over all.

Three twyers were provided, of ¾ in. steam pipe, entering the furnace about 6 in. above the hearth bottom.

The ores employed were the Onderstepoort titaniferous, as in the previous tests, and silicious ore from the Timeball Hill Range, Pretoria.

The former apparently occurs in three parallel reefs, and the latter in two, the results of their analysis being shown below, together with that of the dolomite from Irene, used as flux.

	Onderstepoort.			Timeball.		Dolomite.
	South Reef.	Middle Reef.	North Reef.	Lower Reef.	Upper Reef.	
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Silica SiO_2	1·36	0·76	1·57	14·30	17·93	1·50
Titanium Dioxide TiO_2	14·10	19·52	18·70	undet.	undet.	
Ferric Oxide Fe_2O_3	79·11	76·23	76·41	76·93	73·81	1·70
Alumina Al_2O_3	7·79	6·82	4·31	7·20	7·31	
Lime CaO	undet.	undet.	undet.	undet.	undet.	26·50
Magnesia MgO	undet.	undet.	undet.	undet.	undet.	20·50
Sulphur - -	trace	trace	trace	·03	·03	
Phosphorus - -	trace	trace	trace	·14	·17	

In the first attempt, after getting the furnace hot and melting down scrap in it to carry heat to the hearth, a charge was weighed out of ore, broken to half inch cubes, and mixed with dolomite in the proportions calculated to yield a slag of the composition indicated by the crucible experiments, the coke used being in pieces of 2 or 3 in. diameter. This melted down rapidly and ran very liquid from the furnace, very little iron being reduced.

In two or three further tests the ore was reduced to pass a 10 mesh screen, and coke broken to about 1 in. cubes. As a result, iron

---

*See this *Journal*, vol. x., Feb., 1910, p. 296.

was reduced, but always in the form of spongy blooms of soft iron of not more than 3 or 4 lb weight; this accumulated in the hearth, ultimately stopping the tap hole. The slags, however, were very fluid, and also very corrosive, as the furnace lining was badly cut; the analysis of a typical specimen is as follows:—

SiO_2	... ...	24·5%
TiO_2	... ...	10·0%
Al_2O_3	... ...	30·7%
FeO	.. ...	20·1%
CaO	... ...	8·5%
MgO, etc. not determined	...	6·2%

At about this stage, through the good offices of Mr. Harbord, the authorities of the Central South African Railways arranged for one of the foundry cupolas in the Pretoria workshops to be placed at my disposal for a month, to make experiments on a larger scale, and as a result of the representations of Mr. Kotzé, the Government Mines Department undertook to defray the cost of the materials, etc., used. Further work with this small furnace was therefore abandoned, and indeed seemed unlikely to yield successful results, on account of its small size, and the next experiments were conducted at Pretoria.

The furnace used was one of Thwaite's "Rapid" cupolas, provided with drop-bottom, fore-hearth, and two rows of twyers; the inside diameter at the charge-floor being 32 in., and in the hearth 22 in. The height from charge-floor to top twyers was 12 ft.

For the purpose of the experiments the fore-hearth was stopped off, the bottom doors removed, and a brick-lined steel plate receiver hung from the bottom to form a hearth. Unfortunately, the construction of the furnace itself only allowed of a 3 in. lining in this hearth, and although it was packed round with sand it cooled too rapidly and was discarded after the first two runs. For subsequent runs the drop doors were placed in position and a hearth bottom formed on them of sand and clay, a tap hole being improvised through the fettling door at the side. The slag tap hole was formed by making a hole through the iron casing of the furnace and bolting on a runner, just below the air belt, which latter, of course, made it impossible to place this hole any higher up the furnace, and therefore prevented a hearth lining of more than about 3 in. being used; otherwise there would have been no room for iron to collect.

As already stated, there were two rows of twyers —three in the upper and two in the lower, all of 4 in. diameter at the commencement; these were subsequently altered as will be detailed later on. Fresh quantities of ore and dolomite were obtained from the same places as before: their analyses are shown in the appendix, and are somewhat different from those previously made. These however did not become available till some days after the experiments started.

Details of Experimental Runs.

Run A, February 7.—Samples of the materials were taken, but pending their analysis, it was assumed in calculating the charge that the titaniferous ore was of a composition represented by the average of the previous analysis of the three reefs, since the ore was a mixture of all, and similarly the average of the two analyses already given was taken as the composition of the silicious ore.

The coke used was Westphalian, at first in lumps as received, though afterwards broken, and contained 9·5% ash of the following composition:

SiO_2	... ...	41·3%
Fe_2O_3	... ...	10·0%
Al_2O_3	.. ...	38·0%
CaO	... ...	3·7%

It also carried:

Sulphur ... ... ·86%

The large amount of alumina present complicated the calculation considerably and made it impossible to make the slags of the desired proportions of silica and titanium, and though a monosilicate was calculated, the silica had greatly to exceed oxide of titanium.

After the furnace had been alight for about two hours and was about half-full of coke, four more bags of coke were charged in (about 400 lb.) and on top of this, about 12.20 p.m., the first charge, consisting of 112 lb. silicious ore, 56 lb. titaniferous ore, and 28 lb. dolomite, all broken to about 1½ in. cubes, but without separating fines. On the top of this 200 lb. of coke and another charge of ore and flux, as before, was sent in at 12.40, and a third at 12.50. At 1.10, the slag was observed to be up to the lower twyers, and the furnace was accordingly tapped, but ran slag only. At 1.35, another 200 lb. coke was added and then the same amounts of ore and flux as before, and a bag of coke on top—this to prepare for the next charge, which was added at 1.50, and contained the other bag of coke, mixed in to increase the reducing effect, and as before a bag of coke on top. At 2, another bag of coke was added, and another charge containing coke mixed in. However, nothing but a very fluid slag was produced, so that, hoping to replace some FeO in the slag and facilitate reduction of iron, the amount of dolomite in the charge was doubled from this point onwards.

The effect of this soon became apparent in increasing stickiness of the slag, so that charging of ore was stopped, a bag or two of coke added to keep the furnace hot, and it was then allowed to blow down, blast being cut off about 7 p.m.

Next day the hearth was found to be full of a breccia of coke cemented by a greenish grey slag, containing also about 60 lb. of pieces of soft iron. This grey slag was composed of, approximately:

SiO_2 53%, Al_2O_3 36%, CaO 3%,

and was doubtless formed of coke-ash with a little of the ganister lining of the furnace, which was considerably cut away. It was cleaned out and made ready for the next run.

Three samples of slags were analysed besides the one just mentioned, all being typically ferrous silicates in appearance and containing from 38·3 to 39·5% SiO_2, 1·2 to 4·3% TiO_2 and 37·2 to 50·4% FeO.

Run B, February 9.—The furnace being well alight, 12 bags of coke were added before charging any ore. In order to produce more reducing conditions, the proportion of coke was increased and the blast pressure cut down to 3 ozs., so as to diminish the oxidising effect and also the output, which obviously depends on the amount of coke burned, thereby keeping the ore longer in the furnace. The materials were more finely broken than before, the ore to about $\frac{3}{4}$ in. cubes, and coke to $2\frac{1}{2}$ in., and the proportions altered so that equal parts of the two classes of ore were used, the extra silica required being furnished by sharp sand. In charging, then, one basket of coke (75 lb.) was thrown in, and then a mixture of 56 lb. titaniferous ore, 56 lb. silicious, 14 lb. dolomite, $6\frac{1}{2}$ lb. sand, and 75 lb. coke. Charging commenced at 11.50 a.m., and by 12.30 five charges had been given and the furnace was full. The first slag ran at 1.15 p.m. through the bottom tap hole, which was left open for a while and then closed. Four other charges were made, and at 3.15 the tap hole was opened, but only slag escaped—no iron. From that time each charge had an extra basket of coke and 2 lb. more dolomite. The tap hole was opened from time to time, with increasing difficulty, and it was obvious that the hearth was becoming gradually cooler, so that the slags did not run so freely and no iron ran at all. Consequently, after only three of the new charges had been given, as the casing showed red hot in two places, charging was stopped and the furnace allowed to blow down, the blast being increased to try and raise the temperature. It became impossible, however, to open the lower hole at all, and the last of the slag escaped by the slag hole, and on taking down the settling door the hearth was found to be completely filled with slag and spongy iron weighing about 175 lb.—a specimen of which is exhibited. It was really therefore the fault of the iron, and not of the slag, that the furnace choked up.

Owing to this solidified core the improvised hearth was only removed with great difficulty and was dispensed with in the following tests.

A sample of the slag from this run, taken over the various pots, contained 38·6% SiO_2, 8·1% TiO_2 and 38% FeO: that in the hearth with the iron contained 40·5% SiO_2, 8·57 TiO_2, and 32·6 FeO. A considerable quantity of the furnace lining obviously entered the slag, the walls again being badly cut.

Run C., February 14.—To still further diminish the air supply, but at the same time allow of obtaining a higher smelting temperature at the twyers by using a higher pressure, the bottom twyers were closed off and not used again in any further tests, leaving only the top three in use.

Further, the analyses of the materials having come to hand, which showed the silicious ore to contain more silica and alumina than had been assumed, the proportion of Timeball ore was decreased to half the titaniferous, to make the slag less aluminous, but the coke remained as before.

The blast commenced at 3 ozs. and was slowly increased to 5. After getting the furnace alight and throwing in 12 bags of coke as before, charging commenced at 11 a.m., each charge consisting of one basket of coke followed by a mixture of 112 titaniferous ore, 56 silicious, 10 sand, 35 dolomite, and 2 baskets of coke. Twelve of these charges were given, but very little iron was produced, a little did, in fact, run from time to time, and it was felt to be accumulating in the hearth; another basket of coke was therefore added to each charge for the next three charges, but at this time the furnace walls were observed to be burning through, the iron casing being red-hot in several places, charging was therefore discontinued and the bottom dropped at 9 p.m.

From this run about 300 lb. of iron was obtained, of which about 200 lb. was in the form of soft iron from the hearth, and 100 lb. cast iron from slag pots and runners.

Two samples of slag taken showed SiO_2 35·9% and 37·8%, TiO_2 9·7 and 18·0% and FeO 25 and 28%, which showed a still further improvement in the conditions, and the slags ran perfectly fluid, though they were higher in titanium.

Run D., February 17.—For this run the furnace was completely relined, and the diameters of the three twyers reduced to 3 in. It was also determined to experiment with the silicious ore only, and so be free from any influence of titaniferous ore, till correct conditions were determined.

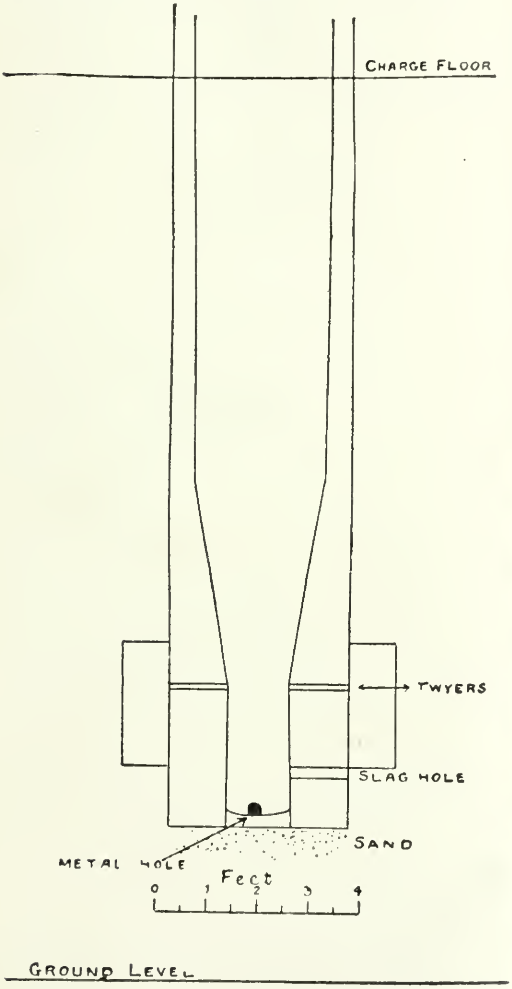
The furnace was started off as before with 12 bags of coke, blast kept about 3 or 4 oz., and the charges consisted of 112 lb. ore, 56 lb. dolomite, and 3 baskets of coke (225 lb.), the highest proportion yet used. Charging commenced at 2.30 p.m. and was maintained at the rate of two charges per hour till eight charges had been fed in. The furnace was running nice and hot, and the slags very liquid, but very little cast iron was produced, while soft iron accumulated in the hearth, so that no more charges were given and the furnace blew down and was dropped at 11 p.m.

The hearth contained a bear of soft iron which dropped clean out, and a fair sample of the slag contained 20·8% FeO, but was not further examined. The lower iron content of the slag was encouraging, though otherwise there was no greater success than with the titaniferous mixture, so for the next trial the twyers were made still smaller.

Run E., February 19.—The charge employed was the same as in D, but the twyers were reduced to 2 in. in diameter. The furnace having been lighted at 6 a.m., and partly filled with coke as before, at 9 a.m. the first charge was given, and, after 4 charges, at 9.50 the furnace was full. Blast was kept between 5 and 6 oz., and charging continued as the stock went down. However, on tapping it was evident that another bear was forming in the hearth, and only slag ran, and therefore at 2.30 the size of twyers was still further reduced, by slipping in pieces of 1½ in. pipe, and the blast increased to 8 oz. Still only slag ran, and the blast was consequently cut down successively to 6 and 5 oz., resulting in the production of some cast iron. This at first solidified in the runner, but on tapping at 5.45 we had the satisfaction of running about 30 lb. of cast iron into the slag pot. At 6.45 two small pigs were cast; at 7.30 again both iron and slag ran very satisfactorily, and also again at 8.45, and if our coke supply had held out there seemed to be every prospect of the run continuing till the lining burnt out. Coke, however, had run short, and the last charge was given at 6.30, after which the furnace had to blow down. In all 17 charges were thrown in; from this 424 lb. of iron were recovered, 70 lb. of bear in the hearth and 354 lb. of white cast iron. Slag from the second pot of the run (2 in. twyers) contained 13·0% FeO; the 3.30 p.m. slag contained 14·1%; 5 p.m. 12·5%, and at 10 p.m. 13·3%; this last also containing 38·5% SiO_2. The slags still contained too much iron to be satisfactory, but it was feared that if a further diminution of size of twyers or of blast pressure were to be attempted, the furnace would not keep hot enough, and even with the diminished output and rate of working secured with the small twyers, the ore only remained about 4½ hours in the furnace, whereas in an ordinary iron smelting furnace it seldom occupies less than 20 to 24 hours in its descent, and has therefore ample time to be reduced, to take up carbon, and in the smelting zone, silicon, etc., and so become fusible. The presence of these impurities also tends to prevent the reduced iron from being reoxidised as it comes in front of the twyers, while in the small furnace such protection is diminished as the iron is purer, and I am of opinion that some at least of the iron oxide in the slag was formed by reoxidation in this manner. It seemed therefore hopeless to expect a slag free from FeO in such a small furnace, on account of the comparatively large proportion of the hearth volume occupied by these small reoxidising areas, if I may so term them, in front of the twyers in the small furnace, as compared with a large one; hearth volumes of course increasing with the squares of the diameters. Further, this iron oxide in the slag would naturally tend to prevent the production of any but white cast iron.

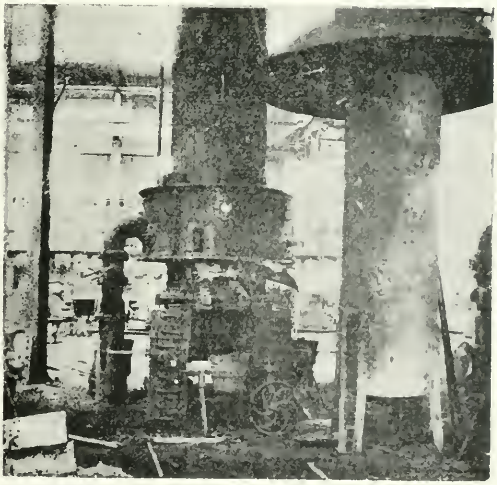
Run F, February 23rd.— More coke having arrived, this was commenced exactly as run E, the furnace being lighted at 10.30 a.m and charging of ore commenced at 2.30, the composition of charges also being the same as in D and E. The blast at the commencement was 5 oz,. which was cut down to 4, as only slag ran. The first iron ran at 9 p.m., and from that time titaniferous ore was added to the charge in gradually increasing proportions till in the fourth charge afterwards, there were 56 lb. titaniferous, 56 lb. silicious, 28 lb. dolomite, and 225 lb. coke, which was maintained nearly to the end, the last three charges having 7 lb. extra dolomite and 5 more sand. Whether it was due to the titaniferous ore or to the diminished pressure, the furnace did not seem to work well, and was very sticky at 4.15 a.m., chiefly through a large admixture of soft iron in the slag. At 7 a.m. the top hole was only opened with great difficulty, and again at 8; and at 9 it was impossible to get through either hole, so at 10 the bottom was dropped, blast having been increased towards the end to increase the temperature. The hearth was found to be completely filled with soft iron; a bar, however, was driven up through it and tapped off some cast iron and slag, in fact the slag ran quite well whenever the top hole was opened, and the closing of the holes was due simply to the accumulation of soft iron in and around them, so that the stoppage was not due to the want of fluidity in the slag, through the presence of titanium. It did seem, however, that the presence of titanium hindered the formation of cast iron, though on this point the evidence was not conclusive, as the furnace had not worked well from the start. The iron weighed about 500 lbs., and a sample of slag taken during the later part of the run contained 42·6% SiO_2, 8·3% TiO_2 and 7·4% FeO, while that tapped from the bear at the last contained 38·6 SiO_2, 8·8 TiO_2 and 12·0 FeO (no doubt due to the increased blast). This low content of FeO still seemed to indicate that improvement might be effected by still further diminishing the rate of output, and as the furnace was again in need of relining, this opportunity was taken of diminishing the size of hearth as well as of the twyers, in order to enable a good temperature to be kept up in the hearth.

Run G., March 3rd.—As relined for this run, the twyers were 1 in. in diameter, and the hearth 14 in. at the twyers, as shown in the diagram.



The hearth had unavoidably to be thirty inches deep, since if the twyers had been placed lower down, the air belt would have prevented inspection of the casing, to note signs of burning through; the air belt also prevented the hearth bottom or tap holes being raised. The upper part of the furnace was unaltered. Charging commenced at 10.30 a.m. with the same mixture of equal parts of titaniferous and silicious ore used in F., *i.e.*, 56 lb. silicious, 56 lb. titaniferous, 28 lb. dolomite, and 225 lb. coke. The blast was kept about 6 oz., which enabled the hearth temperature to be kept up while reducing the rate of charging to nearly once per hour, and therefore increasing the time in the furnace to about 6 or 7 hours. The effect of this was manifest in the fact that small quantities of iron began to run about 9 p.m., from which time the furnace was regularly tapped every 1½ hours. At the same time the old trouble of soft iron bear forming in the hearth was also apparent, and although iron was run regularly till 7 a.m. next morning, it was only possible to open the tap hole with great difficulty.

At this point 24 charges of the mixture had been given, and next 4 charges of 112 lb. titaniferous ore alone, 28 lb. dolomite and 17 lb. sand were added, after which the furnace was allowed to blow down. For a little while a slight improvement occurred, but soon the tap hole completely closed and at 10 a.m. the bottom was dropped; metal and slag were run from the slag hole for a few hours longer, the bear being jammed fast and impossible to shift. In this run again, the slags ran perfectly fluid when the tap holes could be opened, as will be seen from the photograph, and the closing of the holes was entirely due to iron, as before.



The furnace lining was again severely cut and, of course contributed to the bulk of the slag. The iron bear in the furnace weighed 575 lb., and the cast iron collected from the pots 580 lb. Several slag samples were taken and analysed, with the following results:—

	SiO_2	TiO_2	FeO
3rd 5 p.m. ...	43 9%	9·0	13·0%
3rd 9 p.m. ...	40·4	8·6	14·8
4th 3 a.m. ...	44·3	7·7	9·9
4th 5 a.m. ..	39·8	4·0	12·5
(blast increased unintentionally)			
4th 3 p m. ...	38·2	10 2	7·9

TABULAR STATEMENT OF EXPERIMENTS AND RESULTS.

Run.	Composition of Charges (lb.).					Ratio.	Number of Charges.	Iron obtained (lb.).		Slag Analyses.							No. of twyers.	Diameters of twyers (inches).	Blast pressure (ounces).
	Silicious.	Titaniferous.	Dolomite.	Sand.	Coke.	Coke : Ore		Soft.	Cast.	Silica SiO_2.	Titanium Oxide TiO_2.	Ferrous Oxide FeO.	Alumina Al_2O_3.	Lime CaO.	Magnesia MgO.				
A altered to	112 112	56 56	28 56	— —	200 200	1·2 : 1	6 3	60	—	38·85 38·26 39·5	1·2 4·3	44·4 37·18 50·4	17·18	4·4	·32	(M) (H) (M)	5	4	6
B altered to	56 56	56 56	14 16	6·5 6·5	150 225	1·3 : 1 2 : 1	9 3	175	—	38·65 40·5	8·05 8·7	38·0 32·6	9·25	6·00		(M)	5	4	3
C altered to	56 56	112 112	35 37	10 10	225 300	1·3 : 1 1·8 : 1	12 3	200	100	35·85 37·8	9·72 18·0	25·26 28·3	20·05 12·0	7·00 7·00		(H) (M)	3	4	3-5
D	112	—	56	—	225	2 : 1	8					20·8					3	3	3-4
E	112	—	56	—	225	2 : 1	17	70	354	38·5	—	12·5 to 14·0	18·4	16·5	9·25		3 3	2 1½	5-6 5
F 1 2 3	112 56 56	— 56 56	56 28 35	— — 5	225 225 225	2 : 1 2 : 1 5 : 1	10 11 3	500		42·6 38·6	8·3 8·8	7·4 12·0	25·7 26·1	10·0 8 8			3	1½	5-4 6
G altered to	59 —	56 112	28 28	— 17	225 225	2 : 1 2 : 1	24 4	575	580	43·9 40·42 44·27 39·85 38·23	9 0 8·65 7·66 4·00 10·18	13·0 14·79 9·88 12·47 7·88	26·0 25·19 30·01 26·14 25 90	9·0 6·44 5·88 10·88 12·56		(M)	3	1	6

(M) Analysis by Chemist to the Mines Department, Dr. J. Moir.
(H) " A. Heymann.

As the furnace was only lent for a month, and the commencement of the College Session was only a week distant, the experiments had perforce to be concluded with this run.

I am afraid these details make tedious reading, but they are, I think, necessary, in order that the reasons for the various steps taken may be appreciated. The results show, I think, that the experiments were proceeding on the right lines, and with more time and a larger furnace might reasonably have been expected to reach a successful issue. While the general conclusion is that titaniferous ore is certainly more refractory and not so easily reducible as ordinary ores, given a sufficiently large furnace, to ensure plenty of time for reduction, and at the same time a high temperature in the hearth, nothing in the experiments would induce me to anticipate that it would not be successfully smelted, though probably at the expense of a fuel consumption rather higher than the normal. All through the series of experiments our troubles were not due to infusibility of slags—though how far their fusibility was due to the FeO they contained is perhaps a moot point—but to non-carburisation and consequent infusibility of the iron ; this could hardly be the case in a large modern furnace, which would also consume much less fuel per ton of iron than in these tests. In this connection it is well to note that the cupola used for these experiments is not of very different size from the large furnaces formerly used for smelting *soft iron* from ordinary iron ores, and in which, probably, the accidental production of cast iron led to the establishment of cast iron smelting, soon requiring larger furnaces.

Increasing amounts of titanium did not seem to impair the fluidity or fusibility of the slags, and an interesting point which was noted was that although, like all the others, the highly titaniferous slags had black or dark-grey fractures, their surfaces were light-grey or nearly white, this also lending colour to the view previously expressed as to the existence of titanium as a lower oxide than TiO_2, in the slag, which oxidised to TiO_2 on the surface. Assuming that titanium exists in the charge as TiO_2, the question is still open as to whether or not it is reducible to a lower oxide by carbon monoxide or only by solid carbon or other bodies, and at what temperatures. It seems probable that it is only difficultly reducible, and in that case may reach the smelting zone unchanged, and there act as a re-oxidiser of carbon or silicon already taken up by the iron, so retarding the production of cast iron, and although unable to prevent its production in a large furnace, the iron might always be white.

A statement made in the latest American Metallurgical text book is worthy of note here: Professor L. S. Austin, in the second edition of his book, "The Metallurgy of the Common Metals," says, in reference to iron ores, "some of the New York beds . . . have been considered valueless on account of the presence of titanium, it having been asserted that this element produces an infusible sticky slag. This, however, has been proved to be unfounded, and it should not prevent their use as a source of iron."

The analyses of the materials used are as follows :—

	Titaniferous Ore	Silicious Ore.	Dolomite.
SiO_2 -	1·56	22·10	4·15
TiO_2 -	14·31	not determined	
Fe_2O_3 -	not determined	63·75	3·44
Al_2O_3 -	3·36	10·41	·56
CaO -	not determined	·62	28·66
MgO -	not determined	trace	18·10

The charges were calculated as explained in Part I. of this paper, but the high proportion of coke used, together with the large proportion of aluminium in the silicious ore, made it impossible to form a very highly titaniferous slag. Even with the titaniferous ore alone, after adding dolomite, as required to make the aluminium silicate fusible, which in turn necessitates more silica (as sand) there could not theoretically have been more than about 21% of TiO_2 in the slag, and for the same reasons silica had to be greatly in excess of the TiO_2, and not nearly equal, as advocated. With less, and better, coke this would be more nearly possible. In all the analyses, as the form in which the titanium exists is not certainly known, it is expressed as TiO_2.

In conclusion, I have again to thank Mr. Murray for help in the conduct of the experiments, and also Mr. F. W. Penny. My thanks are also due to Sir T. R. Price and Mr. G. G. Elliot and his staff at Pretoria for giving every assistance in their power, and to Mr. Kotzé, as representing the Government Mines Department, which provided the financial backing, and thereby rendered the experiments possible.

Mr. Chas. B. Saner (*Vice-President*): It affords me great pleasure to propose a hearty vote of thanks to Prof. Stanley for his interesting paper. He mentioned in one paragraph that the details may appear tedious, but I think the remarks in that paragraph are particularly interesting to us because they show that he is working on right lines, and he is practically opening up a new and great industry not only for

South Africa but the whole world. For that reason I think he should have a very hearty vote of thanks from this Society. I will ask Mr. Vaughan to second my proposal. He may probably be able to give us some interesting side lights on the recent experiments.

Mr. J. A. Vaughan: I have great pleasure in seconding the vote of thanks though, unfortunately, I was not present when the paper started. As regards side lights, I am afraid it is rather hasty to be called upon at this moment to give you side lights. When certain reports are published then probably I may feel privileged to give you some account of what I have seen, and the deductions I have made. The subject dealt with to-night by Prof. Stanley affects the whole world just as much as the Transvaal. The smelting of titaniferous iron ore has not been looked upon as commercially practicable in other parts of the world where they have a great deal more of it than here. Prof. Stanley's experiments have apparently proved it commercially possible. This may go a a great way towards cheapening the production of iron, but it is not so much that, as the extension of the available iron ore beds of the world.

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

(Reprinted in this Journal, September, 1909, from the Bulletin of the Institution of Mining and Metallurgy, August 12, 1909.)

By W. A. Caldecott (Member).

DISCUSSION.

Mr. E. J. Way (*Member*): The author gives us a very interesting history of the development of the gravitation stamp, from the square wooden non-rotating stamp used by the German miners in the Middle Ages to the heavy-weight rotating steel stamp at present in use on these fields. I must, however, confess to a feeling of disappointment when I found on reading the paper that, while Dr. Caldecott is emphatic in the statement of his personal opinion that the increase of weight already carried out, does show very appreciable economic improvements, this opinion is not supported by any convincing arguments or array of experimentally proved data. I felt this the more particularly since so much has been heard in recent years of the increased economic efficiency due to the corresponding increase of weight of the gravity stamp, especially from the mines with which Dr. Caldecott is associated, that I had been led to hope that the author's unrivalled opportunities for arriving at definite conclusions, based on experimentally determined facts, would have enabled him to throw some very clear light at last on that most important question, what are the limits within which primary and secondary crushing machines would operate at their best efficiency and give the best return for the money invested? I had hoped that this question would have been fully treated from every standpoint, namely, first cost, running and maintenance costs, life of mine, and all the other factors that enter into the question of economic efficiency in contradistinction to mere mechanical efficiency.

The paper opens in a very promising manner. The author states (see p. 109) that the object of the trials carried out at the Knights Deep in 1904-5 by the Consolidated Goldfields was "to discover some means whereby stamp milling efficiency could be increased," and as he goes on to say that the result (see p. 110) of these trials was the adoption of heavy weight stamps at the Simmer East, Robinson Deep and the Luipaardsvlei Gold Mines, it might reasonably have been expected that he would have been able to provide demonstration that increase in weight of stamps results in very appreciable improvements in economy.

The laudable and useful objective which was the *raison d'être* of the 1904-5 experiments seems soon to have been lost sight of, and the experiments to have been carried out merely with the object of discovering how the capacity of the stamps could be increased, as it is stated (see p. 110) that, as the result of the experiments, "it became obvious that to crush more rock it was necessary to strike a harder blow." Moreover, the trials carried out do not appear to substantiate the author's contention that the increase in weight so far adopted has resulted in improved economy. A careful examination of the data brought forward in the paper (see table on p. 5) on which apparently the author based his recommendation that the heavy-weight stamp should be installed at the three mines previously named (a recommendation by the way hardly confirmed by the total working costs per ton milled on these mines at the present time) simply shows that, the number, height of drop, and height of discharge remaining the same, the extra amount crushed is almost exactly directly proportional to the weight of the stamps. In other words, there is no increase of efficiency but only of capacity, as the increase is proportional to the extra weight lifted or to the power input of the mill.

A further examination of the table on p. 112 shows that, in order to obtain an increase in the tonnage crushed proportional to the increase in the weight of the stamp, it was necessary with an increase of 32% in the stamp weight, namely, from 1,216 lb. to 1,605 lb., to increase the water per ton of ore crushed 41%, or from 4·26 tons to 6·02 tons per ton of ore crushed, whilst the amount of +60 (0·01 in.) product increased 126%, or from 5·16% to 11·66%. This would appear to indicate that, so far from there being an increase in economic efficiency due to crushing large tonnages through fine mesh screening by means of heavy stamps, there is actually a decrease in the economic efficiency. It is impossible, however, to come to any definite conclusion on this point as the data supplied are insufficient, the author giving neither a complete grading analysis of the screen pulp, the percentage of extraction obtained, nor any of the other data influencing the final results in connection with, either the economies claimed by him as resulting from a reduction of the number of units, or, the number of units being the same, from the larger total output and consequent shorter life of the mine.

Dr. Caldecott's experiments on fine preliminary breaking with rolls before milling might with profit have been carried a little further by trying the effect of fine breaking in conjunction with lighter stamps (or, what would have the same effect, larger area of shoe), lower drop, and greater number of blows per minute; as it would certainly appear, theoretically, that for every size of feed there should be, in order to obtain maximum mechanical efficiency, a corresponding pressure over unit area resulting from the blow of the stamp and depending upon the three factors, weight of stamp, area of shoe, and height of drop. It would appear reasonable to expect that for a very fine feed a light stamp with a very low drop and running at a high rate of speed would prove most efficient, and it is to be regretted that the author's experiments were not carried further in this direction, as it seems probable that had this been done an increased efficiency would have been found to result from an increase in the fineness of the preliminary breaking when combined with a corresponding alteration in the other factors influencing the result, namely, stamp weight, shoe area, and height of drop. The increase in the number of blows per minute would, as was found to be the case in the pneumatic stamp trials carried out at New Kleinfontein, probably largely increase the duty of the stamp, so that even with the light stamp large duties might reasonably be expected. Had the author continued his experiments along the lines suggested much valuable data bearing on the very important question of the correct size of ore to be fed to the stamps might have been obtained.

The author states (see p. 113) that "fine breaking before milling with heavy stamps does little good," and goes on to say that "the heavier the stamp the coarser the preliminary breaking admissible, and *vice-versâ*," in other words, the heavier the stamp the coarser the preliminary breaking admissible and, conversely, the lighter the stamp the finer the preliminary breaking admissible. I am of opinion that here the author is getting to the very root of the question. I am inclined to believe that, if we were to look upon the heavy-weight stamp as a fine breaker and were to interpolate it, in place of Dr. Caldecott's fine rolls, between the rock breakers proper and the light stamps, very appreciable gains in economy might result, as it must not be forgotten that the heavy-weight stamp even when crushing through 4-mesh screening produces as a by-product some 25% of −90 (0·0062 in.) grade.

The fact, proved by the investigations carried out by the Mines Trials Committee, that maximum mechanical efficiency when using heavy-weight stamps is only obtained by crushing through correspondingly coarse screens, moves the sphere of action of heavy-weight stamps more and more, with increase of weight, in the direction of coarse crushing. The economic limit of the weight of these stamps, leaving out mechanical difficulties, appears to be only reached when they encroach on the province of the jaw breaker or gyratory crusher, with which they cannot compete either mechanically or economically owing to the breakers having a positive feed and discharge and requiring no water feed. This economic limit will be reached when the mechanical advantages of the breaker are balanced by the advantages accruing from the percentage of −90 (0·0062 in.) grade produced by the heavy-weight stamp when crushing through even the coarsest mesh screening.

Stamps of the maximum economic weight when crushing through 4-mesh screening produce, as I have already pointed out, some 25% of −90 (0·0062 in.) grade, and would therefore leave only 75% of the total ore milled to pass to the light stamps. Now it has been proved by the experiments carried out by the Mines Trials Committee that the mechanical value of the pulp is not affected by the weight of the stamp, the other variables remaining constant, and that for every weight of stamp there is a mesh aperture giving maximum efficiency. If therefore a mesh aperture be chosen, giving the desired percentage of +60 (0·01 in.) grade in the battery pulp or the desired mechanical value of this pulp, and the correct weight of stamp for this mesh aperture be determined by means of the data resulting from the investigations carried out by the Mines Trials Committee, and if these stamps be then fed with carefully classified pulp from the heavy-weight stamps, a product of the required degree of fineness would be obtained without resorting to tubemills or other fine grinders.

As stamps of the maximum weight introduce many engineering difficulties, owing to the heavy strains and vibration consequent upon their weight, and to the very large storage bins that they require, I think it is quite a debatable point whether better all round efficiency would not be obtained by crushing through finer screening with stamps of a weight which would not entail these special difficulties. For instance, if stamps of 1,700 lb. weight were used with 400-mesh screening they would have a duty of 10 tons and would produce some 50% of – 90 (0·0062 in.) product, leaving only 50% of the total ore milled to pass to the light stamps.

As both the heavy-weight and light-weight stamps, if properly designed for the size of the ore fed and for the screening used, would be working at their maximum mechanical efficiency, and, as the mechanical efficiency of a tubemill working under the best conditions is only about 25% to 33% of that of a properly designed stamp battery (see report, by Mr. Stadler to the Mines Trials Committee, on Coarse Crushing Experiments at the Simmer East, dated 2nd November, 1902), it would appear that both the mechanical and economic efficiencies of a plant thus equipped should be considerably better than that of the present fashionable heavy-weight stamp plus tubemill combination.

Any difficulty with regard to weight of stamp, in the case of stamps of maximum weight, or to speed of running, in the case of the light stamp, could be overcome either by the use of the Holman pneumatic stamp, as has been recently proved at the New Kleinfontein Mine, or by the Morrison stamp, with its hydro-pneumatic cushioning cylinder operated by a crank shaft and connecting rod, some of which ran two years continuously at the Meyer and Charlton.

On reference to the table on p. 5 it will be seen that with a mesh aperture of 0·016 in. only 5·16% of the screen pulp remained on a 60 (0·01 in.) screen. Now at New Kleinfontein an extraction of 94% is obtained with battery pulp containing as much as 14% of + 60 (0·01 in.) grade. It would therefore appear that, at New Kleinfontein at least, a mesh aperture of 0·016 in. is sufficiently fine. As the stamp to obtain maximum efficiency with this screening is quite a light one it would be possible to run a battery equipped with these stamps at a very high rate of speed indeed, and, owing to the fine feed from the heavy-weight stamps, with a very low drop.

I have very little doubt that a stamp of the Holman type, or of the design suggested by Mr. Behr, could be designed to give a duty of at least 10 tons when fed with carefully classified + 90 (0·0062 in.) product, thus greatly reducing the number of light stamps required, and, as even an inefficient stamp is some three or four times more efficient than a tubemill, at the same time largely increasing the economic efficiency of the plant.

Some difficulty has been experienced, I believe, with the discharge of the pulp from the mortar boxes of the type of light-weight stamp I have been suggesting, but I have no doubt that the difficulty could easily be overcome by a correct design of mortar box, and by a series of experiments to determine the correct height of screening and the correct area for the discharge.

When it is remembered that there are at present large numbers of plants in various parts of the world, the capital expenditure on which per ton milled per 24 hours would probably compare favourably with the capital expenditure on plant on these fields, and which are working satisfactorily both with regard to running and maintenance costs, in which coarse breaking followed by fine breaking takes the place of stamp crushing, it must be admitted that the whole question is still a very open one, and that further experiments, possibly on the lines suggested above, and much careful consideration of all the factors bearing on the question are required before it would be justifiable to assume that the heavy-weight stamp plus tubemill combination is the final solution of the crushing problem.

It appears to me that (considering the very poor efficiency of the tubemill, the heavy capital expenditure involved in its installation, and its very heavy running costs), the caution, not to say conservatism, usually displayed by the leading engineers of the Rand in adopting new appliances having far-reaching effects on metallurgical practice has in the present case, in some instances at least, been conspicuous by its absence, and that it is open to question whether in some of the recently equipped mines, the economic limit has not been exceeded in the over anxiety displayed, by the designers of the plants in question, to reap the benefits claimed by its advocates as accruing from the use of the tubemill. As this impression seems to be borne out by the reduction costs of the mines using tubemills, I have compiled the following table, comparing the costs on the mines where double-stage crushing is in vogue with the costs at New Kleinfontein where crushing by the old single-stage method is still in use.

I at first attempted to compare the milling costs directly, but found that the different systems of apportioning charges in use on the mines rendered this impossible. For the same reason it was impossible to use the total reduction costs as a basis of comparison. I have therefore given in separate columns the reduction expenses, the general charges, and the total costs, and have used the figures in the last column as a basis for comparison, as this seemed the only fair method, as I found the term "General Charges" to be a very elastic one and to have quite different meanings on the various mines. For example, at New Kleinfontein the reduction expenses include the maintenance of surface plant, whereas on other mines a proportion of this maintenance cost is apportioned to general charges.

It will be noticed from the tables that only four mines show lower reduction expenses than the New Kleinfontein Company, namely, the Simmer & Jack, Luipaard's Vlei Estate, Village Deep, and the West Rand Consolidated. The low expenses at the Simmer & Jack are probably due to the large tonnage treated, whilst the low costs on the Village Deep and Luipaard's Vlei Estate appear to be due to a larger share of the costs being apportioned to general charges, which amount to 1s. 9·128d. and 1s. 11·201d. respectively, as compared with 0s. 6·146d. at New Kleinfontein. As to the West Rand Consolidated the costs must be compiled on a different system to the one commonly employed, as the general charges amount to 3s. 7·633d. per ton milled.

Comparison between Reduction Expenses, General Charges, and Total Costs, of the New Kleinfontein without Tubemills and Sundry Rand Mines using Tubemills.

Name of Company.	Tonnage Milled per Month.	Reduction Expenses (per ton milled).	General Charges (per ton milled).	Total (per ton milled).
NewKleinfontein	38,599	4/0·748	0/6·146	4/6·894
Companies using tubemills.				
Simmer & Jack	68,900	3/9·684	1/0·607	4/10·291
Knights Deep ...	49,499	4/2·022	0/11·350	5/1·372
Wit. Deep ...	36,723	4/6·92	0/8·70	5/3·62
Simmer Deep ...	48,952	4/5·504	1/0·671	5/6·175
East Rand Proprietary Mines	157,243	4/11·734	0/6·997	5/6·731
Rose Deep ...	39,883	4/7·736	0/11·541	5/7·277
Simmer & Jack E	29,727	4/3·632	1/4·233	5/7·865
Village Deep ..	32,970	3/10·783	1/9·128	5/7·911
Luipaardsvlei E.	16,119	3/9·437	1/11·201	5/8·638
Village Main R.	43,600	4/8·713	0/11·982	5/8·695
Robinson Deep..	51,366	4/3·552	1/5·581	5/9·133
Ferreira Deep ...	31,772	4/7·561	1/2·876	5/10·437
New Modderfont.	45,200	4/5·617	1/5.577	5/11·194
Crown Mines ...	121,100	4/4·739	1/6·886	5/11·625
Knights Central	22,378	4/5·91	1/6·09	6/0·
Robinson G. M.	57,588	4/8·549	1/3·674	6/0·223
Jupiter	24,570	4/6·843	1/7·079	6/1·922
Wolhuter ...	27,150	5/4·618	0/10·959	6/3·577
Nourse Mines ...	37,928	4/6·410	1/10·381	6/4·791
New Goch ...	24,740	4/9·105	1/10·772	6/7·877
Van Ryn ...	29,852	4/8·303	2/0·073	6/8·376
Geldenhuis Deep	39,753	5/1·866	1/9·175	6/11·041
New Rietfontein	17,317	4/6·456	2/8·352	7/2·808
West Rand Cons.	17,672	3/8·248	3/7·633	7/3·881
Cinderella Deep	13,129	5/8·195	2/1·287	7/9·482
Meyer&Charlton	13,162	5/7·582	2/2·652	7/10·234
Roodepoort Unit.	17,683	5/6·643	2/5·551	8/0·194
Durban Roode. D.	17,197	5/4·539	2/8·135	8/0·674
Treasury ...	8,873	7/2·09	1/3·48	8/5·57
New Heriot ...	11,583	6/8·216	2/0·216	8/8·432

Coming to the table on p. 112, it will be seen that the author states that in May, 1907, the cost of single-stage crushing through 1,600-mesh (0·016 in.) and 1,200-mesh (0·017 in.) screening (the battery pulp containing 10·92% of +60 (0·01 in.) grade) was 1s. 10·424d. per ton milled, and that in January, 1909, crushing with stamps through 200-mesh (0·057 in.) and 400 mesh (0·035 in.) screening and regrinding in tubemills, to a final product of which only 1·61% remained on a 60 (0·01 in.) screen, the cost was 1s. 10·742d. per ton milled. There was therefore a difference of 0·318d. per ton milled in favour of single-stage crushing, a difference which would probably be accounted for by the difference in the tonnage milled during the two periods under review, namely, 35,500 tons in May, 1907, and 29,600 tons in January, 1909.

The costs, as given by the author, can, however, hardly be taken as a fair criterion of the two systems of crushing, as he does not state whether he has taken into consideration the initial capital outlay in the two cases, with redemption and interest charges properly apportioned; or whether he has allowed for all the other factors involved, such as comparative output, life of mine, &c., nor does he state whether he has made allowance for the very large and general fall in working costs that has taken place between 1907 and 1909, due to cheapening of labour and of supplies and to other things not directly dependent upon the method of crushing adopted. Neither has he let us know whether the reduction in the percentage of the +60 (0·01 in.) grade from 10·92% to 1·61% has been followed by an increase in the extraction sufficient to meet the additional cost of the finer grinding. In connection with this point I may again remark that with battery pulp containing some 14% of +60 (0·01 in.) grade an extraction of 94% is obtained at New Kleinfontein, so that it would appear doubtful whether the cost incurred in reducing the percentage of +60 (0·01 in.) grade to as low as 1·61% is defensible.

From the author's summing-up it is evident that he has, as I have previously mentioned, shelved the question of any increase in stamp milling efficiency being obtained by the installation of heavy-weight stamps and tubemills, and that he now bases their claims to adoption on the ultimate economy of the whole plant, due to economies in the erection and running of a mill resulting from a reduction in the number of units, and to an increase in the output with a corresponding decrease in the life of the mine.

In connection with the economies in capital expenditure on plant, the advocates of the heavy-weight stamp plus tubemill combination claim that its adoption has reduced the cost of mine plant per ton of ore milled per 24 hours by more than two-thirds. This reduction I cannot admit to be due to the introduction of the heavy weight stamp and tubemill, as if it was due to this innovation in metallurgical practice it would surely be clearly brought out in the tabular statement on page 365, comparing the cost of mines which have adopted two-stage crushing with the costs at New Kleinfontein, and that this is not the case a reference to this table will at once make clear. The reduction in capital expenditure would appear to me to be mainly due to three causes:—first, the use by the mines of electric power generated at large central stations, secondly, the far larger scale on which milling operations are now carried out, and, thirdly, the general cheapening of labour and supplies during the last few years, and to have little, if anything, to do with the introduction of the heavy-weight stamp and tubemill. That the author fully believes that double-stage crushing is more economical than single-stage crushing can be seen from his statement (see page 111) that "the later use of tubemills as well (namely, with heavy-weight stamps) yielded at the same total cost a product with fewer coarse (plus 0·01 in.) particles and hence a still more perfect exposure of the gold," and from the following summing-up (see p. 115) of the advantages of heavy-weight stamps as compared with light-weight ones:—

1. "Reduction of the initial capital expenditure in erecting, say, 200 stamps at 1,750 lb. with accessories, in place of 280 stamps at 1,250 lb. each."

2. "Reduction in size of mill building, almost proportional to the less number of stamps."

3. "30% less shafting, belts, and other moving parts to maintain."

4. "30% less labour required for dressing plates, lubricating moving parts, changing screens, and other work incidental to milling operations."

Before commenting on these four statements I must say that they appear to me to savour of very special pleading, as though the author is fully aware that it has been conclusively proved by the Mines Trials Committee that the maximum efficiency of the heavy-weight stamp can only be obtained when crushing through coarse mesh screening, thus necessitating the use of secondary grinding machines, and, as I have just shown, admits that the use of tubemills improves the efficiency of a heavy-weight stamp installation, yet he omits all mention of the tubemill when summarising the advantages of the heavy-weight stamp as compared with the light-weight stamp, thus entirely vitiating all his conclusions.

Coming now to these four statements I should like to make the following observations:—

1. The reduction, if any, in capital expenditure on stamps and accessories is not as great as at first sight might appear to be probable, as the total weight of the stamps themselves, in the two cases suggested by the author, is identical, whilst the far greater vibration consequent upon the heavier weight of the individual stamps, necessitates the use of massive concrete foundations in the place of the old timber piles, the replacement of the wooden king post by steel or reinforced concrete constructions, and the installation of far larger and heavier cams and cam shafts, pulleys, friction clutches, etc. Again, although the water required per ton of ore crushed may be reduced (although this is a point still under investigation by the Mines Trials Committee) the total quantity passing through each box will be more than doubled, necessitating a larger water service to each box, whilst the larger ore feed will necessitate bigger and more expensive ore feeders. If to the cost of the stamps be added the cost of the necessary (if the stamps are to work at their maximum efficiency) secondary grinding plant, such as the tubemill circuit with its launders, cones, pumps and other equipment, it will be obvious that the cost of the crushing appliances alone will be higher with heavy-weight stamps than with light-weight stamps.

2. As to there being any reduction in the cost of the mill building, this is more than doubtful if the mill bins are still retained as an integral part of the building. The usual allowance of three days' storage capacity means that the old allowance of 12 to 15 tons per stamp will now be increased up to 60 tons per stamp, or possibly more, necessitating in the case submitted by the author a storage capacity of 60 × 200 or 12,000 tons as against one of 15 × 280, or 4,200 tons. This enormous increase in the capacity would vastly increase the strains throughout the bin structure, and would probably make the actual cost of the mill buildings more with the heavy stamp than with the lighter stamp; whilst as the capacity of the conveying belts will be correspondingly increased they will be both larger and more costly. Mr. Robeson has suggested storing the "fines" outside the mill building and supplying the small feeder bins inside the mill building, which would still be necessary, by means of a system of belt conveyors. If this method were adopted there would still be the additional capital expenditure on the storage arrangements outside the mill building, and, in addition, extra machinery to maintain, and increased running costs to face, as it would be necessary to have the conveying belts running

over the week-end. I do not think therefore that this method would prove cheaper in the long run than the existing plan of having the storage bins an integral part of the mill buildings, though from other points of view it possesses distinct advantages. It must not be forgotten too that to the cost of the mill building must be added the cost of the buildings for the tubemill installation.

3. There will certainly be some 30% fewer moving parts, but as these will all be on a correspondingly bigger scale, and as the wear and tear will be more than correspondingly greater, owing to the duty per unit being increased some two, three, or four hundred per cent., I very much doubt whether the maintenance and renewals costs in the mill alone will be greatly reduced, and if to these costs are added the tubemill maintenance costs, the total maintenance costs will almost certainly be higher.

4. I do not see that there can be as large a decrease as 30% in the labour requirements. Take the question of dressing the plates mentioned by the author. It has been found during experiments carried out by the Mines Trials Committee that, when crushing through coarse mesh screening, it is necessary, owing to the scour due to the large particles and the increased grade of the plates required to allow the pulp to flow freely, to remove the plates from the mill building and place them below the tubemills. Here there will be no reason why their number should correspond with the number of mortar boxes in the mill, and as a matter of fact their number will probably be not less than in a light-weight stamp mill of similar capacity, so that the labour of dressing the plates will practically be the same in both cases; whilst any saving of labour in the other work incidental to stamp milling will probably be offset by work in connection with the running of the tubemill plant. Generally throughout the plant costs will probably be found to be higher in two-stage crushing than in single-stage crushing, due to, first, the heavier running and maintenance costs of the heavy-weight stamp plus tubemill plant as compared with the same costs of a light-weight stamp mill of similar capacity; secondly, the absolute necessity, if the tubemills are to work at even their low maximum efficiency, of really skilled supervision and labour in the carrying out of all the work in connection with the tubemill circuit, such as the classifying of the inlet and outlet pulps of the tubemills and return spitzkasten, banket feed of tubemills, etc.; and, thirdly, except in a few favourably situated reduction works, the increased pumping costs, both running and maintenance, due either to the steeper grade of all launders, &c., or, the grade remaining the same, to the increased water supply, necessitated by the coarser pulps, resulting in either higher lifts or larger quantities to be pumped. It must not be forgotten too that the smaller and more numerous the units the less the effect, on the output and consequently on the costs per ton milled, of one or more units being temporarily put out of action. The above brief glance at some of the reasons for anticipating higher working costs and at least an equal capital expenditure appears to explain the higher costs which actually occur in practice on the mines where double-stage crushing has been adopted, as shown in the tabular statement on page 355, and when these costs are compared with the low working costs and efficient extraction obtaining at New Kleinfontein with single-stage crushing, it is impossible to help wondering if the engineers responsible for the design of the plants on these mines have not displayed less than their usual caution in accepting all that has been claimed for the tubemill; or if they, in adopting the principle of double-stage crushing, have done so without either adequate consideration of its far-reaching economic effects, or the compilation of sufficient experimental data to enable them to demonstrate whether the increased economic efficiency claimed for this system is based on facts or exist only in the imagination of its advocates.

In concluding these already over long remarks I may say that, personally, I am not fully satisfied that the economies in mill practice claimed by the advocates of two-stage crushing have been demonstrated beyond question, and, that until this has been done, I do not consider that we are justified in assuming that the heavy gravitation stamp plus the tube mill is suitable for adoption on the Rand in every case.

After all, what I believe we are all trying to aim at is increased efficiency, not only from a mere mechanical point of view but also from the economic point of view. That is to say, we want to do our milling in a much better way and at a much cheaper cost. And, above all, any new method or any new combination of methods must, in my opinion, be absolutely fool proof. It is all very well to talk about the grading which is now obtained from the tubemill circuit, but those who work with a tubemill circuit know how easily bad superintendence, during a night shift for instance, will vitiate the work of twenty-four hours. Therefore, whatever our new methods are, having always the idea in view of increased economy and efficiency, they must be fool proof, and, in my opinion, the tubemill is at present very far from being fool proof.

There is only one other point which I should like to speak about. According to the report of the proceedings, Mr. Hugh F. Marriott said: "He might point out that the New Kleinfontein mill was burnt by the Boers during the war, but that they had saved Modderfontein." It would be rather interesting to know what was in Mr. Marriott's mind when he made this remark. Is it to be inferred that no attempt was made to save the Kleinfontein mill, or that the extra bravery of the Modderfontein people saved their mill, or what? The whole statement seems to be so irrelevant to the argument brought forward by Mr. W. Fisher Wilkinson. Moreover, at the time that Mr. Marriott's remark about the Kleinfontein mill was made very little of the original Modderfontein mill which was in existence at the time of the Boer war was working, it having been altered and reconstructed out of all recognition. From my personal knowledge of Mr. Marriott I feel sure he did not introduce this remark in any unfriendly spirit.

REPLY TO DISCUSSION.

M. W. A. Caldecott (*Past-President*): In reply to the instructive discussion upon his paper the author desires to express his appreciation of the courteous acknowledgement of the advances which he has endeavoured to record.

As regards the criticisms of individual speakers, as for instance, Mr. S. J. Truscott, he would explain that the term efficiency was used in the paper as synonymous with capacity, in the sense that if one stamp crushed five tons of ore whilst another crushed six tons of the same ore in the same time to the same fineness, the latter would be considered more efficient. He would in no way contend that a heavier stamp crushed more ore than was proportional to its increased weight, nor that there was any economy in power with heavy stamps per ton of ore milled to the same fineness, except for some slight saving due to the smaller number of moving parts. Mr. Truscott's opinion, however, that "any departure from long established practice on the Rand must necessarily be a departure from efficiency" would, if accepted by those actually engaged in metallurgical work on the Rand, be fatal to all future progress; it is indeed disproved by the changes in milling practice on the Rand of late years, as well as by the development of coarse crushing detailed later in this reply, which has already affected much current design of crushing plants.

As to the suggestion that heavy stamps should be used for coarse ore and lighter stamps for fine ore, it must be remembered that the key-note for large modern Rand reduction plants is simplicity of arrangement, concentration of similar operations in the same portion of the plant, and the use of as few and as large units as possible. Moreover, even if coarse ore only be fed to stamps, the mortar-box always contains a mixture of coarse and partially crushed material, so that the same effect is produced as if fine ore had been fed with the coarse. As the author has endeavoured to explain, the disadvantage of feeding fine material only is that the stamp falls upon a more uniform bed, resulting in a low pressure per square inch and in crushing by abrasion instead of by more efficient impact.

In regard to Mr. H. S. Denny's contribution to the discussion, the fact that all recent mills on the Rand have installed heavy stamps renders further investigation here of light stamps largely academic, whilst many other urgent problems in ore crushing press for experimental solution. Mr. Denny's assumption, however, that a stamp can crush more banket ore if finely broken is obviously incorrect in view of the results of actual runs given in the paper. So far as size of ore fed to mortar-box is concerned, the author considers that provided no larger piece is fed than the stamp can shatter by a single blow, at the same time nearly reaching the die before coming to rest, so that the full efficient height of drop is secured, there is little to be gained by finer preliminary breaking. When his previous paper on finer crushing of Rand banket ore was written* the finer preliminary breaking idea seemed very attractive, but the subsequent experiments recorded in the present paper have dispelled it, and this attitude is confirmed by the opinion expressed by Mr. Walter McDermott in the present discussion. No reference was made in the paper to the high-speed Morrison-Bremner stamps tried at the Meyer and Charlton, because they could not be considered as included among the ordinary cam-lifted gravity stamps under discussion, and because after a long test they did not commend themselves sufficiently to millmen on the Rand to ensure their survival.

The reference to the passing of the stamp as a unit of crushing capacity was merely intended to indicate that such variable tonnages are obtainable under different conditions, that of two mills, each with the same number of similar stamps, one might crush twice as much ore per day as the other. Whilst for soft friable ore causing little abrasion there are quite possibly better pulverizers than stamps, yet for hard abrasive ores, such as Rand banket, no serious competitor to stamps in some form with their characteristic impact action has yet appeared.

Mr. S. H. Pearce, to whom the author's acknowledgements are due, has discussed the remarks by Mr. H. F. Marriott upon screening, so that it is hardly necessary to again point out that the I.M.M. standard screens are for grading

---

*Trans. *I.M.M.*, Vol. xiv., p. 48, Oct., 1904.

analyses, while the C. M. & M. tables refer to battery screening.

The suggestion that crushing plant, comprising both stamps and tube-mills, should be considered as a whole is of course correct, and this view has been usual on the Rand since the introduction of secondary crushing appliances in the form of tube-mills. In this connection the author has found very convenient for purposes of calculation what may be termed a "nominal crushing unit" ("N. C. U.") defined as a stamp with a running weight of 1,250 lb. and an assumed new weight of 1,350 lb. The number of drops per minute is assumed to be 100, the set height of drop 8 in. and the actual height of drop 7½ in. A tube-mill 5½ × 22 ft. is assumed to be equal to 30 such nominal crushing units and likewise requires about 100 h.p. to operate; this ratio of stamps to tube-mill is not constant, but varies somewhat according to the conditions of operating. The capacity of the above nominal stamp is rated by the tons of −90 mesh (0·006 in.) product it yields per 24 working hours, and the capacity of stamps with different running weights but operating under similar conditions is calculated in proportion to such different running weight, whilst the capacity of tube-mills of other dimensions than those cited is taken in proportion to the cubic capacity of their shells. Thus a stamp with 1,400 lb. running weight is equal to 1·12 nominal crushing units, and a tube-mill 14 ft. long × 4 ft. diameter is equal to 10 nominal crushing units. This simple rule is of value in determining the crushing plant required when a final pulp containing a given percentage of −90 mesh product is desired. Thus, to estimate how many tube-mills (5½ ft. × 22 ft.) must be added to a plant containing 200 stamps with a running weight of 1,400 lb. in order that 1,420 tons daily of pyritic banket ore should be crushed to a pulp containing 80% −90 mesh product, and assuming likewise that a nominal crushing unit produces 3·3 tons −90 product per 24 hours:—

80% of 1,420 tons = 1,136 tons −90 product.
200 stamps at 1,400 lb = 224 N.C.U.
224 N.C.U. × 3·3 tons = 739 tons −90 product.

$$\frac{1{,}136 \text{ tons} - 739 \text{ tons}}{30 \times 3{\cdot}3} = 4 \text{ tube mills } (5\tfrac{1}{2} \text{ ft.} \times 22 \text{ ft.})$$

The assumption that the production per 24 hours of −90 mesh product in the foregoing illustration is 3·3 tons per N.C.U. in crushing plant is, of course, liable to variation under different conditions. Thus, as shown by the coarse crushing trials detailed later, the production of −90 mesh product per N.C.U. of the crushing plant is greater with coarse than with fine battery screening, whilst if a very fine ultimate product is obtained the tons of −90 mesh product per tube-mill fall off, both because the tube-mill feed is finer on an average, and because the last few per cent of coarse particles in the pulp are naturally derived from fragments of the hardest and toughest pebbles in banket ore, and are thus less easily shattered than the softer matrix. The utility of the −90 mesh product as a working basis for calculating the efficiency of crushing[*] is due to the fact that present knowledge of banket ore treatment demands that as high a percentage of the ore as is economically possible should be crushed fine enough to pass such a screen. In other words, −90 mesh product being that ultimately sought, the more tons of this material that can be obtained per stamp, or per tube-mill or per horsepower the better. At the same time it does not from a scientific standpoint disclose the whole truth, as, for instance, the work done by stamps in reducing the ordinary mill feed of ore to particles fine enough to pass the battery screening but not small enough to pass a 90 mesh. The system above described, however, serves to render approximate comparisons and records possible, by expressing in commensurate figures the relative effect produced by stamps of various running weights, tube-mills of various dimensions, and indeed crushing appliances of any kind.

Mr. McDermott's corroboration of the author's attempt to expose some popular fallacies is welcome, as also his confirmation of the view that the harder the blow struck the coarser the screen advisable, so that steam or similar stamps are unsuited for fine stamp milling. Since even with gravity stamps the maximum efficiency is obtained with coarse screens it is probable that the highest efficiency practicable under existing conditions with steam or pneumatic stamps, striking much harder blows, would be obtained with the coarsest screens compatible with subsequent tube-milling, so that a relatively thin layer of coarse particles on the die suffices to take up the force of impact, instead of a thicker layer of fine particles producing a cushioning effect.

Mr. E. J. Laschinger's arrangement in the City Deep mill of multiple cam-shaft bearings[†] with 2,000 lb. stamps, which have an estimated 11 ton duty, constitutes a logical means of coping with the shocks due to impact of cam with tappet, which naturally increase in severity with increased stamp weight. Whether some form of lifting device other than the cam will finally be adapted to the relatively small crushing unit which the present stamp constitutes it is difficult to say, or whether the encouraging results of recent coarse crushing trials which are given later in this reply will result in the use of large pneumatic or other stamps with coarse screening, and comparable in duty with the steam stamps on Lake Superior. This last system, however, appears the most promising, and it is to be hoped that the pioneer work already carried out at the New Kleinfontein mill in this direction will be further developed. With the elimination of amalgamation in the battery, rolls to reduce ore to pass a screen of 9 holes per square inch would produce a suitable product for tube milling at a low capital cost, could steel be found for shells which would preserve true faces by withstanding abrasion well, and thus minimise stoppages and cost of truing the roll faces. The comparative advantage of the stamp in this respect is that even though the crushing surfaces of shoe and die wear irregularly yet a fair crushing efficiency is maintained.

---

[*] See this *Journal*, Vol. vii., Sept., 1906, p. 72.

[†] *S.A. Mining Journal*, Sept. 11, 1909, p. 771; also this *Journal*, Feb., 1910, p. 299.

Mr. Alfred James' mathematical calculation as to crushing efficiency is doubtless correct so far as the initial blow upon a single cube is concerned, but the author had in mind not only the work done at the first moment of impact of the stamp, but also that done as its fall continued upon the fragments of the cube first struck. As to the use of heavy stamps for light porous ores, the writer would propose increasing the number of drops per minute, reducing the height of drop, and increasing the area of shoe and die. If stamps were employed at all for this purpose, the ultimate result should be the same whether the necessary pressure per square inch were obtained by one 1,600 lb. stamp with 100 sq. in. of shoe area, or two 800 lb. stamps each with 50 sq. in. of shoe area. Whilst Mr. G. H. J. Hooghwinkel's view, that five heavy stamps on one cam-shaft are sufficient, would be generally endorsed, yet his assumption that electric driving necessarily means steady running is not obvious, since variations in the speed of the prime mover would naturally affect the mill motors.

Mr. E. T. McCarthy's remarks as to the possible adverse effects of heavy stamps and high duties upon amalgamation appear to be explainable by the increased velocity and depth of pulp stream, which result from higher stamp duties unless the width of plate be increased. With ore carrying coarse gold amalgamation under like conditions should be facilitated, whilst quite apart from any effect of impingement, it is natural that most gold should be caught at the head of a plate where the pulp is richest. The whole tendency in amalgamation on the Rand has been to eliminate mercury wells, and steps or falls in plates, and to rely upon a single amalgamated inclined plate surface.

Mr. C. E. Brown has justly laid stress upon the the importance of the running weight of a stamp, as distinct from its new weight. In trials the actual running weight of stamps would of course be determined by weighing or measuring, but in practice the average running weight of a stamp without compensating weights and with half-worn shoe may be taken as 8% to 10% less than its new weight, unless the head has been much worn by long abrasion.

Mr. Hardinge's results* confirm the advantages of coarse crushing in the stamp battery, as his ore apparently yielded about 6 tons of – 90 mesh product per nominal crushing unit with a screen of 16 holes per square inch. His view, however, that stamps will soon become obsolete seems to neglect the consideration that the production of fine material is a necessary concomitant to the shattering of the 2 in. cubes by the stamp impact. Provided that this fine product is promptly removed by discharge through the screen so as to avoid cushioning the succeeding blows, its production is a gain as being so much of the final work done. In other words, the step reduction with 0·2 in. aperture might approximate more closely to 1 : 1,000 than to the very high ratio he mentions.

Mr. Urquhart's interesting contribution emphasises the unsolved nature of many problems in stamp-milling, such as the cause of irregular wear of dies.† Any accepted solution of such a problem will probably be due to the careful observation of men whose duties allow constant opportunities for noting and reasoning upon the behaviour of batteries in operation.

It is impossible within the limits of this reply to discuss the many important matters raised by Mr. C. O. Schmitt in his able contribution from the constructional engineer's standpoint, which indeed deserves to be regarded as a paper in itself, and the author hence merely desires to express his indebtedness to Mr. Schmitt for the large amount of reliable information that he has furnished to his fellow workers upon a subject of which much still remains to be known.

Mr. Shockley's remarks as to the desirability of shoes with flat bottoms might be extended with possibly even greater force to flat-topped dies, the uneven wear of which was referred to in the paper. Were shoes and dies to preserve parallel plane horizontal surfaces it would be possible to run with a quite thin layer of ore upon the die, instead of requiring, in order to prevent pounding a certain minimum thickness of ore above the highest part of the die, with the result that elsewhere the ore layer is unduly thick and the blow correspondingly cushioned. Whilst the periodical truing of the crushing surfaces would, no doubt, add to the efficiency of crushing it would be at the expense of labour and metal, and any practicable means of preventing uneven wear would constitute a distinct advance.

---

* *Engineering and Mining Journal*, Jan. 10, 1910, p. 221.

† See this *Journal*, Vol. vii., March, 1907, p. 293.

It is somewhat remarkable that in spite of recent advances by other companies in the weight of stamps the Luipaards Vlei mill which, as mentioned in the paper, was the first of the modern heavy stamp mills erected on the Rand, still holds the record for high stamp duty, that for January, 1910, being 9·53 tons, as against the 9·10 tons duty of the next on the list, the West Rand Consolidated Mines.

From a dialectic standpoint, an adequate reply to Mr. E. J. Way's ably-worded criticism of the heavy stamp-tube-mill policy would require more time and skill in debate than the author can claim. The practical man, however, who judges by actual facts, may consider the following circumstance a sufficient proof of the soundness of the views expressed in the paper. Within the last month the Chairman of the Benoni Consolidated Gold Mines,* doubtless upon the recommendation of Mr. E. J. Way as the Consulting Engineer to the Company, announced that their new crushing plant would consist of heavier stamps (2,000 lb.) than any yet in operation here, with a higher ratio of tube-mills to stamps (1 to 13·75) than has yet been adopted in regular practice by any producing Company upon the Witwatersrand.

The author has recently carried out at the mill of the Simmer East, Ltd., one of the Consolidated Goldfields sub-companies, the following series of trials under the auspices of the Mines Trials Committee to further develop certain points referred to in the paper.

Five stamps with long heads and short stems were extemporised in August last by joining two 24 in. heads together by means of stout pins, and were operated in a mortar-box with the front cut out and replaced by a thin detachable steel plate, as shown in the illustration. A wood and canvas housing prevented splash above the mortar-box, through which the heads projected. This design of stamp was on the lines of the early experiments carried out at the Knights Deep of increasing the weight of stamps by interposing a false shoe between the head and the true shoe, although the advantages of such a design were not then realised. The results of the present trials were quite satisfactory, apart from occasional breakages of the connecting pins, which would not of course occur with solid 48 in. heads. The open-fronted mortar-box facilitated access to the head when required, and the long head combined with short stem lowered the centre of gravity of the stamp to an appreciable and desirable extent as compared with the usual type. Whilst the stamp in question was of the heavy type, obviously the design is applicable to stamps of any weight, and has now been adopted for the new City Deep mill. The following are the relative weights of the different parts of the stamps, illustrated in the accompanying photographs, which are still running regularly.

Stem 13 ft. 6 in. long × 4 in. diam.	= 577 lb.	33·6%
Tappet (3 gib) 20in. × 9 in. diam.	= 235 lb.	13·7%
Head 48 in. × 9 in. diam. ...	= 754 lb.	43·9%
Shoe 9 in. dia. (half worn)	= 152 lb.	8·8%
	1,718 lb.	100·0%

In a battery designed on the above lines the guides could be brought somewhat closer together by lowering the upper guide and cam-shaft so as to still further shorten the stem, and the height of the open-fronted mortar-box could be much reduced as illustrated. The following are the percentages by weight which the various parts constitute of the total weight of the stamp illustrated with new shoe:—

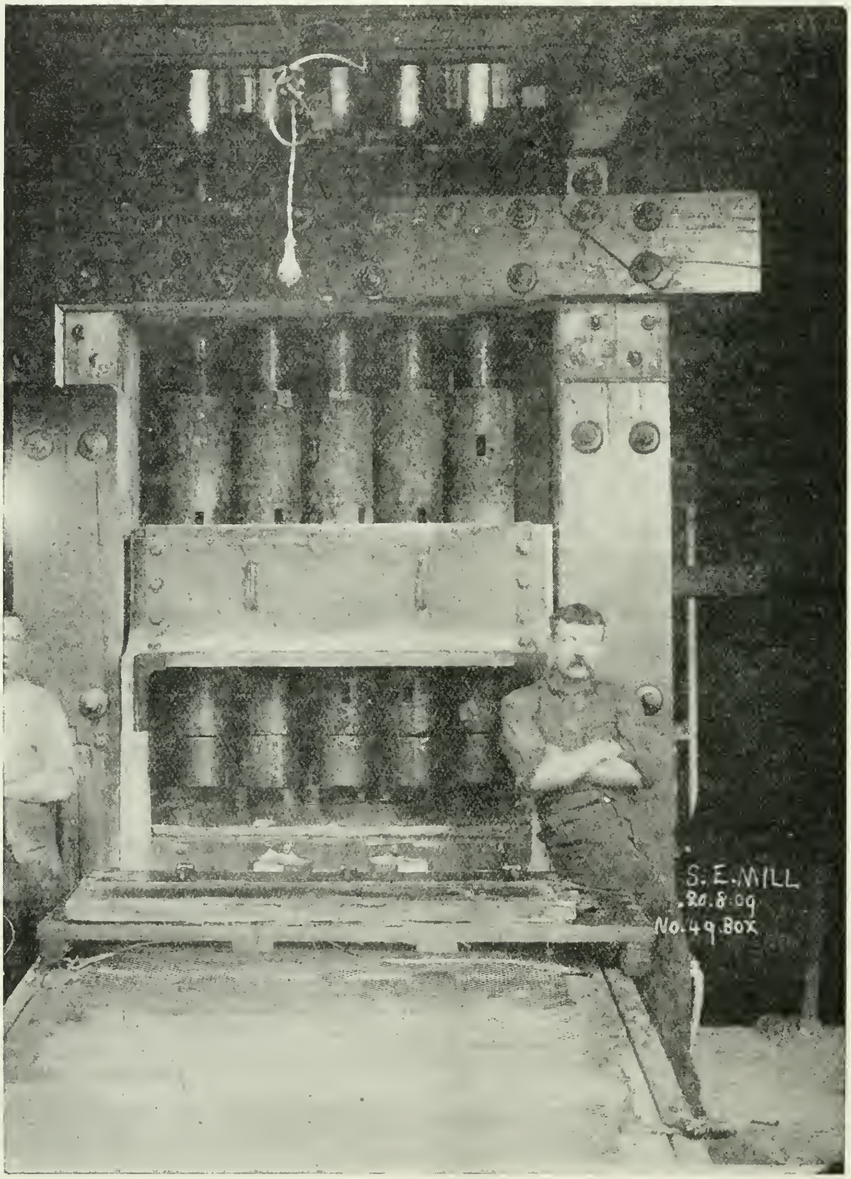
Stem	27·65%
Tappet	12·60%
Head	45·65%
Shoe	14·10%
	100·00%

The appreciable extent to which the centre of gravity is lowered by means of the long head and short stem, as compared with its position with the usual design of stamp, will be noted as tending to maintain the stamp in a vertical position and thus reduce friction and wear of guides.

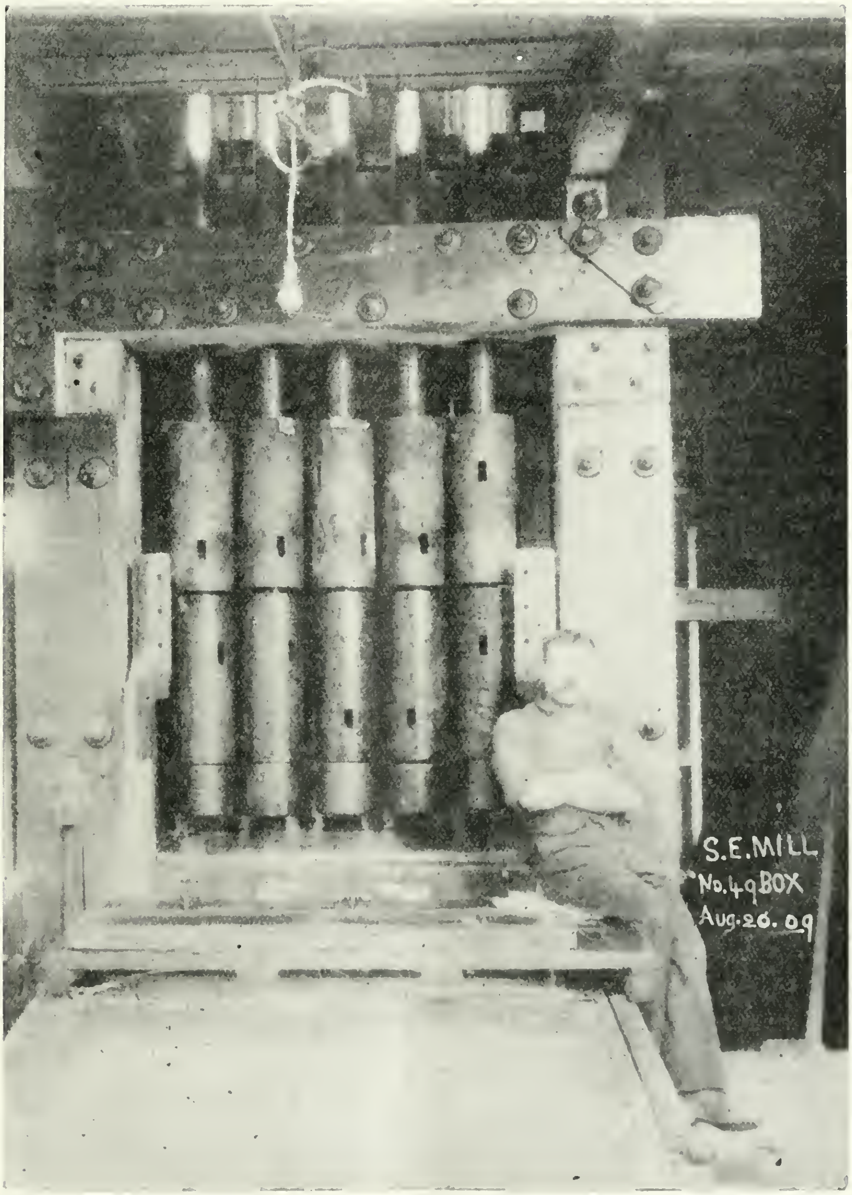
A still more important investigation, conducted with every care in August last in the Simmer East mill under the supervision of Mr. W. R. Dowling, the reduction works manager, concerned the maximum size of particle which could be efficiently pulverised in the usual tube-mill 5½ ft. by 22 ft., or in other words the coarsest battery screening practicable under these conditions. Briefly, the maximum efficient aperture was found to be 0·27 in., equivalent to battery screening of 9 holes per sq. in., and besides obtaining very high stamp duties, the actual tonnage per day of the desired - 90 mesh product of stamps and tube-mills was further found to be much greater by using such coarse screening than with the finer battery screens in common use. The fact that pulp from a coarse battery screen should contain more tons of fine product per stamp than that from a fine screen would hardly be expected, but though the percentage of fine product in the former pulp is low, yet this is more than offset by the high stamp duty.

---

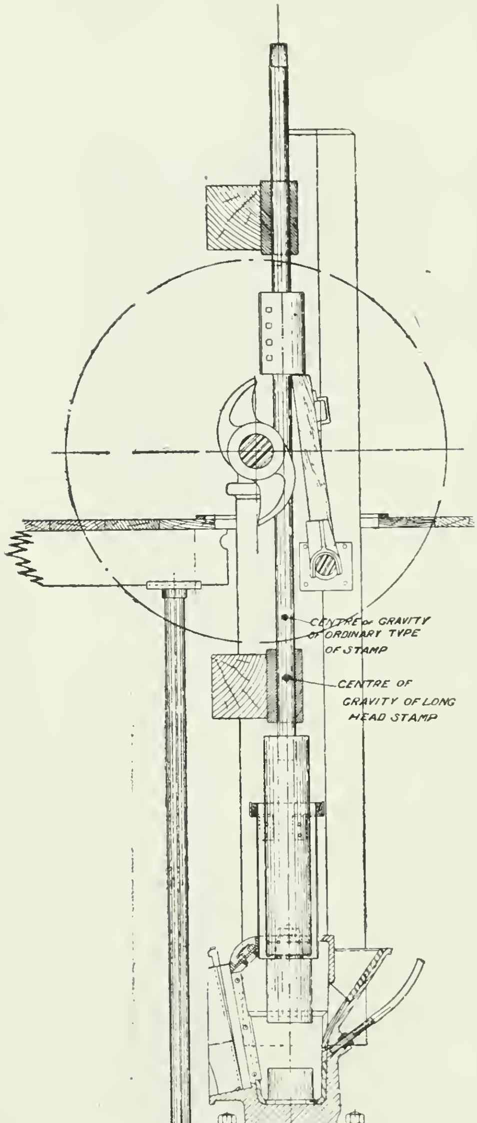
* See Annual Report of this Company, *South African Mining Journal*, March 26, 1910.



Stamps with extemporised long heads operating in mortar-box with detachable front.



Stamps with extemporised long heads operating in mortar-box
with detachable front removed.


CENTRE OF GRAVITY OF ORDINARY TYPE OF STAMP
CENTRE OF GRAVITY OF LONG HEAD STAMP

Stamp with long head and short stem operating in mortar-box with open front.

The general improvement in crushing efficiency is really due to applying the principle of promptly removing from the mortar-box all particles too small for the stamp to readily reduce further, and by feeding the coarser of these to the tube-mill providing the latter with material which is within the limits of its most efficient operation.

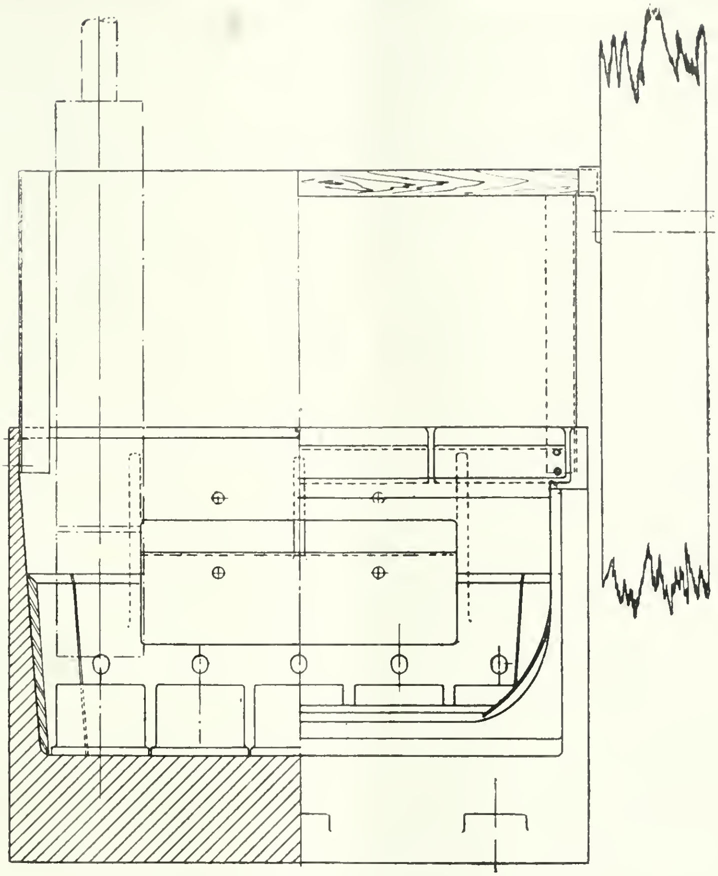
The paper to the discussion upon which this constitutes a reply referred to the saving in capital and operating cost due to the use of heavy gravitation stamps, and likewise the limit of weight imposed by the ordinary method of operation. By coarse crushing the duty of stamps, of a weight which has been satisfactorily operated for several years, is increased far more than is likely to be possible with fine screening, and any practicable weight of ordinary cam-lifted stamp. At the same time under these conditions the ultimate fine product required may be readily obtained by the addition of crushing units in the shape of tube-mills, which cost less to operate than the equivalent in stamps. The capital expenditure is likewise reduced by the addition of such crushing units as tube-mills at half the cost or less than the equivalent even in heavy stamps, and the cost of crushing is reduced in inverse ratio to the raising of the crushing efficiency. The extent to which the capital cost of large modern Rand reduction plants, referred to by Mr. W. F. Wilkinson, has been reduced in recent years is as yet hardly realised, but may be gauged from the fact that, owing to the employment of heavy stamps, coarse stamp milling, sand filter tables, large units throughout and electric power, the Simmer Deep-Jupiter reduction plant when extended is estimated to cost* £107 per ton of ore treated per 24 hours, or well under one-third of the cost of smaller first-class reduction plants erected in the former years.† The extension of crushing capacity of the Simmer Deep-Jupiter plant by 48,000 tons monthly will be effected by means of ten tube-mills ($5\frac{1}{2}$ ft. × 22 ft.) so as to obtain an average 14 ton stamp duty through screening of about 30 meshes per sq. in., instead of by the addition of four tube-mills and 200 stamps with an $8\frac{1}{2}$-ton duty on the lines on which the existing plant was designed. The saving of very large sums in capital expenditure on reduction plant is of especial importance with poor ore, as this consideration may prove the factor which determines whether a low-grade mine shall be brought from the developing to the producing stage or not. In other words, with a given amount of capital available, a large plant for poor ore can now be erected at the same cost as a small plant in former years, which latter might not treat sufficient tonnage to yield any profit at all on such low-grade ore. In any case, reduction of capital cost relieves the subsequent operating cost of a corresponding burden for redemption and interest on capital sunk in plant.

The development of coarse crushing in stamp mills has likewise been carried out in other

---

* See Report of Annual Meeting of Simmer Deep, Ltd., *S. A. Mining Journal*, March 26, 1910.

† According to Ross Browne (*Journal S.A. Association of Engineers*, Vol. xii., May, 1907), the average cost of nine Rand Mines plants, including crushing, cyaniding and power sections was £356 per ton of ore milled per 24 hours.



Section and elevation of mortar-box with open front.

mining districts, although the maximum limit of screen aperture probably varies under different conditions, according to the hardness or otherwise of the ore and the dimensions of the tube-mills and pebbles used. Thus, among other coarse crushing plants in Rhodesia, the Giant Mine with with a large proportion of soft ore has used screens up to $\frac{1}{2}$ in aperture*; the Komata Reefs in New Zealand use a battery screen of three holes per linear inch with subsequent tube-milling†; and Mr. Chas. Butters' various mills at Copala and elsewhere use screening of 4 holes per linear inch, that of the Divisadero Co. (Salvador) achieving with this mesh a stamp duty of over 9 tons. and the aid of tube-mills a final pulp of 55% of − 80 mesh sand and 45% of − 200 slime.

In the Simmer East investigations some trials of 50 stamps with coarse battery screening (16 and 36 holes per sq. in.) and the remaining 80 stamps with fine screening (1,000 to 1,600 holes per sq. in.) were first carried out with encouraging results, though the scouring of the amalgamated plates with this coarse product was found to be excessive and in subsequent trials the plates were covered with linoleum. The cyaniding results of the trials were normal and in proportion to the ultimate fineness of the pulp.

In all trials the usual tube-mill circuit † was in operation whereby the coarse portion of the pulp after elevation was separated by cone classifiers for tube milling. The tube-mill outflow, after passing over shaking amalgamated plates, rejoined the pulp from the batteries for re-elevation and separation of any coarse particles, which again passed through the tube-mills, whilst the fine material overflowing the classifiers gravitated to the cyanide plant.

In the subsequent trials, only stamps with coarse screening were used, and the tonnage of ordinary mill feed of ore crushed was calculated from numerous measurements of ore supplied to the batteries from a bin, which was partitioned off for the purpose, and the surface of the ore levelled before and after each run, 20 cub. ft. of ore being taken to the ton. The duties thus found were checked where practicable and with substanstantially the same results, by the tonnage of sand and slime collected during each run. 50 stamps and 3 tube-mills (5½ ft. × 22 ft.) were available for the trials and were run in varying ratios to each other as stated.

---

* See this *Journal*, Vol. ix.; Nov., 1908, p. 152.

† *New Zealand Mines Record*, 16th April 1909; also this *Journal*, Oct., 1909, p. 151; and Feb. 1910, p. 297.

† G. O. Smart, in this *Journal*, Vol. x., Feb. 1910, p. 282.

Trials were made with battery screening of 4 holes per sq. in., but it was found that with this product the tube-mills discharged very coarse material, which accumulated in the tube-mill circuit, many particles in the pulp being apparently too large to be shattered by the falling pebbles, and being merely rounded by abrasion during their passage through the tube-mill.

Regular and frequent sampling of battery pulp during each run was carried out to obtain samples for grading analyses, and to determine the ratio of water to solids. The grading of the final pulp was calculated back from the grading of the sand charges, in accordance with the percentage by weight they constituted of the ore milled. The production of – 90 mesh product per stamp was determined from the percentage of – 90 mesh in the battery pulp and the stamp duty, whilst the production of – 90 mesh product per tube-mill was calculated from the tonnage of ore milled during each run, divided by the number of tube-mills in operation, and multiplied by the difference between the percentage of – 90 product in the battery pulp and the final pulp.

The average running weight of the stamps in all trials was 1,400 lb., with 8 in. set height of drop at 98 drops per minute, and 2 in. height of discharge, whilst all mortar-boxes were fitted with back water feed as well as the usual supply from above. The utility of the back water feed in coarse crushing may be attributed to its washing towards the screen particles which are fine enough to pass through, though too coarse to readily become suspended in the splashed pulp. The stamp duty of the 16 mesh and 36 mesh trials, and consequently the tons of – 90 mesh per stamp, is probably slightly high in proportion to the grading analysis of the screen pulp, as the first mentioned were determined for single battery trials with a higher ratio of water to ore than that of the longer runs detailed in the table.

For the purposes of the trials detailed, special wooden launders with 10% grade were installed and served satisfactorily for all the varieties of pulp produced, including that from screening of 4 holes per sq. in. With screening not coarser than 9 holes per sq. in., somewhat less grade should serve, and in the Simmer Deep mill extension cement launders of U cross-section and wide-sweeping bends will be used, having 7·5% grade on the straight, and 9% round bends or where the volume of pulp is small.

A large number of single 5-stamp battery runs were made to confirm or elucidate certain points, but these have not been detailed as the information they afforded is equally well illustrated in the tabular statement.

The Simmer Deep trials, made somewhat later under the careful supervision of Mr. J. E. Thomas, the reduction works manager, were on similar lines and yielded similar results. The stamp weight was somewhat greater, as shown, and the actual height of drop 8½ in., at 98 per minute.

In the case of both plants the average results of a month's run, under ordinary conditions, are given in parallel as a basis for comparison.

The following table, abstracted from the foregoing, clearly illustrates the increase in efficiency due to coarse crushing in the battery, as compared with the usual finer screening:—

Even making allowance for improved results owing to the special care taken by operators during the progress of a trial run as compared with routine work, yet the increase above shown,

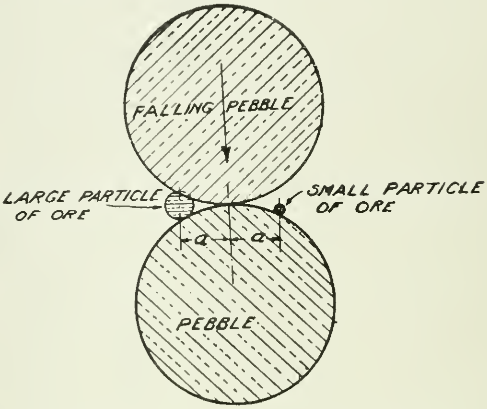
Company.	Number of Trials.	Battery Screening Meshes per sq. in.	Duty per Nominal Stamp.	Tons of – 90 Product per 24 Hours.		
				Per Nominal Stamp.	Per Tube-Mill.	Per N.C.U. of Crushing Plant.
Simmer East	2	4	15·68	3·85	—	—
Simmer East	6	9	13·33	3·69	132·8	4·08
Simmer East	1	16	12·34	3·84	—	—
Simmer Deep	1	30	12·74	3·83	137·5	4·25
Simmer East and Simmer Deep ...	2	36	11·00	3·83	126·1	4·01
Average of Coarse Crushing ...	12	19	13·02	3·81	132·1	4·11
Simmer East	Nov., /09	200	7·24	3·15	106·1	3·30
Simmer Deep	Jan., 1910	600	6·49	3·36	107·7	3·44
Average of Fine Crushing ...	—	400	6·86	3·25	106·9	3·37

	SIMMER & JACK EAST, LTD.														SIMMER DEEP-JUPITER.			
	August and September, 1909.													Nov., 1909.	Nov. and Dec., 1909			Jan., 1910.
Number of trial	1	2	3	4	5	6	Av'ge of 3,4,5,6	7	8	Av'ge of all 9 mesh	9	10	Av'ge of 9 & 10	Whole Mill.	11	12	Av'ge of 11 & 12	Whole Mill.
Tons crushed during trial	570	250	309	242	119	566	312	516	613	228	196	112	154	29,880	1,240	1,317	1,278	73, 194
Stamps:—																		
(1) Number	50	50	50	50	50	50	50	50	40		35	30		130	75	75	75	300
(2) Duty per 24 hours with running weight of:—															(1,640 lb.)	(1,580 lb.)	(1,610 lb.)	(1,600 lb.)
(*a*) 1,400 lb.	11·92	13·82	14·93	15·20	15·12	14·93	15·05	14·91	14·52	14·94	17·17	17·95	17·56	8·11	16·71	14·37	15·56	8·31
(*b*) 1,250 lb.	10·64	12·34	13·33	13·57	13·50	13·33	13·44	13·31	12·96	13·33	15·33	16·03	15·68	7·24	12·74	11·37	12·06	6·49
Screening:—																		
(1) Meshes per square inch.	36	16	9	9	9	9	9	9	9	9	4	4	4	200	30	36	33	600
(2) Aperture inch	·121	·197	·272	·272	·272	·272	·272	·272	·272	·272	·404	·404	·404	·053	·135	·119	·127	·028
(3) Per cent. discharge area	52·85	61·78	66·50	66·50	66·50	66·50	66·50	66·50	66·50	66·50	65·28	65·28	65·28	55·95	54·75	50·69	52·72	48·36
Ratio of water to one of ore	4·44	3·65	5·04	5·47	5·38	5·16	5·26	4·84	5·06	5·16	4·86	4·75	4·81	6·00	5·5	4·8	5·2	6·8
No. of tube-mills (5½ ft. × 22 ft.)			3	3	3	3	3	2	2					3	4	4	4	7
Per cent. nominal crushing units in:—																		
(1) Stamps			38·4	38·4	38·4	38·4	38·4	48·3	42·7	40·8				61·9	45·1	44·1	44·6	64·6
(2) Tube-mills			61·6	61·6	61·6	61·6	61·6	51·7	57·3	59·2				38·1	54·9	55·9	55·4	35·4
N.C.U. in stamp-mill per tube-mill ...			18·5	18·6	18·6	18·6	18·6	28·0	22·4	20·8				48·53	24·6	23·7	24·2	54·9
Screen pulp grading:—																		
+ 6 (·121 in)		5·5	11·5	13·0	8·6	12·9	11·5	12·0	11·1	11·5	18·8	19·0	18·9					
+10 (·080 in.)	3·2	8·4	8·1	7·9	8·2	8·9	8·3	9·4	9·2	8·6	7·2	9·6	8·4					
+20 (·034 in.)	19·5	18·7	18·0	18·4	19·0	18·7	18·5	18·6	19·2	18·7	18·0	16·7	17·4					
+40 (·015 in.)	19·1	17·1	16·9	16·1	17·4	16·5	16·7	16·0	16·3	16·5	15·4	15·1	15·2					
+60 (·010 in.)	10·1	8·8	7·9	7·6	8·0	8·4	8·0	8·2	7·8	8·0	7·9	7·0	7·5	42·5	57·2	53·2	55·2	31·7
+90 (·006 in.)	11·4	10·4	9·7	9·3	10·0	8·4	9·4	8·2	8·7	9·0	7·8	8·5	8·1	14·0	12·7	13·7	13·2	16·5
−90 (·006 in.)	36·7	31·1	27·9	27·7	28·8	26·2	27·6	27·6	27·7	27·7	24·9	24·1	24·5	43·5	30·1	33·1	31·6	51·8
Final pulp grading:—																		
+60 (·010 in.)			4·5	2·8	6·0	6·4	4·9	11·9	5·8	6·2				4·8	5·0	2·5	3·7	5·8
+90 (·006 in.)			17.7	14·7	19·2	19·9	17·9	17·7	18·3	17·9				21·0	21·0	17·6	19·3	15·0
−90 (·006 in.)			77·8	82·5	74·8	73·7	77·2	70·4	75·9	75·9				74·2	74·0	79·9	77·0	79·2
Tons −90 product per 24 hours:—															(1,640 lb.)	(1,580 lb.)	(1,610 lb.)	(1,600 lb.)
(1) Per actual stamp (1,400 lb.) ...	4·38	4·30	4·17	4·21	4·35	3·91	4·16	4·11	4·02	4·13	4·28	4·33	4·31	3·53	5·03	4·76	4·90	4·30
(2) Per nominal stamp (1,250 lb.) ...	3·91	3·84	3·72	3·76	3·88	3·49	3·71	3·67	3·59	3·69	3·82	3·87	3·85	3·15	3·83	3·76	3·80	3·36
(3) Per tube-mill (5½ ft. × 22 ft.) ...			124·1	138·8	116·1	118·0	124·3	159·7	139·8	132·8				106·1	137·5	126·1	131·8	107·7
(4) Per nominal crushing unit of crushing plant			3·98	4·29	3·88	3·77	3·98	4·53	4·02	4·08				3·30	4·25	4·01	4·13	3·44

in tons of – 90 product, of more than 20% per crushing unit of plant with coarse battery screening is very great. In consequence of these results the Benoni Consolidated, as already mentioned, have decided to adopt very heavy stamps with a high tube-mill ratio, whilst the largest gold producer in the world, the East Rand Proprietary Mines, having lately confirmed the advantages of coarse crushing in the battery by independent trials, now proposes to shut down 380 stamps and to crush more ore with the remaining 440 stamps, plus extra tube-mills, than with the whole of their 820 stamps at present in operation.*

It will be observed that the ratio of water to ore in coarse battery pulp is appreciably less than is usual with finer screens. The absolute amount, however, is much greater, as 5·5 tons of water with a 15-ton duty is equivalent to 412·5 tons of water entering a mortax-box per 24 hours, whilst a 6·5 water ratio with a 7-ton duty only equals 227·5 tons of water per mortar-box. The large volume of water fed into the mortar-box naturally facilitates the discharge of fine particles, whilst the ready discharge through the large screen apertures serves the same purpose, and the large percentage of discharge area prevents the level of the pulp in the box rising unduly. This ready discharge of particles fine enough to pass the screen serves to explain the influence which the water ratio has upon stamp duty with coarse screens, and to some extent either finer screens and more water, or coarser screens and less water, can be used to crush the same tonnage.

The increased efficiency of tube-milling, as measured in tons of – 90 mesh product, with coarse battery screening and a consequent coarse tube-mill feed, may be explained as due to the fact that such coarse particles provide work to be done by the falling pebbles throughout the full length of the mill, whereas with a finer feed many particles are early reduced to the required size and hence the pebbles near the outlet of the mill are not supplied with sufficient coarse particles to secure their full crushing efficiency. The following sketch may serve to explain in some measure why coarse ore particles are more readily crushed by tube-mill pebbles than finer particles, since the latter must be almost exactly at the point of contact of two pebbles to receive the impact of the falling pebble, whereas large particles distributed over a much larger area are liable to be shattered. The problem is in essence much the same as that due to the uneven wear of shoes and dies or of roll faces, and indeed constitutes a practical reason why the difficulty of ore reduction increases with the fineness of the final product sought, through the difficulty of ensuring that a particle to be shattered is in simultaneous contact with two opposite crushing faces.



An effect of coarse crushing, which of course results from say doubling the stamp duty, is the need for larger bins to contain the usual two to three days supply of ore for the batteries, and various methods of meeting this requirement have been proposed, as mentioned by Mr. Schmitt.

In designing a crushing plant, it is not advisable to consider the maximum stamp duty with 9-mesh battery screening as the average duty, since the supply of ore delivered to the mill is not always regular, so that at times a finer or coarser screen must be used. If the normal duty be taken as that with a 9-mesh screen no coarser screen and higher duty are practicable, on account of exceeding the limit of efficient tube-milling, whereas if sufficient stamps are erected to allow the average tonnage to be crushed through 16 or 25-mesh screens, variations in screening and duty can be made both above and below the average requirement.

Although amalgamation of banket ore on plates in the mill is practised with screening of 64 meshes per square inch, the scouring of the plates is considerable, and with coarser screening a worse effect is naturally produced. Assuming that plates in the mill are dispensed with, which course presents other advantages, amalgamation can be subsequently carried out in the tube-mill circuit and of the overflow of the tube-mill classifiers. This matter has been systematically investigated by continuous running for some months during the past year in several Randfontein mills by Mr. F. Maxwell, with results that show 90% of the amalgamable gold to be recovered on the amalgamated plates in the tube-mill circuit. The remaining 10% of the finest amalgamable gold, equivalent to say 6% of the total gold contents of the ore, passes with the tube-mill classifier overflow into the cyanide plant, where nearly all remains with the sand. Given good classification for tube-milling and efficient cyanide treatment of sand by cyaniding, it is not apparent that the presence in the final pulp of a small amount of fine metallic gold, which is readily and totally dissolved by cyanide solution, is sufficient to warrant the cost of installing and operating amalgamated plates for the classifier overflow. With the large ratio of tube-mills to stamps involved by coarse crushing, the number of tube-mill classifiers and their total settling area is of course far greater than with fine crushing in the battery, and the consequent use of a limited number of tube-mills to "finish off" the tailing pulp.

---

* See Annual Report of this Company, *S.A. Mining Journal*, March 26, 1910.

STOPE MEASUREMENTS.

(Read at May Meeting, 1909.)

By Otto S. Tonnesen (Member).

REPLY TO DISCUSSION.

Mr. O. S. Tonnesen (*Member*): With reference to the criticism on my paper on stope measuring I regret not having been able to reply earlier, owing to my absence on a holiday trip. I am glad to notice that, on the whole, the paper seems to have been favourably received, and that there are no vital criticisms of adverse nature.

Mr. A. A. Coaton agrees with me that "the transit method is necessarily slow on account of having to book two sets of angles for each sight," and his "chief objection to the use of the transit for stope measuring is at that stage of the operation where the horizontal fathomage is converted to inclined fathomage." But he claims that the question of setting up the transit rapidly and safely on steep inclines not exceeding 60° can be overcome by adopting the bolt stand, as used on the Wit Deep and Simmer Deep. The time it takes to superintend the 18 in. holes being correctly drilled must, of course, also be taken into consideration. (Mr. B. Travers Solly goes a step further, using two 18 in. holes.) Admitting that this is an improvement on the ordinary method of setting up a theodolite, I still maintain that "in a steep stope the setting up of a theodolite is a very lengthy and difficult operation," and see no reason for employing a slower and difficult operation, when a quicker and easy method can be substituted with accurate result. I would also point out as a disadvantage in using the transit and bolt stand, that the bolt, when used for future reference and set up, is likely to come out of its true centre, owing to rock falling and being shovelled. Although this boltstand has been used in a couple of mines for several years it has not been generally adopted on the Rand, or won many friends and advocates among mine surveyors.

Mr. Coaton's suggestion for the determination of the true dip of a stope is correct and smart, but it entails still more observations and office work and is not likely to be used by the majority of mine surveyors, but only by the most conscientious of them, and is, therefore, not justified unless no other correct method be available.

Mr. Coaton refers to a sentence in my paper, wherein I state that "the true error is greater than shown in above table owing to the difference in length between the face-line and straight-line AB." What was in my mind was, that the irregularity of the face-line will cause further complications and inaccuracies in the measurements which would increase the error. But as this would be the result of offset errors and personal errors rather than sagging tape, I agree that the sentence has got a misleading form and had better been left out. Admitting my proofs of the danger of the tape triangulation system, arising from a sagging tape, Mr. Coaton gives a formula for obviating this error by measuring the vertical sag. This method may, to some extent, lessen the effect of sag error, but again, as above, this increases the observations and calculations and sources of errors, and would only be justified if no better method were available. The method is also unreliable in practice, for the sag is not a constant one during the offsetting process, varying according to the tension the boys are putting on to the tape, and with many other conditions. And finally, how is Mr. Coaton to measure the vertical sag? Simply by measuring the vertical distance from the hanging to the middle of the tape. Or is he going to sight in a candle from the ends of his tape and then measure the vertical distance from this candle to the middle of the tape? The last method he will find in most cases impossible, when the tapes are held in the hanging, which is never a true plane, and in every case utterly impractical and slow. The first method is absolutely wrong and misleading, for the reef plane is never a true plane but somewhat convex or concave, and local irregularity of hanging wall would enlarge or minimise the measurements of the vertical sag. These several reasons render such determination of sag errors incorrect and impractical. Mr. Coaton's formula is nevertheless aiming in the right direction. Now I consider that the best method of getting rid of the sag errors is to bring them down to a minimum by shortening our lines so as not to greatly exceed 100 ft. My critics find my example on p. 377, 1909, of a 316 ft. long tape line rather exaggerated, which I, to some extent, grant. My aim was to investigate and distinguish the different sources of errors and to call for discussion in order to further clear the matter and therefore I necessarily had to select pointed examples. But there is a clear method in my argument: I measure a stope of 250 ft. back, dipping 50% (surely a very common and ordinary case). I measure it in two ways: (1) by a straight offsetting, line from top to bottom and, (2) by intersecting intermediate pegs (Fig. IV., p. 378, 1909). The first method distinguishes sag errors, the second gives a combination of sag errors and intersection errors.

Mr. Coaton proposes to avoid projection errors by avoiding the use of drive pegs. There are, however, a multitude of cases, particularly in steep stopes, where it is impossible in practice to neglect the usefulness of drive pegs. It is therefore of importance that we know the cause and effect of projection error in order to beware and keep them within allowable limits. There is no reasonable objection to using drive pegs, if they are on reef and the sighting candle is so suspended as to be approximately on the plane measured.

Mr. Coaton has never found any difficulty in judging the right angle, where the offsets do not exceed 30 ft. I shall refer him to Mr. Travers Solly's remark on p. 213, 1910, where my opinion is backed up.

I quite endorse Mr. Coaton's method of protractor plotting. It is good and quick. I am also thankful for his investigations as to the effect of the plane of the instrument not being held on the plane of the reef. However, my instrument is so constructed that when the back and foresight points coincide in the prisms then the plane of the instrument must necessarily be in the same plane as the three survey points. Nevertheless, Mr. Coaton's calculation holds good, viz., when the reef changes in dip.

I wish to thank Mr. Coaton for his suggestion for obtaining a distinct light, and generally for his valuable contribution to the discussion.

I also wish to thank Mr. B. Travers Solly and Mr. Tom Johnson for their able contributions. I have never thought nor said that proper and painstaking stope measurements are not done on the Rand. It is bad enough that mistakes and errors often do occur and are found out, and it is our aim to gain the confidence of all concerned, not by deceit and bluff, but by straight and correct methods worthy of the support of this institution, "whose proceedings are noted all over the world."

The main aim of my investigations has been to get to the bottom of the truth in this manner. I investigate the cause of errors. (This was not an easy task, and one not previously seriously attempted.) These errors may not always occur, though when they do they sometimes compensate one another. But more than often they must occur and even accumulate. Having investigated and pointed out the different sources of errors, I introduce means of avoiding them, or keeping them within reasonable limits. It is not to the point to instance cases where a miner disputing measurements has been found wrong. It happens often, and so does the contrary case. I have also some slight experience with miners and grumbling and I know as well as Mr. Tom Johnson that some of them will grumble for anything, but that is one of the reasons why I have gone to the trouble of finding a method, which does away with our own doubts and enables a quick and reliable check. Mr. Johnson asks, did I ever think of how the idea of stretched strings could be used. I did, and so did our ancestors. It was one of the first primitive means of estimating an angle. It is unreliable, since it is dependent on the sag and conditions of the lines and on a rough estimation of the right plane. It measures the angles between the tangents and not the chords. I think Mr. Tom Johnson has no right to slight Mr. Saner, myself, nor anyone else for being conscientious in our work. Bluff is a gift not possessed by all of us.

Dr. Moir's hairsplitting attempt at criticism has been so thoroughly dealt with by both Mr. Coaton and Mr. K. Tonnesen, that it is hardly necessary to further refer to it. His own blunders render his results essentially wrong. In his latest reply Dr. Moir attempts to conceal his own retreat by pointing out an error in a formula brought forward by Mr. Coaton, and to further prove how extremely wrong it is to use the parabolic curve. He instances that even a circular curve may give accurate results enough. To this I would only say, that it would have been quite justified for this purpose to treat the tape curve as a circular, if this had given easier solution. As it is, however, the parabolic area besides being a closer approximation is ever so much simpler solution as the "area of a parabolic segment equals two-thirds of the circumscribed parallelogram." This is a perfectly clear and well known definition, and I do not see Dr. Moir's justification in charging me with obscurity on a point which he must have misunderstood. However, I agree that some of my calculations could have been still more comprehensively explained, but I chose, what I considered the happy medium, in order not to make the paper too long and tiresome. Finally, I wish to thank Dr. Moir for his criticism, which I believe, was not made so much in order to seriously condemn my formulae as in a desire to break the ice for further discussion.

I thank the gentlemen who have taken part in the discussion for their keen interest and also for the many flattering and encouraging remarks.

Our endeavours, I am confident, will bear the desired fruits of improving and simplifying stope measuring. We seem already pretty well unanimous in avoiding long tape lines in order to minimise sag errors, and also long offsets in order to lessen the personal errors, and when you now get the opportunity of trying the stereometer, I think we shall also agree, that it will give us the best and easiest means of obtaining these points and keeping the errors within control.

Tonnesen's Stereometer. — Construction and Manipulation.—The instrument consists of two main parts, the alidade or moveable part and the limb or immoveable part. It carries two large brightly polished prisms, the top prism A fastened to the alidade and turning with it, and the bottom prism B fastened to the limb. The alidade is turned by the pivothead or "bigmotionscrew" C and the "slow motionscrew" D. D can be pushed slightly to the right and comes then out of action at the same time releasing C. The stereometer can be used on a light stand or simply by hand. It is furnished with two spirit levels E and F for adjusting it to the horizontal plane. The level E is so placed that when looking straight into the bottom prism its reflection can be seen immediately above the other bubble F. The instrument can thus be horizontally adjusted, and the survey points bisected at one glance. The plate surfaces immediately over and underlying the prisms are brightly polished so as to serve as view finders or reflectors, whereby an image seen in one of the reflectors can be brought into view in the corresponding prism by a slight twist of the hand. This latest type of the stereometer can be read very accurately, the smallest graduation on the micrometer screw being 20 seconds. Such fine observations are, of course, only justified when a stand is employed or for fairly long sights, when the utmost care and accuracy is observed in centreing and bisecting and for distance measuring. For stope measuring it is accurate enough to read to, say, 10 minutes. J is aluminium plate, which is run into slits in the limb plate, and, when in a position, it is clamped by a screw. It is for direct plotting of angles and serves the same purpose as the drawing paper on a plane table. Therefore the edge of the alidade is bevelled to enable the angles to be struck off. The plate can easily be detached and replaced by a new one, and a sufficient number of these should be kept in stock. When finished with they can easily be cleaned, and are then again ready for use. It is sometimes particularly in stope measuring awkward to read the angle. It should then be drawn on the plate and also a number of reference. Even when reading, it is a safe plan to mark the angle as well. It takes very little time and furnishes a most effectful check on the correctness of the readings. The stereometer is not restricted to measuring horizontal angles, but may be used for measuring the angle between any points in the space. The plane of the instrument must be held in the plane of the points observed.

The use of the instrument is briefly as follows: An angle between any three points, "back-sight" point, "set-up" point and "fore-sight" point, may be found by turning the alidade until images of the "back" and "fore-sight" points coincide at the contact of the two prisms. The angle between the prisms is then equal to half the angle between the points, but the instrument is so graduated that the correct angle is read directly, 1° of arc being marked as 2° and so on. The stereometer should not be handled too close to the eye, but some distance away, strained observations being quite unnecessary. The main point is to hold the plane of the instrument in the plane of the three points observed, as otherwise the points cannot bisect at the contact of the prisms. Before taking an observation it is a good plan to judge this plane as near as possible, and also to make a rough guess of the angle, and set the alidade to it, so that there is little adjusting necessary when taking the sight. The stereometer is generally held square in front of the observer, who takes such position that he has the "back-sight" point about "eyes right.' If the point does not appear in the bottom prism at once, the instrument should be slightly turned in the hand until it appears. When observing angles near to 90° the observer should move his head sideways so that it is not reflected in the prism at the same time turning the instrument in such a way that the "back-sight" point travels from "eyes right" towards "eyes front."

When Stope Measuring.—Stopes are measured with the stereometer by radiation or the ordinary theodolite method, applied to the inclined plane of the stope (see p. 384, 1909), by traversing and offsetting directly on the inclined plane or by combination of both principles.

Traverse Method.—This method is tape triangulation extended in order to avoid accumulation of errors and to overcome practical difficulties caused by pillars and stulls and difficult climbing. When using this method two points must be known, one in the top and one in the bottom of the stope. A traverse is run between these two known points as close to the face as practicable, using the one known point as the first back-sight and the other as the last fore-sight. This traverse is then plotted on a tracing paper, the two extreme points of the traverse joined and adjusted on to the two given points on the stope plan and the intermediate traverse points pricked through.

The main framework thus obtained, the offsets are measured and plotted in the usual way. The theoretical effect of this method is the same as if an ideal mathematically straight line between the two known points had been available, and the stope measured and plotted from same. I do not measure the angles at the known points as is necessary in an ordinary drive traverse (working out from two given points to the unknown). In a stope one is working between, not from two known points. It is not only a waste of time to measure the angles at the known points, but it is also introducing errors caused by the unevenness of dip at top and bottom of stopes and exaggerating and accumulating of careless centering. When a great number of angles are observed they should be plotted by bearings and parallel ruler in order to avoid accumulation of protractor errors, the bearing being the last angle—180° and the new angle. Thus all the angle plotting is done from the same position of the protractor. The top and bottom pegs should always be kept up-to-date, traversing them in from the old top and bottom pegs. In a new stope, which has not yet reached the bottom level, the two known points would be one in the top of each stope. The bottom angle in the winze is often inaccessible for measurement, particularly in steep stopes, and can easily be determined by intersection of the two traverse lines. When the stopes are yet very small and good straight lines can be run through, giving well conditioned intersections, then the ordinary tape triangulation can be used with good results, or better still, the radiation method with the stereometer. But as soon as complications arise, long lines and offsets, bad intersections, pillars and stulls, and the old points in the stope become inaccessible, then an independent traverse should be run as above described, from top to bottom of the stope Some surveyors would prefer to put in new stope pegs and measure the angles before the time of stope measuring, while others would measure the angles when stope measuring. The survey lines should not be much longer than 100 ft. to avoid sag errors, but on the other hand, not too short nor too many, so as to avoid sudden change in dip, which longer lines would bridge over. Although an eccentricity of 1 in. even is of small account when the surveying and plotting is done as above described, the centring of the instrument should be done with care.

When stope measuring, the instrument should be held so that its pivot point nearly touches the point of the plumb bob. The distance between the pivot point and the centre of the instrument does not cause eccentricity as it projects right angled on the plane of the reef. The measurements of distances and offsets are most conveniently taken on the hanging wall or in very wide stopes on the footwall.

To save drilling holes , where permanent pegs are not necessary, I have made a patent pitch peg. It consists of a piece of leather or wood on to which is smelted a block of pitch. The pitch is heated over a candle and applied to the rock with a strong pressure. The rock ought to be fairly clean or brushed down to make the pitch stick well. These pegs can be used over again several times. When taking the peg down the place should be marked with tar or paint so as to know where to find the point if it should be required later on.

For underground survey it is handy to have well defined and quickly fixed lights to sight to. I have made some very simple plummet candles for this purpose. Fig 3 is for drive survey; Fig. 4 for measuring, where it is important that the light should not be mixed with other lights in the stope. In wide stopes, where hanging cannot be reached, the pegs or marks in the footwall are surveyed by reversing the plummet candle, hanging a weight on to the hook-end and plumbing over the mark.

Adjustments.—(1.) The reflecting surfaces of prisms must be perpendicular to the plane of the bottom plate. They are adjusted by the makers and not very liable to be thrown out of position.

(2.) To determine index error: Adjust the instrument to the horizontal plane. Bring the reflected image of a plumb-line, suspended in some distance, into view in the lower prism, turn the alidade until the reflected image of the same plumb-line appears in the top prism in such a position as to be a true unbroken continuation of the image in the first prism. At this position the vernier should read zero, and if it does not, the reading is the index error, and a correction should be made to the readings. The error may, however, be corrected by adjusting the vernier. The correction is equal to the error, and is, of course, of contrary sign. If the index stands on the right 0°, all the instrument readings are too small, hence the index correction is positive: but if it stands on the left of 0°, the readings are too large, and the correction is negative.

Other Applications.—Underground, the instrument is also of great value for rapid check of drive survey, setting out of works and rapid survey where the setting up of a theodolite is difficult or two slow. The candle flames used for sighting should then be suspended in about the same vertical distance from the rail. The true horizontal plane can more accurately be determined by making the spirit levels play in.

The stereometer can be used as a rangefinder by measuring the angle between two crossbars on a staff, the vertical distance between the two bars being known and a table made for converting angles into corresponding distances. For this purpose the stereometer is made to read as close as 20″. The stereometer is also provided with a small vertical spirit level, and two lines engraved in the bottom prism. It can therefore with a light stand be used as a level and clinometer When used on the surface for detail work the stereometer will prove of great value. The sighting points are then survey rods held vertically. When two rods are made to coincide in the two prisms as one line without any break, this is mostly sufficient indication that the instrument is held on the horizontal plane.

A ROTARY EXTRACTOR FOR PRECIOUS METALS FROM SOLUTIONS.

(*Read at December Meeting, 1909.*)

By W. D'Arcy Lloyd and Ernest T. Rand (Members).

REPLY TO DISCUSSION.

Mr. E. T. Rand (*Member*): We are sorry that there has been so little discussion on our paper. Our one critic, Mr. King, we thank for his remarks, to some of which we shall attempt to reply.

Mr. King remarks that he thinks the baffle plate is carried too far down. We do not think so; but this is a matter so easy of adjustment that further comment upon it is needless.

Our critic does not think it quite clear that there would be any saving by using zinc clippings in place of filiform zinc; we can best reply to this by referring him to a more careful perusal of our paper, where he will find that the acid treatment is merely a bath, not a total consumption of the zinc subjected to it, as in the case of filiform zinc. From experiment we found that the loss of zinc under the conditions mentioned in the paper is 1 8%, and approximately half or less of the zinc in the plant would be subjected to this bath but once a month. If we estimate that but 25% of the filiform zinc in use on the plant is the consumption per month, twelve times the original weight of the zinc will have been consumed in four years. On the other hand, some of the zinc clippings, though subjected to repeated acid baths and though losing in weight each time of immersion, would still be in use after 10 years It occurred to us when considering that part of the criticism relative to the extraction, where comparative figures are given, that great prominence is given to the results of a plant thoroughly set and well run. We beg again to point out in extenuation of our slightly inferior results that the best possible results were not to be expected seeing that our experimental machine was a crude one, and lacking in one of its greatest essentials to the very best work, *i.e.*, the settling tank beneath the rotor-casing.

As regards initial cost, it is difficult to give actual costs of construction; but where large quantities of solution are to be precipitated, we suggest that large units be used, handling from 750 to 1,000 tons or even more per 24 hours, and built in series with the individual machines divided by a longitudinal division plate. Built on this plan, we consider that the cost of construction would not exceed 35% that of the boxes required to handle the same quantity of solution. This extra initial cost is much more than counterbalanced by the saving of zinc, acid, etc., and most important of all, the almost total immunity from loss of gold by theft and careless handling. Another great advantage of the rotary extractor is its adaptability for handling large quantities of solution per unit, thus concentrating the gold in place of having it distributed over a large area, as in boxes. One machine, 7 ft. in diameter and 18 ft. long, would be ample to precipitate 1,000 tons per day of weak and slimes solutions. We may state further that the formation of that bugbear, white precipitate, we found but small, when experimenting with weak and slimes solutions. We concluded that this was due to the movement of the rotor. As the tendency now is to work with very weak cyanide solutions this machine would thus save the large consumption of zinc due to the formation of this precipitate.

As stated in our paper, the rotor in precipitating 225 tons of solution per 24 hours absorbed 1 h.p., whilst a 1,000 tons machine would require approximately 2½ h.p.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Significance of the Abel Heat Test of Guncotton and Nitroglycerine. — "This paper attempts to explain the significance of the results obtained when the Abel heat test is applied under standard conditions to purified ungelatinised guncotton, nitroglycerine, and mixtures of these, and to decide, if possible, among the various views held regarding the test, whether results obtained proceed from a decomposition of the explosive during test, from a trace of impurity, or from dissolved nitric peroxide. Methods were described for detecting the time of the heat test indication with the concentration and also with the quantity of nitric peroxide required to produce the tint. It was shown that, knowing these constants and the rate of decomposition of the explosive, its heat test could be calculated. For guncotton the data of Will connecting rate of decomposition and temperature were taken, and the calculated values were in good agreement with those observed. Evidence was adduced to show that the presence of neither dissolved nitric peroxide nor of any adventitious impurity was the cause of the reaction. For the decomposition of nitroglycerine the data of Robertson were taken, and the heat test was calculated in the same way. Again, a good agreement was obtained between calculated and observed results, when the nitroglycerine was so comminuted on inert fibres (silica wool) that great surface was exposed. When nitroglycerine was tested in the usual way, as a liquid, the test simply was a measure of the solution pressure of nitrogen peroxide in nitroglycerine, and this was made up of the initial quantity of nitric peroxide in solution and the quantity liberated by decomposition during the heat test. Experiments were quoted showing that nitroglycerine could frequently be freed from some of the initial dissolved nitric peroxide. The views advanced also threw light on the heat test of mixtures of guncotton and nitroglycerine ('pastes'), the low heat test of which was due to comminution of nitroglycerine on the fibres of guncotton. They showed that the subject must be looked at from a quantitative point of view, and that the question of exposed surface of explosive under test was all important. The methods developed were applicable for studying other indicators, and afforded a more solid foundation for choice than empirical comparisons."—R. Robertson and B. J. Smart, *Journal of the Society of Chemical Industry*, Feb. 15.—*Times Engineering Supplement*, Jan. 26, 1910. (J. A. W.)

Light Reactions at Low Temperatures.—"In a recent evening discourse at the Royal Institution Sir James Dewar showed some experiments to prove that chemical action is not entirely in abeyance at very low temperatures. Henri Moissan, he said, had held this view, and was justified when it was found that fluorine reacted violently with liquid hydrogen at a temperature of 20° abs. He himself had found that ozone was formed when ultra-violet radiation was passed into liquid air; and when oxygen solidified in liquid hydrogen was exposed to ultra-violet light, the solid oxygen, after the hydrogen had evaporated, also contained ozone. Such experiments indicated that chemical action did take place at low temperatures, since the transformation of oxygen into ozone involved the absorption of energy, and thus there was a low temperature chemistry which required to be worked out. In the course of the lecture a number of experiments were performed to illustrate the fact that various bodies, especially such as were of an albuminous character, which were not phosphorescent at ordinary temperatures, phosphoresced brightly when cooled in liquid air and exposed to ultra-violet light. Again, it was shown that ordinary phosphorescent bodies, such as sulphide of calcium, could absorb light-energy at low temperatures, though they did not exhibit luminosity until they had been warmed up to normal temperatures. Illustrations were also given of the destructive effects of ultra-violet radiations at low temperatures on the luminosity of living organisms such as *Bacillus phosphorescens*, which was essentially connected with their vital activities." — Sir James Dewar.—*Times Engineering Supplement*, Jan. 26, 1910. (J. A. W.)

METALLURGY.

The Computation of Crushing Efficiency of Fine Grinding Machines.—"Having exhausted all his arguments, the author charges engineers of the Rand as follows:

> 'Considering the very poor efficiency of tube mills, their high cost price and high working costs, a doubt may be allowed that many leading engineers have in their fancy for this mill so much in fashion on the Rand, exceeded the economical limit, at least in regard to new plants.'

This does not really concern the writer, but, in view of the fact that twelve months have passed since it was sent out into the world, and seeing that no one rose to contradict it, something should be done to dispel any doubt in the minds of our friends abroad. It was only the other day that Mr. W. F. Wilkinson spoke at a meeting of the Institution of Mining and Metallurgy, London, expressing doubt about the wisdom of installing so many tube mills.

When the paper, now under discussion, was written, it was generally considered a fair but by no means an ample allowance to provide one tube mill for every 50 to 75 stamps, depending on their weight, the underlying idea being that this combination could at that time reduce the ore from 2 in. cubes to the fineness required for satisfactory extraction with the least expenditure. It was then thought, and this still holds good, that the final pulp entering the cyanide plant should not contain more than 5% of +60 and possibly over 75% of −90. This was 12 months ago. To-day, owing to a considerable amount of further experience, the proportion of stamps to tube mills has been considerably modified, and, I trust, Mr. Stadler will not be horrified, it is considered that the right allowance of tube mills provides for one tube mill for every 15 to 20 stamps.

I shall endeavour to justify this by giving the result of a test carried out during last month at the Simmer Deep, Ltd., at the suggestion of Mr. Caldecott, following closely upon tests carried out some time ago on behalf of the Mines Trials Committee at the Simmer East, Ltd. Not being in the the confidence of that august body, I cannot give you the result then obtained:—

Coarse crushing experiment at the Simmer Deep, Ltd., 21st November, 1909.

Number of Stamps ... = 75.
Actual weight of stamps = 1,640 lbs. each.
Number of drops ... = 100.
Actual height of drop = 8¼ in.
Order of drop ... .. = 1, 3, 5, 2, 4.
Screen ... ... = 30 mesh per sq. in.
  Diameter of wire = 0·0482 in.
  Aperture ... ... = 0·1348 in.
  Discharge area ... = 54·75%
Ore crushed ... . = 1,240 tons.
Running time ... ... = 23·75 hours.
Duty per Stamp.. ... = 16·71 tons per 24 hours.
Ratio of water to ore = 5·5 to 1.
Grading of pulp leaving battery:—
  Percentage of +60 = 57·2%
  ,, −60+90 = 12·7%
  ,, −90 = 30·1%
Number of tube mills = 4.
Size of tube mills ... = 22ft. × 5½ft. dia. of shell
Speed of tube mills ... = 30 rev. per min.
Grading of pulp entering cyanide works:
  Percentage of +60 ... = 4·7%
  ,, −60+90 = 21·4%
  ,, −90 ... = 73·9%

Tonnage of (– 90) produced per tube mill per 24 hours ... ... = 137·2 tons.
Tonnage of (– 90) produced per stamp per 24 hours ... ... = 5·03 tons.
Tube mill equivalent to $\frac{137\cdot2}{5\cdot03}$ = 27 stamps.

It appears from the result of the above tests that 75 stamps + 4 tube mills crushed ore at the rate of $\frac{1{,}240 \times 24}{23\cdot75} = 1{,}253$ tons per 24 hours.

Ordinarily this plant is run with 75 stamps to one tube mill, and the amount of ore crushed to give a similar grading in the pulp entering the cyanide plant is at the rate of 8·2 tons per stamp per day or 615 tons for 75 stamps.

The cost of installing the 75 stamps was approximately £24,000, and after deducting £6,000* for ore bin, etc., £18,000, and that of the 4 tube mills was approximately £12,000. We have therefore, in the first case the fact that for a capital outlay of £30,000 we could handle 1,253 tons per day, or, to put it in another way, in order to crush one ton per day we had to spend about £23 18s. in capital outlay. In the second case the capital outlay was £18,000 + $\frac{£12{,}000}{4}$ = £21,000 to crush 615 tons per day, or £34 2s. per ton per day.

The interest on the capital can be taken at 7%, to which must be added 3°/o for redemption during the life of the mine. This on £30,000 amounts to £3,000 per annum, £250 per month, and say £8 6s. 8d. per day, or to 15·9d. per ton in the first case and to 22·7d. per ton in the second case.

To this must be added the charges for power, stores, maintenance, labour, water, light, etc. With regard to power, each stamp takes 4·2 B.H.P. at the motor, and each tube mill 100 B.H.P. on the same basis. Therefore the stamp battery, *i.e.*, 75 stamps, in both cases takes 315 B.H.P. and the tube mills 400 B.H.P. and 100 B.H.P. respectively. As both plants have the same load factor, the cost of power per K.W.-hour is identical, and amounts to about ·52d. per B.H.P. hour. Hence the cost of power per day in the first case is 715 × 24 × ·52 = 8,923·2d., or 7·12d. per ton; and in the second case 415 × 24 × ·52 = 5,179·2d. per day, or 8·42d. per ton.

The charges for labour, stores, maintenance, water, light, etc., can be taken from the monthly statements, and these are for the battery £30 for 75 stamps per day and £8 6s 8d. per tube mill per day. In the first case these would total £30 + £33 6s. 8d. = £63 6s. 8d. per day or $\frac{15{,}200}{1{,}253}$ = 12·13d. per ton; and in the second case £30 + £8 6s. 8d. = £38 6s. 8d. per day, or $\frac{9{,}200}{615}$ = 14·91d. per ton.

The figures obtained are summarised in the following table:—

CASE.	I.	II.	Difference in favour of I.
Type of Plant	75 Stamps of 1,640 lb. and 4 tube mills.	75 Stamps of 1,640 lb. and 1 tube mill.	
Capital Outlay } Total ...	£30,000	£21,000*	30°/o
Capital Outlay } per ton per day ...	£23 18s.	£34 2s.	30°/o
(1) Interest and Redemption per ton	15·90d.	22·7d.	299°/o
(2) Power @ ·52d. B.H.P.-hour per ton	7·12d.	8·42d.	15·5°/o
(3) Labour, Stores, etc., per ton ...	12·13d.	14·91d.	18 6°/o
(4) Total of 1, 2 & 3	35·15d.	46·03d.	23·7°/o
(5) Total of 2 & 3 ...	19·25d.	23·33d.	17·5°/o

Working costs on the Rand do not usually allow for interest and redemption of capital invested, but in the table line 4 represents the total charges, while line 5 gives what is usually called working costs.

However, since the money for building a plant must be obtained, and has to be paid for, the capital charges should in every case be taken into account. This I have done, and the result is that by providing one tube mill for every 17 stamps as compared with one tube mill for 75 stamps, a saving on capital account of 30°/o can be made, while the saving on the sum of capital charges and working costs amounts to 23·7°/o. Working costs, as usually understood, will be reduced by 17·5°/o.

For the benefit of those not familiar with Rand conditions, it should be stated that a final pulp containing not more than 5°/o of + 60 and not less than 75°/o of – 90 is quite satisfactory from the extraction point of view, although in some plants the final pulp has to be reduced to, say, only 2°/o of + 60 in order to get the best results. Naturally, in such a case the tube mill allowance must be further increased, thus confirming the above remarks.

I now leave it to Mr. Stadler to reconcile the figures with his statement that '*tube mills are very poor machines.*' This may be true from a 'mechanical' point of view, but nevertheless the fact remains that they do a portion of our work, and, as I have shown, a considerable portion of it, which, as far as we know, we cannot get done for the same expenditure by any other machine. A machine, or a method of working, may be intrinsically very inefficient, and still it is retained. Why? Anyone can supply the reply. Shall I have to remind you of the very poor thermal efficiency of our steam plants? How many per cent. of the energy stored in the coal do we get at the end of our mill line shafts, or by the way of compressed air down in our mines? I should say 10°/o and 5°/o respectively.

The author might have made a few qualifying remarks for his statement alluded to, and thus have saved one the trouble of writing for the purpose of defending the engineers on the Rand. I trust I have succeeded in showing that it is not fashion but dire necessity which compels the use of tube mills on these fields."—C. O. SCHMITT.—*Journal of the S. A. Association of Engineers.* (W. R. D.)

---

* This £6,000 is really more than the amount due to the ore bin, and the result is thus favourable to the stamp mill.

* When considering the capital cost of case II. it should be mentioned that the final pulp obtained is not really equal to that obtained in case I., and in order to equalise matters, an allowance for 1⅓ tube mill should be made, adding about £1,500, which would be in favour of case I.

Wet Assay of Tin Ores.—"The wet assaying of tin ores is just now in an interesting stage of development, recalling that of the wet assay of copper ores of twenty or thirty years ago. Wet methods are being used; those in control of mines are gaining experience of them, and their judgment, from which there is no appeal, will depend on results rather than arguments.

In the first place I venture to predict that the sale of dressed tin ore by wet assay will be subject to the control of at least two methods agreeing in their results. This opinion probably rests on the analogy of copper ores, which may be bought and sold on the results of either the 'iodide' or the 'electrolytic' assay. Again, the ore will be finely pulverised; there will be no talk of 80-mesh sieves, except for the purpose of grading. The necessity for this fineness, just as with telluride gold ores, arises from the needs of good sampling. Of the volumetric methods, two, namely, the titrations with permanganate and with bichromate of potash, are precluded because they demand solutions free from iron, and iron is everywhere. There remain the well-known titrations with ferric chloride and with iodine. These should survive as mutually confirmatory operations. But the iodine method will have the wider field of usefulness because the use of ferric chloride demands assay liquors free from colour and turbidity, and the iodine does not.

Using a volumetric method for tin, the assayer should recognise, without attaching over much importance to it, a quarter of a milligramme of tin. With a half-gramme charge of ore this quantity represents about one pound of tin per ton; consequently, when pounds of tin per ton are reported, a larger charge, say, two or three grammes, must be used. The assay-liquor in any case must contain the tin as stannous chloride in a fairly acid solution; such liquors must be protected from atmospheric oxygen as also from that dissolved in the water and reagents. The tin in the treated ore may be got into solution directly as stannous chloride ready for titration, or it may be got as stannic chloride from which stannous chloride may be prepared by a subsequent treatment. Other things being equal, it is obvious that the direct method is more advantageous.

In the direct method it is necessary to reduce the cassiterite in the ore to metallic tin and to dissolve the metal in hydrochloric acid. The difficulty in reducing finely pulverised cassiterite to metallic tin does not depend on any difficulty in selecting a suitable reducing agent. Heated coal gas will do it slowly; magnesium vapour acts with almost explosive speed; and zinc vapour with an intermediate rapidity. But each fails in most cases to bring the whole of the tin into a form soluble in hydrochloric acid. Even litharge heated in coal-gas can show a similarly incomplete reduction. My method is to mix the tin ore with zinc oxide, magnesia, or lime before treating with the reducing agent. This prevents fritting or fusion during the heating and yields a mass that breaks down completely when treated liberally with hydrochloric acid; with insufficient acid it yields a clot of hydrated silica, but this is easily avoided. My preference is for zinc vapour and zinc oxide, but clearly these are not essential to the process. One could mix the ore with magnesia and reduce in coal-gas, then dissolve in hydrochloric acid, and titrate with ferric chloride. This would be essentially the same method, though not as I prefer to work it. When zinc vapour is used as a reducing agent, the tin exists in the mass as an alloy with zinc and so dissolves in the acid more quickly than if it existed as small tin buttons. With dressed tin ores the solution is clear and evidently complete; but with two or three gramme charges of slime or tailing the liquor is more than a little turbid with silica and other material. This barely interferes with the ease of the titration when iodine is used, and it is hard to believe that anyone will exert himself much to remove the turbidity. It may be urged that such turbidity obscures the evidence of the complete solution of the tin and so destroys security: but there are other grounds for confidence, and resting on these I anticipate that a large proportion of the samples will be assayed by such a direct method.

There are some ores and products from the dressing-floors that are unsuitable; some of these will demand a preliminary cleaning in acid; but others may yield to an attack by chlorine, supplementing the action of the hydrochloric acid. In this case the liquor will contain stannic chloride and the method becomes what I am calling an indirect one.

The tin of the ore can be got in solution as stannic chloride by first reducing to metal and then dissolving in hydrochloric acid and chlorine as just indicated. Or it can be effected by Mr. Richard Pearce's method in which the ore is fused with caustic soda or potash or with sodium peroxide in a nickel or iron dish, ladle, or crucible; to be followed by the solution of the fused mass in an excess of hydrochloric acid. This has the advantage of getting everything, quartz and all, into solution, and the disadvantage of being available only for rather small charges of say half a gramme or thereabouts. The preparation of stannous chloride from the stannic chloride liquors can be done by the old method of precipitating tin as a metallic sponge on zinc and then dissolving it in hydrochloric acid. But by using iron (as recommended by Mr. Pearce) or nickel (as I recommend) instead of the more active zinc, the reduction stops at the stage of stannous chloride and the liquor is ready for titration; here again the advantage lies with the more direct method. The nickel imparts a green colour to the liquor that is fatal for the ferric chloride titration, but does no harm to the determination by iodine. The longer method with formation of stannic chloride and subsequent reduction by nickel is the easier to teach to beginners, and is the one described in my text book on assaying. But the direct method presents no special difficulty. The changes in the instructions are as follows: The mass, cooled, after treatment with zinc vapour, is placed in an 8-oz. flask, and the crucible and cover are cleaned by immersion in 60 c.c. of hydrochloric acid. The acid is next poured all at once into the flask, which should then for safety's sake be connected with a Kipp to collect the evolved hydrogen. When the action has slackened, the acid is heated to boiling; the boiling is continued for five minutes and the acid vapours are led into a Woulff's bottle, which serves as a condenser. The hydrogen in the Kipp is burnt. At the end of five minutes, boiling water is poured into the flask, without removing the burner, until the bulk is about 150 c.c. A nickel coil is then introduced, just as in the longer method, and the boiling is continued for five minutes. The rest of the work—cooling in carbonic acid and titrating with iodine—remains unaltered.

That any particular ore is suitable for the direct treatment is easily determined. For example, a charge from the middle product from a Buss table of 2·24 gm. by the direct method gave a result of 14·5 lb. Two charges of 2·8 and 1·4 gm. by the longer method gave 14·6 and 14·7 lb. respectively. Another charge of 1·4 gm., cleaned with acid and then treated by the longer method gave 14·7 lb."—J. J. Beringer. —*Mining Magazine*, Nov., 1909, p. 231. (J. A. W.)

MINING

Methods of Deep Lead Mining.—"The two principal methods are by shafts and by adits, after the depth of the lead from the surface and its position in the lease of the company have been ascertained by one or several series of bores to bed rock. This determination of the position and gradient of the lead is essential, preparatory to the fixing of the site of the shaft or adit, the sinking of the shaft, and the making of the adit and drives. The neglect in carrying out such preliminary and obviously necessary work has led to the almost criminal waste of hundreds of thousands of pounds of capital in the State.

The initial work of defining the courses of the leads and their depths from the surface is often very costly, owing to the thick superincumbent layers of hard basalt or the running character of saturated sands. In many of the southern and central northern fields, such as in the Pitfield, Durham, Loddon, and Campaspe systems, the cost ranges from 9s. to 15s. per foot; but in many other districts, such as at Chiltern and Rutherglen, the cost varies from 1s. 6d. to 3s. per foot; for the strata overlying the leads consist of clay, sand and gravel, which are easily and quickly bored through.

The mining operations following the sinking of the shaft are usually carried out according to either the Chiltern or Ballarat system. The work of shaft sinking and commencing a main reef bottom level is practically the same on all the fields, the cost varying with the density of the strata being sunk through; but the subsequent mining methods in the Chiltern district differ considerably from those employed in Ballarat, Maryborough, and Creswick mines. In the Chiltern system on completing the sinking of the shaft, two chambers are cut in the bed rock—one at from 40 to 60 ft., the other within a few feet of the bottom of the shaft. The bottom level or 'reef drive' is extended from the lower chamber to a point under the centre of the lead, where drainage bores, extending well up into the wash and drift, are made, and pumping is continued until a block of ground is sufficiently unwatered to safely allow the extension into the wash of the upper level or 'leading wash drive.' From the upper chamber a drive or level is made at such a depth that, allowing for a slight gradient (usually from 1 to 4 ins. to 100 ft.) it will intersect the lead at the deepest part where it is intended to commence work. This, on reaching the wash, becomes the 'leading wash (or pilot) drive,' and then follows the course of the lead. From the 'leading wash drive' cross drives at right angles and at intervals of 100 to 300 ft. are extended on each side until 'high reef' (the bank of the lead) is reached. Thus the dual purpose is served of cutting off any remaining water from the ground through which the leading wash drive is made and of proving the direction of the lead. Near the chambers at the main shaft a vertical chute and 'pass by' (ladderway) connecting the leading wash drive and reef drive, are made. Similarly all chutes subsequently completed in the mine rise from the reef drive into the leading wash drive, and not as in the Ballarat method into solid and sometimes only partially drained ground. This eliminates all risk of a burst of water or drift when the chute rises into the auriferous stratum, and, by providing dry ground, cheapens the work and reduces the discomfort and risk of the miner.

As the face workings are extended, a timber drive and return (upcast) air-course is made in solid ground. It is on the same level as the leading wash drive and generally parallel with but away from the lead. This leads to an upcast compartment in the main shaft or to a timber shaft, some thousands of feet away, up or down the lead. The cross drives previously referred to are holed through, where necessary, to the air and timber drive, providing good ventilation and means of escape in the event of the main shaft being lost or the mine flooded. They also allow of the transport of a maximum quantity of dirt to the main shaft by keeping the bottom level entirely clear of timber trucks. The reason for having the air and timber drive in the solid bed-rock is for better security, as the ground crushes down on the worked out portion of the lead. At some of the mines in the Chiltern district the long timber and air drive is replaced by two or three shorter drives, each connected with separate shafts. This allows the sending of timber to the various working faces up and down the lead, without having to truck it from the main shaft along the bottom level, laboriously haul it up a short or balance shaft, and then along the wash drive to the faces. Between the cross drives and at right angles to them, 'blocking drives' are made at every 46 ft. This allows 20 ft. of ground to each opposite blocking party after deducting 6 ft. for the width of the drive. About 2 ft. of wash dirt and from 12 to 18 ins. of the 'reef bottom' (bed-rock) are usually taken out, the whole width of the lead being worked evenly across in one face. No wash dirt is allowed to be stowed away in the old workings. The reef drives are subsequently nearly under and in the same direction as the leading wash drives, the faces of which are always kept for some 300 to 400 ft. ahead, to pilot, as previously explained, the course of the reef drive.

Mr. John Cock, of Chiltern, to whom the credit of this system must be given, has proved that under very favourable conditions the proportion of ground, relative to the whole lead workings, taken out in driving, and which, for economic mining, should be as low as possible, is about 20 %. The average at Chiltern is from 25 to 30 %.

In the Ballarat system, assuming that some preliminary data by surface boring have been obtained, the shaft is sunk to the necessary depth. Near the bottom a chamber or plat is cut, and the reef drive extended in the supposed direction of the lead. Bores at frequent intervals are put up into the alluvial strata (sometimes from small chambers or recesses cut in the side of the drive) to drain the ground and prospect for the gutter, which, however, occasionally takes a different course from that anticipated and results in a waste of driving that might have been avoided had the sinuosities of the lead been properly determined by surface boring.

When the flow of water from the drainage bores has been sufficiently reduced an intermediate drive above the reef drive is made, from which rises are put up into the wash, which is then opened out by wash drives. In a number of mines the intermediate drive commences from a balance shaft near the face of the bottom drive, and forms the only passage or exist from the wash workings to the reef drive and shaft—a more or less dangerous system of opening out an alluvial mine, more especially in any district where bursts of water occur.

As branch drives off the intermediate drives are extended in various directions, and rises are made, the wash is prospected, and all areas considered payable are cut up into irregular blocks, many of which are, however, subsequently left as unprofitable to mine. The following method, known as panelling, is adopted when taking out the wash dirt. Commencing along the one edge of a block of ground, a low drive 6 ft. wide and high enough to include the auriferous stratum and sufficient of the bottom (bed-rock) is made at right angles to the wash drive, the back or roof being secured by panelling laths supported at the ends by short light props placed close together and always kept well up to the face as the wash dirt is taken out. On reaching the limit of the panel, usually about 16 ft. in length, the miner returns to the wash drive and commences another strip or panel along the edge of the block, repeating the process until the whole of the wash dirt in the block has been removed.

Some controversy has occurred as to which is the better system—the blocking out in a face across the whole width of the lead, as is done at Chiltern, or the panelling as at Ballarat.

In the writer's opinion each method has much to recommend it, and each is suitable for a different class of ground. The difference between the two systems of mining really lies in the method of developing the wash dirt workings. In the Chiltern system the wash dirt faces are always ahead of the reef drive, and no intermediate drive is required, the leading wash drive or the air and timber course taking its place. In the Ballarat system either the reef drive or the intermediate drive is ahead of the wash workings, and the position of each depends upon the assumption of the manager or directors regarding the course of the lead ahead of the mine workings. There are usually some data, of the nature of surface bores, to assist in arriving at a decision, but too frequently, the supposed course of the lead differs widely from its actual position, and the mine is therefore handicapped by the burden of unnecessary dead work."—STANLEY HUNTER, Memoir No. 7, Geological Survey of Victoria.—*London Mining Journal.*—Oct. 16, '09, p. 77. (A. R.)

NEW METHOD OF GOLD MINING.—"In fulfilment of a long-projected idea, it is announced that on Modder B the workings are to be planned on somewhat the same system as generally adopted in coal mines. The shaft is to be placed in the centre of the property, and levels driven along the reef to the boundaries of the property, with a view of establishing its lie and value. It is proposed to drive the permanent levels in the solid so as to be independent of any variations in the reef, and enable them to be driven straight to obtain the best and cheapest method of haulage. The main levels are to be laid with 30 lb. track and 30 in gauge, the trucks to carry one ton. These trucks will be tipped into a bin at the shaft, capable of containing over 500 tons of ore. From these levels will run rises a thousand feet apart to the outcrop, from which again levels will be driven, as in coal mining, to convey the ore to the inclines rising from the permanent levels. The advantages claimed for this system of working are simplicity and cheapness, whilst the ventilation of the workings ought to be simplicity itself. The handling of the ore ought to be reduced to a minimum. The success of the method will, however, altogether depend upon the details of mining followed."—*London Mining Journal.* Nov. 20, '09, p. 285. (A. R.)

COST OF MINING IN CHILE.—"At the September meeting of the American Institute of Mining Engineers, William Braden gave a short account of the conditions of labour and the cost of mining in Chile, drawn from his experience in connection with the Braden copper mines. It is usually supposed that Spanish-American workmen are inferior to those in the United States, but Mr. Braden finds that, with proper treatment and organisation, labour in Chile yields as much man for man, and more dollar for dollar, than in the Western United States. The first necessity for the mine manager is to organise a staff capable by precept and practice of teaching the native the details of their work and of encouraging sobriety, decency, and self-respect. The customs of the country must not be overridden rough-shod, but only altered gradually when they happen to interfere with efficiency. Comfortable quarters and other uplifting elements of life should be provided. That Mr. Braden's policy has yielded excellent results may be seen from his figures relating to costs. To understand them fully it is desirable to know something of the mine and the method of working. The deposit is a zone of fractured diorite having a hanging wall of brecciated tuff, which has a dip of 65°. The mine is quite dry, and the only timbering required is for the framing of the ore-chutes. The system of mining is one of overhead mining in a series of transverse stopes, the intervening pillars to be extracted subsequently by caving. The ore is moderately hard. Air-drills are used wherever possible, and are intelligently handled by the Chilean miners. In fact, before long, hand-drilling will cease. Power for haulage and other purposes is supplied by electric current generated by Doble water-wheels. The ore is sent to the mill, 2,900 yards distant and 600 yards lower, by an aerial ropeway. Mr. Braden quotes the operating report for November, 1908. Upon the basis of 7,304 tons extracted and milled the cost was 5s. 6d. per ton. The air-drills ($2\frac{3}{4}$ in.) operated 164 man-days of 9 hours each and drilled 5,082 ft. at a cost of 1d. per foot. By hand-work 13,855 ft. was drilled in 1,008 man-days at a cost of 4d. per foot. More than one-half of the labour was performed by contract, and in the stopes the contracts were let on the basis of the number of feet of hole drilled. Including all labour, 12 tons of ore were broken per man-day. Development was in the nature of drifts, cross-cuts, and raises, with an average section of 5·4 sq. yd, and the cost of labour, supplies, explosives, shovelling, and tramming was 14s. 8d. per linear foot, or 2s. 6d. per cu. yd. The amount of development was 379 linear ft., with $876\frac{3}{4}$ man-days, making 13 ft. per month per man. An example may be given: a main adit with a section of $7\frac{1}{8}$ sq. yd. was driven $132\frac{1}{2}$ ft. in 30 days; the contract price was 13s. per ft, exclusive of drill-sharpening and repairs."—*Mining Magazine*, Nov., 1909, p. 245. (A. R.)

MINE FIRES.—"A novel type of temporary barrier in case of mine fires is in use in Saarbruecken, in Rhenish Prussia (according to *Bergbau*), for the isolation of the affected part. The material used is compressed hay in blocks, and when the passage has been closed up with these the interstices are filled with loose hay and the structure is then saturated with water. This causes the hay to swell greatly and by thus preventing the access of any air gives quickly and easily the first condition necessary for getting the fire under control. The material is very light and two men could easily close up any ordinary entry by its means in half an hour, using in the presence of overpowering gases and smoke a helmet to enable them to breathe while erecting the partition. If necessary a more permanent wall can be built behind the hay shield. This method has been tried and found highly successful."—*Mining Magazine.*—Nov. 1909, p. 242. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE METALLURGY OF THE COMMON METALS. By L. S. AUSTIN. 2nd Edition. Large 8vo. 494 pages. (San Francisco: *Mining and Scientific Press.* London: *Mining Magazine.*) $4.

This can be commended as a very useful and, on the whole, up-to-date textbook, though the description of South African practice in gold cyaniding is rather behind the times. It deals chiefly, however, with modern American practice in the metallurgy of gold, silver, iron, copper, lead and zinc, and in a sufficiently exhaustive manner, except in the case of iron, which only receives about twenty pages, steel being barely touched upon.

The scope is somewhat broader than the title would indicate, and there are excellent sections on general metallurgy (dealing with thermochemistry, fuels, refractory materials, sampling, etc.), plant and equipment, and commercial considerations.

It is written in very clear and concise style, and very little space or time is devoted to obsolete processes, which, combined with the thin paper used, has resulted in a surprising amount of information, much of which was formerly only to be found after lengthy searching through technical periodicals, becoming available in comparatively small compass. This is, perhaps, particularly noticeable in the chapter on roasting, in which, besides full descriptions of modern methods, including pot-roasting, full details of outputs and costs are given so that a very fair idea of the relative performances of the furnaces described can be arrived at. In fact, this attention to the commercial and practical side of the subject is most pronounced all through the book, though without in the least obscuring the presentation of the theoretical part or principles underlying the practice.

The illustrations, nearly 200 in number, are extremely well done, and, in short, the book is a very creditable production, and can be confidently recommended for the perusal both of students and those in practical life. (G. H. S.)

THE CHEMISTRY OF CYANIDE SOLUTIONS RESULTING FROM THE TREATMENT OF ORES. (Second Editon). By J. E. CLENNELL. New York; McGraw Hill Book Co. pp. 202. Net 10s. 6d.

"In this second edition of this book the text has been revised and enlarged. The author gives a short description of well known methods, with discussions where necessary. Modifications of existing methods are described. In addition to reliable laboratory methods, some rough tests are given, that suffice for daily control of routine operations. As a thorough understanding of the composition of the cyanide solution is important in the efficient control of cyaniding operations, a knowledge of the methods given in this book should be of value to managers and chemists in charge of cyanide mills."—*The Mining World.*—Feb. 26, 1910, p. 475. (W. A. C.)

Anglin, S. The Design of Structures. A practical treatise on the building of Bridges, Roofs, etc. 5th Edition. Revised. Cr. 8vo, pp. 536. *C. Griffin.* 16s.

Foster, Sir Clement Le Neve. A Treatise on Ore and Stone Mining. 7th Edition. Revised. 8vo, pp. 832. *C. Griffin.* Net 28s.

Marine Oil Engine Handbook, The. Cr. 8vo, pp. 132. *E. J. Larby.* Net 1s.

Savoia, H. Metallography applied to Siderurgic Products. 12mo. *Spon.* Net 4s. 6d.

Warren, William Henry. Engineering Construction in steel and Timber. 8vo. *Longmans.* Net 18s.

Abstracts of Patent Applications.

(C.) 43/10. John Lowther. Improvements relating to bearers for mine shafts. 10.2.10.

This refers to the use of an "I" beam girder as a shaft bearer. The "I" beam being cut in two and bolted together between fish plates after being put into place. The claim is for any metallic beam in halves to be rigidly joined together.

(C.) 25/10. Georg Schwabach. Improvements in regulating devices for gas and the like supply plants. 28.1.10.

This refers to means of regulating the discharge of gases where one stream of gas issuing from a nozzle drives the other stream of gas in the form of an ejector. The claims are the means of adjusting the two streams relatively to each other by the insertion of a movable body whose shape makes the adjustment automatic for each setting.

(C.) 2/10. George Boulton Brown Elliott. An improved grating for mills for pulverising quartz and similar material. 4.1.10.

This refers to a form of grating or discharge screen for crushing mills operating on the centrifugal principle. The claims are that the screen does not choke as most others do.

(C.) 513/09. Claude Albermarle Bettington. Improvements in and relating to apparatus for pulverising coal and other materials. 12.11.09.

This patent consists of a revolving beater for pulverising coal and mixing it with air or gas with the outlet formed so as to throw the crushed material vertically downward into a classifier, the oversize of the classifier returning to the crusher and the undersize going on to be consumed.

(C.) 217/09. John Hudson Hughes. Improvements in the operation of the stamps of stamp batteries. 1.5.09.

This refers to means claiming to increase the rotation of ordinary gravity stamps and thus assisting their crushing action. The means employed consist of inserting a roller between the cam and the tappet, the roller being supported by a flexible or hinged arm to keep it in position.

(C.) 274/09. William Cullen (1), Gilbert Frederick Ayers (2). Method of precipitating zinc from solutions. 8.6.09.

The above application describes a method of recovering zinc as zinc hydrate from solutions of zinc sulphate, as produced in the acid treatment of zinc gold slime, by admixture with finely divided magnesium hydrate.

(C.) 318/09. J. Oliver. Improvements in the lining of tube mills and the like. 7.7.09.

This application relates to improvements in tube mill liners, which consists of employing "radially disposed pipes arranged in abutting longitudinal rows." These pipes are cut into short lengths of 5 in., forming a honey-comb series of recesses which are intended to retain a portion of the mill charge and form a surface upon which the remainder of the charge moves when the mill is rotated.

(C.) 8/10. Arthur Eilert Drucker. Continuous slimes vacuum filter. 7.1.10.

This application relates to an improved form of vacuum slime filter, of the tailing wheel type. This filter has the "leaves" arranged tangentially on the periphery of the wheel, with the usual vacuum and compressed air connections to the wheel hub. It operates in connection with a hopper bottomed tank, lifting out a certain amount of slimes with each revolution. Wash water is added at the highest point of elevation and the wash cake discharged just before entering the supply tank.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 128/10. Stephen Benko. Improvements in or relating to galvanic elements. 24.3.10.

(P.) 130/10. Henry James Shedlock Heather. A means of facilitating the control of induction motors of the slip-ring or equivalent type. 26.3.10.

(P.) 132/10. Henry James Shedlock Heather. An improved method of controlling the starting and retarding of induction motors of the slip-ring or equivalent type. 29.3.10.

(C.) 133/10. August Brandes. Improvements in an relating to the treatment of liquids. 29.3.10.

(C.) 134/10. Alfred John Cotton. Means for utilising the engines of motor cars for imparting power for useful purposes. 29.3.10.

(P.) 137/10. Elisha Tippett. Improvements relating to hammer drills for removing the detritus or cuttings from the bore hole. 30.3.10.

(P.) 138/10. Peter Arthur Walsh. Improvements in linings for tube mills. 31.3.10.

(C.) 139/10. Carl Neff (1), August Brandes (2). Improvements relating to the prevention in incrustation in steam boilers and the like- 31.3.10.

(P.) 140/10. Edward George Clifford-Jones. Improvements in accelerating devices for gravity stamps. 1.4.10.

(C.) 141/10. Francois Dallenagne (1), Henri Dallenagne (2). Improvements in ore concentrating and separating apparatus. 1.4.10.

(C.) 142/10. Quintin Moore (1), James Cunningham (2), James William Bradbeer Stokes (3). Improved process and apparatus for the production of gas from bituminous fuel in gas producers. 1.4 10.

(C.) 143/10. Quintin Moore (1), James Cunningham (2), James William Bradbeer Stokes (3). Improved process and apparatus for the production of recoverable ammonia in the manufacture of producer gas. 1 4 10.

(P.) 144/10. Benjamin Donkin Johnson (1), Charles Edwin Marson (2). Improvements in means for handling and transporting cable drums and the like. 2.4.10.

(C.) 145/10. Thomas James Francis (1), Ryland Stainslaus Louis (2). Improvements in or relating to expanding pulleys. 2.4.10.

(C.) 146/10. Otto Schonherr (1), Johannes Hessberger (2). Improvements in and means for the production of long continuous electric arcs. 2.4.10.

(P.) 147/10. Reuben Jones. Improved means for the connection of cutters, or bits, of rock drills with their drill bars or shanks. 2.4.10.

(P.) 148/10. Peretz Maniom (1), Benerd Certoon (2). Improvements in means for checking runaway skips and the like. 2.4.10.

(C.) 149/10. Robert Rosenthal. Device for automatic closing and opening of gates at railway level crossings. 2.4.10.

(P.) 150/10. Wilfred Brereton Crofton. Improvements relating to road vehicles. 4.4.10.

(P.) 151/10. Edward Harrison. Improvements in dump forming apparatus. 4.4.10.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Allan, J. R., *l/o* Brixton: Maraisburg Hotel, Maraisburg.
Barrett, S. S., *l/o* Eureka: P. O. Box 87, Barberton.
Blyth, W. B., *l/o* Day Dawn; Vivian Gold Mine, Harris, *via* Lawlers, W. Australia.
Boright, S. H., *l/o* Springs; Lonely Reef G. M. Co., Ltd., Private Bag, Inyati, Rhodesia.
Carruthers, J., *to* 89, High Street, Brixton.
Cooke, R. C. H., *to* Salisbury Club, Salisbury, Rhodesia.
Douglass, Ross E., *l/o* Nevada, U.S.A.: c/o Braden Copper Co., Rancagua, Chile, South America.
Dunning, F. G., *l/o* Germiston; Gwelo, Rhodesia.
Francis, W., *l/o* Penhalonga; Union G. M. Co., Ltd., Macequece, Portuguese East Africa.
Halford, J. E., *l/o* Randfontein: P. O. Box 2286, Johannesburg.
Homersham, E. C., *l/o* Randfontein: 19, Clevedon Mansions, Lissenden Gardens, Highgate, London N.W.
Macfarlane, T. M. M., *l/o* London: Apartado 1913, Mexico, D.F.
Mannheim, H. C., *to* P. O. Box 155, Bulawayo.
Pinckvos, E., *l/o* Sumatra: 44, Eimsbüttlerstr., Hamburg, Germany.
Reyersbach, L. J., *l/o* Johannesburg; c/o Messrs. Wernher, Beit & Co., 1, London Wall Buildings, London, E.C.
Richardson, A., *l/o* Cleveland; Anglo-French Exploration Co., Ltd., P. O. Box 3133, Johannesburg.
Roche, H. F., *l/o* Johannesburg: Salisbury Club, Salisbury, Rhodesia.
Rogers, N., *l/o* Cleveland: Brakpan Mines, Ltd., P. O. Box 3, Brakpan.
Schaap, M., *to* Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.
Wagner, P. A., *l/o* Johannesburg: c/o South-West Africa Co., Grootfontein Nord, German South West Africa.
Wisdom, G. E., *to* Don Proprietary Mines, Selukwe, Rhodesia.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. MAY, 1910. No. 11.

Proceedings
AT
Ordinary General Meeting, May 21, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, May 21st, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

50 Members: Dr. J. Moir, Messrs. F. F. Alexander, E. H. Croghan, K. L. Graham, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, A. F. Crosse, E. H. Johnson, A. C. Adams, S. Beaton, B. V. Blundun, W. A. Coles, M. H. Coombe, W. M. Coulter, T. Donaldson, C. E. Hall, W. L. Hamilton, C. B. Hilliard, J. H. Johnson, J. Lea, G. Melvill, P. T. Morrisby, W. Nicklin, F. B. Ogle, E. A. Osterloh, E. Pam, J. F. Pyles, G. A. Robertson, O. D. Ross, A. Salkinson, A. H. Scarf, W. Sharp, B. C. Solly, S. H. Steels, W. A. C. Tayler, A. Thomas, J. R. Thurlow, C. Toombs, J. F. Walker, J. P. Ward, E. M. Weston, A. Wilkinson, W. O. Williams, L. J. Wilmoth and P. H. M. Whyte.

12 Associates and Students: Messrs. A. R. Adams, E. A. Barry, E. Brennan, C. E. Deakin, W. J. R. Hunter, A. King, G. Musson, N. N. Newland, S. H. Olivier, H. G. Roake, H. Stadler, and W. Webb.

13 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the April *Journal*, were confirmed.

NEW MEMBERS.

Messrs. E. H. Croghan and G. Melvill were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Bluett, Herbert Alfred Newton, M.M. Siman, Lebong Tandai, Benkoelen, Sumatra. Assayer.

Chambers, Alexander Edmund, P. O. Box 29, Maraisburg. Cyanider.

Fuller, John Patrick, M.M. Siman, Lebong Tandi, Benkoelen, Sumatra. Mining Engineer.

Gifford, Henry John, Champion Reefs P. O., Mysore State, South India. Mining Engineer.

Holtby, Alfred Charles, P. O. Box 4336, Johannesburg. Consulting Electrical Engineer.

Margerison, Samuel Burton, Telok Betong, South Sumatra. Mining Engineer.

Robertson, George Anderson, Post Office, New Primrose, Germiston. Amalgamator.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associate.—

Minsch, Walter Cornelius, Telluride, Colorado, U.S.A. Mining Engineer.

As Student.—

Olivier, Salomon Hercules, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Cyanide Learner.

The President: Before beginning the general business of the meeting I would like you, as members of the Society, to express your regret at the loss which the Empire has sustained by the death of King Edward VII., and also your sympathy towards the King, the Queen, the Queen Mother and all the members of the Royal Family.

The members all rose to their feet in silent acquiescence.

General Business.

THE ANNUAL BALLOT.

The President: The next business is the election of scrutineers for the annual ballot for President, Vice-Presidents, Honorary Treasurer, and Members of Council, to take place at the next meeting of the Society.

The following gentlemen were elected:—

Messrs. R. Stokes, T. T. Nichol, S. Newton, A. D. Viney, A. Thomas and W. A. C. Tayler.

TEST FOR PRUSSIC ACID.

Mr. A. Whitby (*Member of Council*): I have a few remarks to make on Dr. Moir's test for prussic acid. If there is one thing I avoid more than another, it is the construction of chemical equations. It appears to me, however, that Dr. Moir has placed an equation before us which is not strictly accurate. Bearing well in mind that Dr. Moir is gifted with a powerful vein of sarcasm, I feel that I am, in a measure, inviting an exercise of that same vein in venturing to dispute his equations.

You will remember that the original equation put upon the board by Dr. Moir did not balance, and that H_2O_2 formed the basis of his claim. I observe that he has modified his empiricism in the *Journal*, and now only claims oxygen as the product of reaction. It is therefore evident that Dr. Moir is himself a little uncertain about his chemical equations, which fact somewhat encourages me to proceed with mine.

$$Cu\bar{A}_2 + 2HCN = CuCN + 2H\bar{A} + CN$$

This empiricism, however, might be modified thus

$$CuA_2 + 3HCy = HCuCy_2 + 2H\bar{A} + CN$$

The acid $HCuCy_2$, according to Dr. Moir himself, is very unstable and breaks up in this manner,

$$HCuCy_2 = CuCy + HCy$$

But there is yet another equation

$$6HCy + 3Cu\bar{A}_2 = 3CuCy_2 + 6H\bar{A}$$
$$3CuCy_2 = [Cu_2Cy_2CuCy_2] + Cy_2$$

This substance is mentioned in the text books, and Comey gives two others which are hydrated compounds.

These equations therefore seem to show that cyanogen, and not oxygen, is the gas product, but in corroboration of my statement I am pleased to be able to call Dr. Moir himself to witness to the point.

On p. 211 of the *Journal* of December last he makes the statement that when $CuSO_4$ is added to cyanide solution we get

“ $CuSO_4 + 3KCy = KCuCy_2 + K_2SO_4 + CN.$

In presence of acid we get CuCy precipitated as, for example,

$$KCuCy_2 + H\bar{A} = CuCy + K\bar{A} + HCy.”$$

Now, I am not denying the value of his test by any means. It appears quite possible that cyanogen thus liberated might attack the hydrogen, say, in a hydroxyl radicle of an organic compound, thus liberating oxygen, but here I must confess I am at fault, and must allow Dr. Moir scope in his particular sphere of organic chemistry to thrash out this side of the subject.

Dr. J. Moir (*Vice-President*): I am obliged to Mr. Whitby for having brought forward these equations, for it is just as well that these disputed points should be properly discussed. The main answer is, I think, that it is quibbling about terminology. Mr. Whitby is mistaken in supposing I wrote a wrong equation on the board last month. What I wrote was:

$$Cu\bar{A}_2 + HCN + H_2O = CuCN + 2H\bar{A} + \tfrac{1}{2}H_2O_2,$$

which is essentially what appeared in the *Journal* except that I then broke down H_2O_2 into $H_2O + O$. My personal belief is (1) that Mr. Whitby's first equation represents what happens in strong acid solutions; (2) the equation quoted from the December *Journal* is what happens in strong alkaline solutions; and (3) the equation given in this reply represents what happens when the cyanide is very small in comparison with the copper. I consider the equation I published in the *Journal* is the simplest one, but I am quite willing to accept any of the other equations. The main point is that when a cupric salt becomes cuprous, anything else oxidisable will become oxidised.

GRADING ANALYSES AND THEIR APPLICATION.

By H. Stadler (Associate).

The universal tendency of modern metallurgical practice towards fine grinding made the desirability of introducing and adopting a uniform system of series of standard screens for comparative purposes strongly felt, and after laborious and careful investigation a standard table of laboratory screens has been established and adopted by the Institution of Mining and Metallurgy.

Although fully appreciating the merits of the work achieved and the considerations and principles which have guided the I.M.M. in establishing and adopting the present table of standard laboratory screens, I have found it necessary to establish a slightly different ratio, and to extend the classification, in order to more exactly secure a grading which is equivalent to a successive reduction by one half of the volume (or weight) of the particles, or, if only the alternate grades are taken, of a reduction by 1 : 4. As however, the sizes of this grading scale are in agreement with, or very close to, certain of the I.M.M. standards; I have shown in the attached table the nearest corresponding standards of laboratory mesh.

The following notes will show how the varying proportions of certain standard sizes of particles may be valued in units of energy (E U.), which will allow of an exact expression of the efficiency of the crushing operation for purposes of useful comparison.

I.—Grading Analyses.

Mines Trials Committee (M.T.C.) Grades.—

1. The most rational and logical system of classification into grades, is undoubtedly to base it on the reduction of the *volume* (or weight) of the particles, especially for mining purposes, as the assay-values are given with reference to this unit.

TABLE A. STANDARD GRADES AND SCREENS.

MINES TRIALS COMMITTEE—(M.T.C.) STANDARD GRADES.

Ordinal Number or Mech. Value.	Mesh Aperture (=sides of theoret. cubes.)		Volume (or weight) of theoret. cubes.		Area of Fracture from Unit.	Commercial Denomination of Screens. Approx. number of Meshes.		Ordin. Number or Mech. Value of Grade.
EU.	In.	mm	$\frac{1}{1000}$ cub. in	cub. mm	sq. in.	p. lin. in	p. sq. in	EU.
30	·00098	·02480	·000000	·00002	3072·0	↑		30·0
29	·00123	·03125	·000001	·00003	2438·1	–200	Grade incl. Washed Slimes	29·0
28	·00155	·03937	·000003	·00006	1935·4			28·0
27	·00195	·04961	·000007	·00012	1536·0			27·0
26	·00246	·06250	·000015	·00024	1219·1	200	40000	26·0
25	·00310	·07875	·000030	·00049	967·68	150	22500	25·0
24	·00391	·09922	·000059	·00098	768·0	130	17000	24·0
23	·00492	·1250	·000119	·00195	609·52	110	12000	23·0
22	·00620	·15750	·000238	·00391	483·84	**90**	8000	22·0
21	·00781	·19844	·000477	·00781	384·0	75	5500	21·0
20	·00984	·250	·000954	·01562	304·76	**60**	3500	20·0
19	·01240	·3150	·001907	·03125	241·92	50	2500	19·0
18	·01562	·39688	·003815	·06250	192·0	40	2000	18·0
17	·01969	·50	·007629	·1250	152·38	35	1200	17·0
16	·0248	·630	·015259	·250	120·96	30	900	16·0
15	·03125	·07938	·030517	·50	96·0	25	600	15·0
14	·03937	1·0	·061035	1·0	76·19	20	400	14·0
13	·04961	1·260	·122070	2·0001	60·480	18	300	13·0
12	·06250	1·5875	·244140	4·0002	48·0	14	200	12·0
11	·07875	2·0	·488281	8·0003	38·102	11	120	11·0
10	·09922	2·520	·976562	16·000	30·240	8	65	10·0
9	·1250	3·1750	1·9530	32·001	24·0	6	35	9·0
8	·15750	4·0003	3·9060	64·003	19·051	4	15	8·0
7	·19850	5·040	7·8120	128·02	15·120	3·5 Ap'r Cre	12	7·0
6	·250	6·350	15·625	256·03	12·0	1/4″	9	6·0
5	·3150	8·0005	31·250	512·06	9·520	5/16″	6	5·0
4	·39690	10·080	62·50	1024·1	7·560	3/8″	4	4·0
3	·50	12·70	125·0	2048·3	6·0	1/2″	3	·0
2	·630	16·0	250·0	4096·5	4·763	5/8″	2	2·0
1	·79370	20·160	500·0	8193·0	3·780	3/4″	1	1·0
0	1·00	25·4	1000·0	16386·0	3·0	1 inch	—	0·0
Arithmetical Progression R = 1	$R = \frac{1}{\sqrt[3]{2}}$ = ·7937		$R = \frac{1}{2}$ = 0·5		$R = \sqrt[3]{2}$ = 1·2599			
	Geometrical Progression.							

Institution of Mining & Metallurgy. Standard Laboratory Screens.

Ordin. Number or Mech. Value of Grade. EU.	No. of Mesh. lin. in.	Mesh Aperture. in.	Diam. of Wire. in.	Area of Discharge.
26·02	200	·0025	·0025	25·0
24·81	150	·0033	·0033	24·50
23·77	120	·0042	·0041	25·40
23·01	100	·005	·005	25.0
22·60	90	·0055	·0055	24·50
22·08	80	·0062	·0063	24·60
21·49	70	·0071	·0071	24·70
20·81	60	·0083	·0083	24·80
20·0	50	·01	·01	25·0
19·03	40	·0125	·0125	25·0
17·80	30	·0166	·0167	24·8
16·99	25	·020	·020	25·0
16·02	20	·025	·025	25·0
15·06	16	·0312	·0312	24·92
13·81	12	·0416	·0417	24·92
13·01	10	·05	·05	25·0
12·08	8	·062	·063	24·60
10·6	5		·10	25·0

The M.T.C. grades are based on the assumption of a successive reduction in volume (or weight) of the screen products by one-half from grade to grade.

2. By reducing the cube of the unit successively by one-half of its volume and assuming these fractures to be again of cubical shape, each size of this series of theoretical cubes obtained, represents a grade of a reduction scale of the ratio 2.

3. If to each of these grades an ordinal number be given, beginning with 0 as representing the unit of 1 in. cube; the following formulæ determine the values of the data for the series of theoretical cubes.

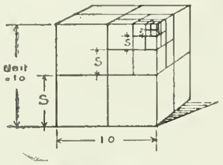
4. By taking the sides of this series of cubes, as clear mesh apertures of a set of screens, it is admissable to assume that the functions of the irregularly-shaped average particles, determined

TABLE B.

Relations between values of the data for

SERIES OF THEORETICAL CUBES

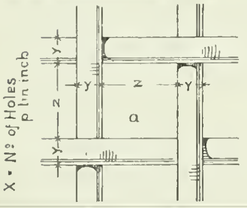
Obtained by Successive Reduction of Cube of Unit.



		Number of Pieces P.	Sides of Cubes S.	Volume of Cubes V.	Area of Fracture F.	Ordinal or Energy Number N.
	Number of pieces produced from Unit P =	—	$\frac{1}{S^3}$	$\frac{1}{V}$	$\frac{F^3}{27}$	2^N
V. & S. Ratio values in regard to Unit.	Sides of cubes (mesh-apertures) S =	$\frac{1}{\sqrt[3]{P}}$	—	$\sqrt[3]{V}$	$\frac{3}{F}$	$\frac{1}{\sqrt[3]{2^N}}$
	Volume of cubes (or weight) V =	$\frac{1}{P}$	S^3	—	$\frac{27}{F^3}$	$\frac{1}{2^N}$
	Area of fracture with reference to Unit F =	$3\sqrt[3]{P}$	$\frac{3}{S}$	$\frac{3}{\sqrt[3]{V}}$	—	$3\sqrt[3]{2^N}$
	Ordinal or energy number of grades N =	$\frac{\log P}{\log 2}$	$\frac{\log\frac{1}{S^3}}{\log 2}$ $= -10\log S$	$\frac{\log\frac{1}{V}}{\log 2}$	$\frac{\log F - \log 27}{\log 2}$ $= \frac{N}{10} + 0{\cdot}477$	—

N.B.—The Surfaces exposed are double of the Area of Fracture (F).

TABLE C.



WIRE-SCREENING.—Relations between DIAMETER OF APERTURE, DIAMETER OF WIRE, SCREEN AREA and NO. OF HOLES per linear inch in any Screen having the same number of holes each way and manufactured with one gauge of wire only.

	Disch. area A. Diam. Wire Y.	Disch. area A. Apertures Z.	Disch. area A. No. of Holes X.	Dia. Wire Y. Apertures Z.	Diam. Wire Y. No. of Holes X.	Apertures Z. No. of Holes X
% of Discharge of Area : A =	—	—	—	$\left(\frac{10\ z}{y+z}\right)^2$	$100(1-xy)^2$	$100z^2x^2$
Diameter of Wire : decimals of an inch Y =	—	$\frac{10z}{\sqrt{a}} - z$	$\frac{10-\sqrt{a}}{10\,x}$	—	—	$\frac{1-xz}{x}$
Diameter of Mesh Aperture : in decimals of an inch Z =	$\frac{y\sqrt{a}}{10-\sqrt{a}}$	—	$\frac{\sqrt{a}}{10\,x}$	—	$\frac{1-xy}{x}$	—
Number of Holes per lin. in. X =	$\frac{10-\sqrt{a}}{10y}$	$\frac{\sqrt{a}}{10z}$	—	$\frac{1}{z+y}$	—	—

The diameter, in decimals of an inch, of a steel wire is approximately $y = {\cdot}00315\sqrt{w}$, where w = the weight in milligrams per lin. inch of the wire.

by two consecutive screens, vary in the same ratio as the grades of the theoretical cubes.

For the sake of completeness the relations existing between the factors determining wire woven screens are given in Table C. (see p. 384).

5. The absolute values of the true average sizes are, of course, smaller than the theoretical cubes, in a proportion which varies with the physical nature of the ore, method of crushing, shape of mesh hole, etc. As far as relative values only are concerned we have not, however, to trouble about their exact relative place in the scale, and we may take the cubes of the respective grade as representatives.

6. Over nine grades, from 1 in. mesh down to 30 mesh (·0166 in. aperture), the true average weight of the particles of each grade has been ascertained, and the ratio of decrease in weight of these particles proved to correspond remarkably well with the correct reduction scale of the theoretical cubes, especially in consideration of the fact that the measurements had to be executed with the product coming from an ore breaker, containing consequently many splinters and flat shaped pieces, not amenable to perfect classifying by size. The relation of the true average size of the particles to the corresponding theoretical cubes representing the mean grade between the grades of the screens, varied in the nine tests between 58·3% and 66·9%, and the average of 61·4% can therefore be considered as fairly representative for the Rand ore and for a product coming from ore breakers.

7. With this factor the values, (as E.U., etc.,) of the larger pieces remaining on the 1 in. screen, which would offer difficulties to classifying by screening, can easily be determined, by using the formulæ of Table B, from the average weight of pieces of approximately equal sizes, of divers groups, into which the material to be examined is first assorted by hand.

8. The average value of screen products determined by two consecutive—possibly far distant—screens, are those of the grade having a number which is the arithmetical mean of the ordinal or E.U. numbers of the two screens, and therefore corresponding to those obtained by the application of E. J. Laschinger's formulæ ($_9$).

9. The *product passing the finest screen* used for grading, may by practical experience, be assumed to be represented by the following values :—

Average Product Passing.	Ordinal or E.U. Number.	Corresponding Mesh Aperture.
200 m (·00246 in.) screen	28	·00155 in.
90 in. (·00620 in.) screen	27	·00125 in.

For the estimation of the average grade of the *remainder on the coarsest screen*, the battery screen used gives, in most cases, valuable information concerning the size of the coarsest particles, allowing to take the mean as above. The *amount* of the percentages may also be used as a guide in this direction, as a large percentage indicates evidently a coarser average grade than a small percentage, as in the case of final pulps, where the average grade is bound to be very near to that of the coarsest screen used.

10. The number of grades obtained by this scale at the crushing ratio 1 : 2, being too great for general use, each alternate grade only has been retained, and this restricted set determines therefore a reduction scale at the ratio 1 : 4.

11. The present scheme, as will be noted, standardises *not the screens, as such*, but establishes an unerring standard for the sizing and classifying of the screen products into *grades*. It is consequently valid for all screens of any description, and completely independent of purely practical and commercial considerations, which have to guide millmen and manufacturers for meeting the varying requirements in practice.

12. The standard grades apply to screens in practical use as well as to laboratory screens, and the only difference to be made is, that for the latter, as instruments of measurement, a high degree of accuracy must be required. In this regard commendable efforts have been made by the I.M.M., resulting in the creation of a specially made set of standard laboratory screens, which individually are most accurate with regard to mesh apertures and uniformity of discharge area, but, in part do not fill the requirements of the system of classification which I advocate, for reasons given below.

13. Summarising the advantages of my system of standardisation of grades, they are :—

(*a*) No really new departure from standard practice.

(*b*) Equality of steps with regard to reduction in volume.

(*c*) A sufficient number of grades.

(*d*) Independence of wire gauge, and therefore, immediate applicability to screens of any description, with round or square holes.

(*e*) The standard *grades* are exactly the same for screens in practical use in metallurgical operations as for laboratory purposes, whilst the *screens* may vary in accordance with the gauge of wire used, with degree of accuracy in their manufacture, etc.

(*f*) No limit to extension in either direction, or by interpolation.

(*g*) The variations of all other functions of the particles, besides the diameter, can be expressed by equations, or plotted out in regular curves.

(*h*) Any odd screen of which the clear aperture is known, can readily be inserted in the scale by the use of the formulæ given in Table B.

(*i*) The ordinal numbers represent at the same time the relative value of the energy required to produce the respective grade from the unit.

(*j*) The average values of screen products determined by possibly far distant screens are readily obtained.

(*k*) Any projected standardisation of battery screens can easily be made to fall in with the present scale.

(*l*) The so-called 60 and 90 mesh screens (aperture ·01 in. and ·006 in. respectively) are already in general use on the Rand.

Accuracy.—

14. The measurement of the size of particles by grading with screens can never claim to be scientifically exact, as too many factors of inaccuracy have to be reckoned with, as for instance, the limits of accuracy in the manufacturing of the screens, the alterations in their apertures by wear, and the fact that the actual maximum size of the particles passing through any given screen varies with different ores, different grinding systems, and different methods of conducting the grading analyses, etc.

15. Even if it were possible to bring the measurement by screening to a high degree of perfection, it would be futile to pretend to excessive refinements, as the working conditions of milling plants are, in practice, subject to unavoidable fluctuations, and the samples subjected to grading are only more or less representative averages of mill products, varying within considerably wide limits.

16. For scientific and accurate work the full set of M.T.C. grades should be used, but for the daily control of operations on the mines, the 60 and 90 M.T.C. grade (50 and 80 mesh I.M.M. screen) are quite sufficient.

17. Considering the above-mentioned factors of inaccuracy, it is unnecessary to work closer than to the first place of decimals in the percentage ratios.

Methods of Conducting Grading Analyses.—

18. Although the wet method may give more accurate results, the dry method is for practical reasons almost exclusively in use on the Rand. If a sufficient number of screens be taken, a displacement of even a few percentages from one grade to the next adjacent one, will not appreciably affect the result. More care must be taken with regard to the slime, which adheres to the coarser particles, and if the operation of screening be not prolonged over a very long time, a displacement of about 2 takes place from the very finest to the coarser particles. This inconvenience can, in practice, be overcome very satisfactorily by adopting the *mixed method*.

19. A convenient weight of the sample is dried and then the sands are washed in a suitable circular metal frame having the finest screen it is proposed to use, fitted into its bottom. The washed product freed from the finest grade is dried in the same screen-frame, and the grading is then continued and finished on the re-dried sand.

20. The following advantages are claimed for this mixed method:—

(*a*) Greater accuracy.

(*b*) Saving in time in spite of the double drying, as with washed sands, in a comparatively short time a clearly defined end of the operation of screening is reached.

(*c*) No dusting.

21. For the drying of the sand, steam or waterbaths should be used, as too great a heat in drying the sand causes the bursting of about 2% of the coarse particles, the splinters going to increase the + 120 grade especially.

22. The practical limits of manufacturing are reached with the 200 mesh (·0025 in. aperture) screen, and for further classification the metallurgist will have to resort to the elutriation method by which the particles are, in a current of water of varying velocity, classified on the basis of their hydraulic value. In consequence of the influence of the varying shape and specific gravity of the particles, this method gives, with regard to the *volume* of particles, no clearly defined grades, but on the other hand, the gradings are more in accordance with practical working conditions. The portion to be examined by elutriation, must be separated from the wet sands before they are dried, in order to avoid any alteration, due to heat, of the particular physical properties of the slimes.

II.—Application of Grading Analyses.

Theory.

23. The area of fracture over which the cohesion of the molecules has to be destroyed, multiplied by a coefficient determining the resistance which the molecules oppose to their separation by the exercise of any stress (crushing, tensile, shearing, etc.) represents the *force* required to cause the fracture.

24. In order to perform mechanical work, this force has to run through a *distance*, represented by the deformation which the body can stand before reaching the breaking point. It is in this connection immaterial that this distance of deformation within the limits of elasticity and plasticity is, for not perfectly homogeneous bodies, subject to variations, which for highly elastic and unplastic bodies as quartz, glass, etc., are too insignificant to be considered, and in addition they are by the nature of our crushing appliances averaged to such an extent, that these averages are as good as exactly defined figures. Dealing with relative values only, we have not even to care about the exact extent of this deformation, and all that we need, is to be satisfied, that this factor is a constant function of the diameter of the particles to be crushed.

25. The *mechanical work done* is represented by the product of the *force by the distance*, but as in a regular scale of reduction by volume the diameters of the particles decrease at the same ratio as the area of fracture increases the product, or the mechanical work required for reducing the volume (or weight) of the unit from one grade to the next following, *is a constant for each grade*, called the crushing or energy unit (E.U.).

26. The same conclusion is also arrived at by the application of "Kick's Law," which reads :—

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

The volumes of the particles decrease from grade to grade at the same ratio as the number of the particles, constituting in their total the volume of the unity, increases, and the product of the volumes into the number of the particles of that grade is, therefore, constant for each grade. As in conformity to the above law, the amount of energy absorbed is proportional to the volume of the body to be crushed, it follows again also that the total energy required for reducing the weight of the unit, *is constant for each grade.*

27. The ordinal numbers of any arithmetical progression given to these grades represent consequently the relative values of the energy which has to be spent upon producing this respective grade from the initial unit, or the *mechanical value* of the grade.

28. For obtaining the *mechanical value of mixed sands*, we need only to multiply the percentages of the gradings by the number of the energy units of the respective grade and add the products.—This possibility of having the grading of pulps condensed and expressed in *one representative figure* proves to be of great value.

29. The *useful work done per unit* by any crushing machine is determined by the difference between the mechanical values of the samples, taken at the inlet and the discharge of the machine, and for obtaining the *total work done* this difference has to be multiplied by the tonnage dealt with.

30. The *relative mechanical efficiency* is the value obtained by dividing the total work done by the unit of energy (for instance, H.P.) :

Relat. mech. efficiency =

$$\frac{\text{Tonnage} \times \text{work done per unit in E.U.}}{\text{Unit of energy (H.P.)}}$$

Examples of efficiency calculations (Rand ore)

31. *A. Stamp.*—

Battery screen : 64 mesh 0·097 in. aperture.
Running weight : 1,150 lbs. (energy 1,174 lb. ft. secs.)
Power consumption : 2·6 H P.
Duty : 9 tons per 24 hours.

Mines Trials Committee. Standard Grades.			Feed.	Discharge Pulp.		
Denomination of Screen.	Aperature.	Mech. Value of mean grade.		Grading Analyses.		Mechanical Value of Discharge.
No.	inch.	EU				E U
+ 12	·03937	13	Subject to variations, but by practical experience it has been ascertained that the O grade (1 in. screen) represents a fair average size.	19·9	+60	2·59
+ 20	·02480	15		12·1	53·2	1·82
+ 30	·01562	17		9·7	+90	1·65
+ 50	·00984	19		11·5		2·19
+ 80	·0062	21		10·9	11·5	2·29
+ 120	·00391	23		2·6	−90	0·50
+ 200	·00246	25		4·2	35·9	1·05
− 200 incl. washed slimes.	·00246	28		29·1		8·15
				100		20·34
			less Mech Value of Feed			0·0
			Work done by Stamp per unit (say 1 ton)			20·34

Total work done p. Stamp in 24 hours = 9 tons × 20·34 EU = 183·06 EU

Relative mechanical efficiency p. H.P. $= \frac{9 \text{ tons} \times 20{\cdot}34 \text{ EU}}{2{\cdot}6 \text{ H.P.}} = 70{\cdot}40$

B.—Tubemill.

Dimensions 22 ft. × 5′ 6″ diam.
Banket Ore-feed 6·75 tons p. 24 hours.
Speed : 30 revolutions p. min.
Power consumption say 100 h.-p.
Duty : 290 tons p. 24 hours.

Mines Trial Committee Standard Grades.			Intake.			Discharge.		
Denomination of Screen.	Aperture.	Mech Value of Mean Grade.	Grading Analyses.		Mechanical value of Intake.	Grading Analyses.		Mechanical value of Discharge.
No.	Inch.	E.U.			E U.			E.U.
12	·03937	13	27·6	+60	3·59	—	+60	—
20	·02480	15	15·7	76·9	2·36	—	29·9	—
30	·01562	17	13·7		2·33	7·4		1·26
50	·00984	19	19·9	+90	3·78	22·5	99·0	4·28
80	·0062	21	15·5	15·5	3·26	29·0	29·0	6·09
120	·00391	23	2·0		0·46	6·0		1·38
200	·00246	25	1·6	−90	0·40	7·7	−90	1·93
200 including washed slimes.	Do.	28	4·0	7·6	1·12	27·4	41·1	7·67
			100		17·30	100		22·61
			Less Mechanical value of Intake					17·30
			Work done per Unit, say 1 ton					5·31

Total work done by Tubemill in 24 hours :

Reduction of Pulp feed =
290 tons × 5·31 EU. = 1539·2 EU.

Reduction of Banket ore =
6·75 tons × 30 EU. = 202·5 „

Total work done in 24 hours ... 1742·4 EU.

$$\textit{Relative mech. efficiency} \text{ p. HP.} = \frac{1742{\cdot}4 \text{ EU}}{100 \text{ HP.}} = 17{\cdot}42$$

32. Under the conditions of the above examples, taken from practical work, the efficiency per HP. of Tubemills (17·42) is only about $\frac{1}{4}$ that of the stamps (70·40).

33. For general guidance some instructive figures of mechanical values might be of interest :

Battery pulps varying with the increase of the fineness of the screening from 17·0 EU to 24·5 EU.

Tubemill intake depending upon more or less perfect removal of the finest particles by the classifiers two to three EU. less.

Tubemill discharge, varying with working conditions, especially coarseness of feed, from 21·0 EU to 24·5 EU.

Final (cyanide) *pulp* varying between the following extreme limits :—

	+60	+90	−90	
coarse pulp	15%	20%	65%	= 24·45 EU.
fine pulp...	3%	10%	87%	= 26·13 EU.

34. The difference between the mechanical value of the Final (cyanide) pulp and the Battery pulp represents the work done per unit (say 1 ton), by the secondary crushing appliances, and their total work done, *i e.*, the sum of the products :

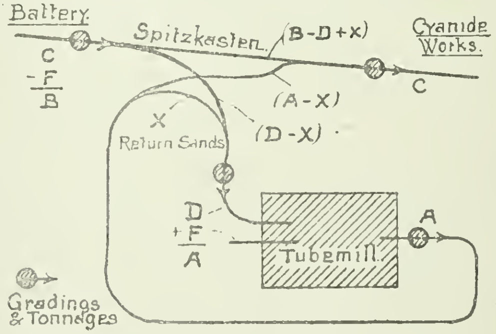
Tonnage × (Mechanical value Discharge minus Mechanical value Inlet)

of the single units must therefore be equal to

Battery tonnage × (Mechanical value Final pulp minus Mechanical value Battery pulp.)

From this fact the correctness of the measurements can readily be checked or inversely the efficiency of the secondary grinding machines can be controlled by the difference of the Mechanical values of the Final and the Battery pulps, of which reliable average samples are taken in any case for assay purposes.

35. The great advantage connected with my method of expressing the values of the gradings in *one* representative figure is that it allows of the solution of many interesting problems : for instance, the redistribution of the battery and tubemill discharge pulps, which in the spitzkasten become mixed up, can by calculation be exactly analysed into the amounts delivered to the cyanide works or returned to the tubemill for regrinding.



X = tons per 24 hours tubemill return sands.
A = tons per 24 hours total sand and banket feed.
B = tons per 24 hours total net battery duty.
C = tons per 24 hours total ore milled.
D = tons per 24 hours tubemill sand feed only.
F = tons per 24 hours banket-feed only.
c = Difference mech. values of cyanide and battery-pulps.
e = Difference mech. values of tubemill intake and cyanide pulp.
h = Difference mech. values of tubemill intake and discharge pulps.
g = Mech. value represented by reduction of banket pieces to size of cyanide pulp.
i = Mech. value represented by reduction of banket pieces to size of tubemill discharge pulp.

The mechanical work done in tubemills being determined

(1) By difference of work done represented in Intake and discharge pulps $= Dh + Fi$.

(2) By difference of work done represented in the portion of the tubemill discharge delivered to the cyanide works and the tubemill inlet pulp $= (A - X)\frac{De + Fg}{A}$

(3) By difference of work done represented in Cyanide and Battery pulp $= Bc + Fg$ we obtain by equalising the above values for instance for the amount of the return sands :

$$X = A - \frac{(Dh + Fi)\ A}{De + Fg}, \text{ or, } X = A - \frac{(Bc + Fg)A}{De + Fg}$$

By neglecting the Banket ore feed the formula would become reduced to :

$$X = A - \frac{Dh}{e}, \text{ or, } \qquad X = A - \frac{Cc}{e}$$

36. The energy absorbed in the reduction of the Banket ore feed to the size of the Final pulp can as an average be estimated at 30 EU. (26 EU. being the mech. value of the final pulp with reference to the initial unit, plus 4 EU. representing the energy absorbed for the reduction of 4″ pieces to the size of the initial unit or O grade.) For 22 ft. tubemills, as in general use on the Rand, the work done in reducing the Banket ore represents, depending upon the working conditions, 5 to 25% of the total work done.

37. The mechanical values of the pulps may in many cases, where a great accuracy is not required, be calculated from the gradings of the three grades only, determined by the usual screens Nos. 60 and 90. This simplified calculation compares for inst. forabove tubemill example as follows :—

Grade.	INTAKE.			DISCHARGE.		
	Wghts.	Mech. value of Mean Grade.	Mech. value of Intake.	Wghts.	Mech. value of Mean Grade.	Mech. value of Discharge.
	%	E.U.	E.U.	%	E.U.	E.U.
+60	76·9	15·5	11·92	29·9	18	5·38
+90	15·5	21·0	3·26	29·0	21	6·09
−90	7·6	27·0	2·05	41·1	27	11·10
	100		17·23	100		22·57

CONCLUSION.

38. All through the experimental work, executed on behalf of the M.T.C., it could be ascertained that this method of efficiency calculation is for fine as well as for coarse crushing, correct and reliable, and in perfect accordance with the results of practical experience. Based on these facts it can be stated *that we are now in a position to determine with a comparatively high degree of accuracy the relative merits of different crushing appliances, or the mechanical efficiency of one and the same machine working under varying conditions.*

39. In the author's reply to the discussion on his paper on Heavy Gravitation Stamps, Dr. Caldecott comes forward with a method of his own for computing efficiency, in tacit opposition to mine, the merits or otherwise of which, as an active and interested member of the Mines Trials Committee, he has had under close observation for more than a year. The discussion on his paper being closed, I may be allowed, under these circumstances, to take the controversy up in the present paper. In order to stimulate Dr. Caldecott to redouble his efforts to discover flaws in the correctness and reliability of *my* method, I begin by saying squarely and in plain terms, that *his* method is not only unscientific, but also inaccurate. All the figures obtained by, and the conclusions drawn from it, are incorrect, misleading, and therefore worthless.

Dr. Caldecott first finds it very convenient to introduce a new "nominal crushing unit." This he defines as a stamp of an arbitrary description as regards weight, drop, and number of drops per minute. With no reasons given, it is difficult to see the desirability or necessity of introducing a new unit for the measurement of the energy used by different stamps when such units as ft.-lb., sec. or h.p are already available and easily understood. This device is so much the more needless as the value in normal crushing units of any other stamp of different weight or working conditions is to be determined by the ratio of the ft.-lb. sec. or h.p, of the particular stamp to that established as the normal crushing unit. There is, therefore, no reason why the usual units should not be taken as a standard. It is then further arbitrarily assumed that a tubemill of assumed dimensions, with an assumed power consumption is equal to 30 such nominal crushing units. The admission that this ratio "varies somewhat according to the conditions of operating" is appropriate, provided that the limit of elasticity of the term "somewhat" be made wide enough to stand the enormous strain of the very great effect of these operating conditions on the efficiency of tubemills of equal cubical capacity inside shell.

I do not lay too much stress on the above points, which are only "somewhat" inaccurate ; but a very serious error is made by taking the production of the −90 grade as a standard of the work done in crushing. By the use of his theory, which Dr. Caldecott bases on his finding that "the more tons of this material that can be obtained per stamp, or per tubemill, or per horsepower, the better," undue credit is given to the "finishing" work performed by tubemills on an already very fine feed; whilst the hard work done by other crushing appliances in reducing the coarse particles to the size of this feed is completely ignored. By similar reasoning Dr. Caldecott might have said that the ultimate object sought is the reduction of the +60 grade and in this case he would have come to just the opposite conclusion, namely, that practically all the work done in crushing is performed by the preliminary crushers and very little is left for the tubemill to do. The fact that the figures obtained by his method do not show any advantage in favour of coarse crushing will perhaps shake his confidence in his method more than any possible criticism. Indeed, according to his figures, the production of the −90 grade is, for the same stamp weights, about the same for any coarseness of battery screens, whilst by my method, which never fails to accord perfectly with practical experience, the greater efficiency claimed for coarse crushing stamps is clearly shown by the figures obtained. They are for a stamp of 1,250 lb. running weight, as follows :—

Mesh and Aperture of Battery Screen. Mesh.	Mesh and Aperture of Battery Screen. Inch.	Duty per 24 Hours. Tons.	Mechanical Value of Pulp. E.U.	Relative Mechanical Efficiency per H.P.
9	·272	13·0	18·0	234·0
200	·053	7·5	21·5	161·3
600	·028	6·5	23·2	150·8
1,200	·017	5·2	24·3	126·4

40. In conclusion, I wish to record my appreciation of the courtesy of the Mines Trials Committee in granting permission to use data obtained in the course of my investigations whilst in their employ, but I am personally responsible for the mathematical reasoning and considerations derived therefrom together with the opinions expressed.

Besides many minor articles and contributions scattered in technical publications—too numerous to be quoted—especially the following literature has been consulted on the subject :—

1. The relative chapters in well-known mining books.
2. I.M.M. London. *Bulletin* No. 38, 1907.
3. Report of Sub-committee on the Standardisation of Battery Screening. Supplement to the *Journal of the Ch. M. M.*, Johannesburg, June, 1906.
4. C. de Kalb, *Engineering and Mining Journal*, July, 1905.
5. E. T. Henderson, *Australian Mining & Engineering Review*, November, 1908.
6. W. McDermott, Standardisation of Screens. Crowther and Goodman, Printers, London.
7. W. A. Caldecott, this *Journal*, November, 1905.
8. Philip Argall, *Mining and Scientific Press*, 1908.
9. S. H. Pearce and W. A. Caldecott, this *Journal*, Sept., 1906, and contributions from H. A. White (Oct., 1906); Prof. R. H. Richards (Feb., 1907); E. J. Laschinger (Jan, 1907), etc.
10. Arthur Yates, this *Journal*, Dec., 1908, and contributions from H. Stadler (April and Aug., 1909), H. A. White (May, 1909).
11. Harry S. Denny, *Journal S.A.I. of E.* . and contributions by Messrs. G. E. Busch, J. A. Vaughan, etc.
12. G. A. and H. S. Denny, *Journal S.A.I. of E.*, June, 1908.
13. H. Stadler, *Journal S.A.I. of E.*, Dec., 1908, and Feb., 1910, etc.

Mr. H. A. White (*Member of Council*): I wish to propose a very hearty vote of thanks to Mr. Stadler for this very able and interesting paper, which represents not only a great deal of thought but a large amount of work. His method of substituting ideal grades for the mechanically imperfect screens, actually used, seems a step in the right direction. It is obvious to all present, who are interested in this question, that the author's theoretical results are entirely based on Kick's theory, which has not yet received universal assent. The conclusion arrived at on this basis which denies the tube mill an efficiency greater than a fourth that of the stamp mill, and shows a doubled efficiency for the latter when 9 mesh is used instead of 1,200, can scarcely be said to "accord perfectly with practical experience."

I think some little allowance is due to the tube mill upon a consideration of the fact that it has to work on particles which the battery discriminates against, and debit must be made against the battery for that portion of the material it deals with, which forms the clayey portion of the slime produced, as this must be not only more easily reduced, but must cause planes of weakness in the rock to be crushed.

It may be considered, however, that the law which refers work done to area of surface exposed has the advantage in our present circumstances, where increase of this area is the desideratum for further extraction of the precious metals. This increase of area is acknowledged to be obtained at a cheaper rate in tube mills, and so a greater proportion of the work is profitably thrown upon them when larger screens allow a greater capacity for the battery. But if the large majority of our engineers are really at fault in adopting the tube mill, and if it be really as mechanically inefficient as Mr. Stadler's deductions show, I am at a loss to understand why the author has not gone a little further and proposed stamps for his secondary grinding, and shown us how to save three-quarters of the power, now wasted in the tube, by that or some similar means.

It is certainly to be hoped that so valuable a paper as this may meet with a full and vigorous discussion.

Mr. E. H. Johnson (*Past President*): I would like to second the vote of thanks to Mr. Stadler for his interesting paper. Mr. Stadler has devoted himself very closely for some time to the study of a method for deducing relative efficiencies of crushing appliances from the grading results, and his paper forms an interesting addition to the literature of the subject. The translation of grading results into the horse power required to produce them is somewhat difficult with the local conglomerate ore owing to the different resistances to sub-division offered by the varied constituents of the ore. For instance, using a screen mesh on the stamp mill of 0·27 in. dia., there is produced approximately 26% of material passing a screen of ·006 in. aperture, which apparently requires little mechanical energy to produce. This and similar observed facts do not coincide with the "equal technological state" required to render Kick's law perfectly applicable.

EXPERIMENTS ON THE PRECIPITATION OF GOLD FROM CYANIDE SOLUTION BY CARBON IN LIME.

By Edward H. Croghan (Member of Council).

The subject matter of the following contribution was first mentioned at our meeting of April of last year, when Dr. Caldecott gave a short note on "the precipitation of gold from solution by carbonaceous matter," and at that time I also touched upon the subject. In view of the statement then made it would seem that this half burnt coal or carbon, as some prefer to name it, might have a deleterious effect in precipitating a small quantity of gold during the cyanide treatment. At the above mentioned time our late deceased and most esteemed member, Mr. Wager Bradford, asked certain questions which may be classified as follows:—

1. What is the maximum percentage of carbon which may be present in lime without causing precipitation of gold in the treatment vats?

2. Are there any figures to prove that 0·1%, for example of carbon in the lime used, results in the precipitation of gold to any appreciable extent?

3. Is it possible that a very small percentage of carbon in a lime may be present without exerting any deleterious influence?

4. Is it certain that the results of the laboratory tests will be obtained in a 500 ton vat; in other words, will working results confirm laboratory experiments?

Dr. Caldecott, to whom these questions were put, gave a reply the principle of which was, that even if only a trace of gold were precipitated the value of this would represent thousands of pounds on the tonnage of ore yearly milled on the Rand, and therefore if it is possible to obtain carbon-free limes only those limes should be employed. It was practically impossible to make any deductions from working results as the limits of experimental error of sampling and assaying would probably be exceeded. This then briefly sums up the position to date.

In view of the fact that metallurgists still disagree on the matter, I decided to carry out some experiments. These trials were made in such a way that if results were positive it might be possible to make some deductions and establish a guide for specifications, by noting the effects of different percentages of carbon, and in accordance with the quantity of lime used per ton of ore by any particular mine, so enabling a standard or maximum allowance to be fixed.

The details of the experiments are as follows:

1. *Reagents: Gold Cyanide Solution.*—This was prepared by making first a very strong solution by dissolving a known quantity of gold in a minimum amount of pure potassium cyanide and diluting with tap water until it had an assay value of about 1 dwt. per ton. A weighed amount of potassium cyanide was then added to give a strength of about 0·1% and filtered lime water to act as protective alkali to about ·004%. This was covered and allowed to stand over-night. The next morning the clear solution was tested, and carefully decanted from any precipitate into glass stoppered Winchester quart bottles. The clear solution thus obtained, when tested gave the following results:—

Free Potassium Cyanide = 0·106% KCN.
Protective Alkali as Lime = 0·0036% CaO.
Assay value = 1·12 dwt. per 2,000 lb.

2. *Carbon.*—An indifferent quality of lime was secured. 100 grammes were mixed with a litre of tap water and treated carefully with small additions of strong hydrochloric acid until most of the lime dissolved, the solution being kept as cool as possible. The dissolved lime was decanted and the residue thoroughly washed by decantation and dried at 100°C. Fortunately some large pieces of carbon were present. These were carefully selected, crushed to 60 mesh and the carbon determined, by difference, from the ash contained. It was not considered necessary to adopt a combustion or chromic acid procedure. The result gave 60% carbon.

3. *Experiments.*—Ten experiments were made. The maximum amount of lime that might be applied for any ordinary class of ore was taken at 10 lb. per ton, and limes containing intermediate percentages from 0·5% carbon down to less than 0·01% were considered. In each case 500 c.c. of the gold cyanide solution was used. These quantities were placed in wide mouthed glass stoppered

bottles of about 1 litre capacity, which had been carefully cleaned. The 60% carbon was then added in various quantities, as shown in Table 1.

The bottles were stoppered and the mixture shaken for 1 minute each, left in contact for 18 hours and then filtered through first quality filter paper. The carbon residues were all transferred to the filters, and washed until perfectly free from any trace of cyanide. These were then carefully dried, burnt in new scorifiers, and scorified with 20 grammes of gold free lead, a very small quantity of silver being added at the same time. The assays were proceeded with as usual, adopting every precaution in view of the extremely small amount of gold expected. The gold obtained in each case was subsequently mounted on a slide. The following table briefly shows all details and results at a glance :—

Table 1.

Lime per Ton of Ore lb.	Carbon in Lime %	Solution Employed. Potassium Cyanide %	Protective Alkali CaO %	Assay Value dwt.	Quantity c.c.	Weight of Carbon added 60% value mgm.	Weight of Gold Precipitated, mgm.	Remarks on Slides.
10	0·45	0 106	0·0036	1·12	500	18·5	0·01	1 large speck
,,	0·34	,,	,,	,,	,,	14 0	,,	,,
,,	0·25	,,	,,	,,	,,	10·5	,,	,,
,,	0·17	,,	,,	,,	,,	7·0	,,	,,
,,	0·085	,,	,,	,,	,,	3·5	,,	4 specks
,,	0·068	,,	,,	,,	,,	2·8	unweighable	1 speck
,,	0·051	,,	,,	,,	,,	2·1	,,	,,
,,	0 034	,,	,,	,,	,,	1·4	,,	,,
,,	0·017	,,	,,	,,	,,	0·7	,,	,,
,,	0·008	,,	,,	,,	,,	0·35	minute and doubtful	undiscernable

It would have been interesting to try and observe if differences in the gold from 0·068% carbon downwards could be ascertained on a micrometer scale, but I do not think this would prove satisfactory on my slides, which do not pretend to be masterpieces.

In the following table, No. 2, I have calculated out the weights of carbon per ton (2,000 lb.) and also the corresponding carbon percentage from the amount contained per ton (last column), assuming for convenience practically the next highest figure for the carbon percentage as compared with the percentages taken in the experiments. This can be safely applied to the data obtained. This throws a little more light on the subject. The table is useful, for by utilising it in conjunction with the gold precipitated as per table I., any mine wishing to control the carbon factor can fix its own standard in accordance with the amount of lime used per ton.

Table 2.

Lime per Ton of Ore, lb.	Carbon in Lime, %.	Carbon per Ton of Ore lb.	Carbon per Ton of Ore oz.	Carbon per Ton of Ore, grains	Carbon per Ton of Ore grammes.	Carbon per Ton of Ore %
10	0·5	0·05	0·8	350	22·680	0·0025
,,	0·4	0·04	0·64	280	18·144	0·002
,,	0·3	0·03	0·48	210	13 608	0·0015
,,	0·2	0·02	0·32	140	9·072	0·001
,,	0·1	0·01	0·16	70	4·536	0·0005
,,	0·08	0·008	0·128	56	3·629	0·0004
,,	0·06	0·006	0·096	42	2 722	0·0003
,,	0·04	0·004	0·064	28	1·814	0·0002
,,	0·02	0·002	0·032	14	0·907	0·0001
,,	0·01	0·001	0·016	7	0·454	0·00005

From a study of the actual percentage of carbon from the tables given it would be easy to devise others suitable to varying conditions. From a general point of view these show that even a lime with ·5% carbon indicates only traces of carbon per ton of ore, and should play no important part. I state this because it will seem that the figures indicate that matters are being carried too fine. The necessity of having carbon-free lime, or as nearly carbon-free as possible, is a matter for discussion and settlement by those more competent to judge especially in studying figures on a large scale.

In my final deductions I desire to refer again to Table 1, and on the basis of 3 lb. of lime per ton of ore required on the Rand indicate what seems to be the maximum percentage of carbon allowable for improving even this detail. In doing this it will be quite legitimate to accept the carbon percentage as in previous tables. It will not affect the data obtained or conclusions suggested.

By a study of table 3 and considering what lime burners can reasonably control, the following may be stated for the Rand on the foregoing basis of 3 lb. lime per ton. Limes containing

a. 0·5%, 0·4%, 0·3% carbon should not be accepted.

TABLE 3.

Carbon in Lime.	Lime per Ton of Ore, lb.	Carbon per cent. of Ore.	Weight of Gold Precipitated mgm.	Lime per Ton of Ore lb.	Carbon per cent. of Ore.
0·5	10	0·0025	0·01	3	0·00075
0·4	,,	0·002	,,	,,	0·0006
0·3	,,	0·0015	,,	,,	0·00045
0·2	,,	0·001	,,	,,	0·0003
0·1	,,	0·0005	,,	,,	0·00015
0·08	,,	0·0004	unweighable	,,	0·00012
0·06	,,	0·0003	,,	,,	0·00009
0·04	,,	0·0002	,,	,,	0·00006
0·02	,,	0·0001	,,	,,	0·00003
0·01	,,	0·00005	minute and doubtful	,,	0·000015

b. 0·2% carbon is open to question.
c. 0·1%, ·08%, 0·06% carbon might be accepted.
d. 0·01%, 0·02% carbon should be satisfactory.
e. Less than 0·02% carbon can be taken as excellent.

The preceding remarks are in a sense difficult to deduce, but possibly others have data and I trust will go fully into the matter. I am not a lime burner, but I do not think we ought to make their conditions too stringent. On the other hand I have investigated numerous samples of limes and noted that they hardly ever exceed 0·2% carbon.

The endeavour now-a-days is to make the cyanide gold extraction process perfect by trying to approximate as near to ideal conditions as possible, and as every little helps I hope this paper will be amply discussed, with particular reference to the questions put forward by the late Mr. Wager Bradford.

The President: It affords me great pleasure to-night to propose a hearty vote of thanks to Mr. Croghan for the excellent contribution he has just read. Most of you know that I am specially interested in drawing up specifications, and some of you will remember that in a previous paper I stated that the buyer should always be willing to listen to what the seller has got to say. That point I wish to emphasise to-night, and Mr. Croghan therefore affords me an opportunity of seeing what a man who is not interested in either side is willing to bring forward as a basis for our acceptance. There is one thing I can say from my own experience, that is I can corroborate the last paragraph of Mr. Croghan's proposals where he says "less than 0·02% carbon can be taken as excellent." That I do agree with. The other remarks I do not wholly accept, but I may take an opportunity of discussing them later. One point I would bring forward which Mr. Croghan introduces, and that is his method of assaying this carbon on which gold has been precipitated. He says "These were then carefully dried, burnt in new scorifiers, and scorified with 20 gm. of gold free lead, a very small quantity of silver being added at the same time." That I am not quite in agreement with. I am a little bit afraid that Mr. Croghan by using that method would be losing gold in a volatile form. I mention this so that Mr. Croghan may have an opportunity of trying it himself. I do not wish further to criticise the paper to night, so I will ask you to extend to Mr. Croghan a hearty vote of thanks for the contribution he has given us.

Mr. J. H. Johnson (*Member*): I have very much pleasure in seconding the vote of thanks. There is one question I should like to ask Mr. Croghan and that is whether he has experimented to prove that soot and volatile matter has the same precipitating effect as half burnt coal or coke would have on the precipitation of gold?

Mr. E. H. Croghan: I have not made any experiments.

Mr. A. Salkinson (*Member*) suggested that a discussion should be raised as to the deleterious effects of organic matter coming from underground.

THE SMELTING OF GOLD PRECIPITATES AND BULLION WITH OIL FUEL.

(*Read at June Meeting, 1909.*)

By ARTHUR YATES, M.I.M.M. (Corresponding Member of Council).

REPLY TO DISCUSSION.

The Secretary read the following reply:—

Mr. A. Yates (*Corresponding Member of Council*): The possible use of oil for smelting is not so limited as Mr. A. Thomas' statements might lead one to believe. Oil is relatively cheap when compared with the prices and heating qualities of other fuels, and is exceedingly convenient for transport. Where reverberatory furnaces are used for smelting the heavy outlay on them and the large number of crucibles used is often overlooked. A tilting furnace is inexpensive and capacious. Mr Thomas' criticism of the life of the retorts is in our case to some extent justified, but I would ask him to consider the large amount of selenium present in the precipitates; the selenium is only partly removed by the acid treatment and calcination of the gold zinc slimes and remains to form a highly corrosive matte of selenide of silver. With clean precipitates the retorts serve for eight or nine charges;

an adjacent mine has used them twelve and fourteen times. The addition of silica to the flux helps to save the retorts and improves the bullion, but heavy fluxing requires extra time and materials, and it is a question as to which are the cheaper—fluxes, time and oil, or retorts. The present shape of the retorts does not lend itself to the use of clay liners which would probably effect a saving, but I am pleased to say the makers have offered to study the matter both of shape and manufacture and see what can be done to further increase the life of the retorts.

A SKETCH OF THE SMALL CYANIDE PLANT AS ERECTED AND WORKED IN RHODESIA.

(Read at September Meeting, 1909.)

By F. J. Thomas (Member).

REPLY TO DISCUSSION.

The Secretary read the following reply :—

Mr. F. J. Thomas *(Member)*: There is but one point calling for reply in the light criticism to which my few notes on the above matter were subjected. I think few cyanide men with experience of Rhodesian mines will be able to accept without demur Mr. Johnston's dictum that a slime plant for the treatment of the current product is essential even on a small mine. A glance at the monthly returns, as published in the Rhodesian papers, will show that the duty of many of the small mills is extremely low. Mr. F. P. Mennell in his "Miners' Handbook" states "Two stamps or a Tremain mill crush about 300 tons of ore per month. A five-stamp battery will get through about 750 tons a month." In practice such results are seldom achieved. It is almost impossible to arrive at the actual daily duty of the small mills as the length of the month's run is not published, and there must be taken into consideration many serious stoppages, some permanent, others temporary, due to lack of labour, water, fuel and to various breakdowns. In the returns of the mineral output of a recent month of 31 days I note that only 13 out of 54 five-stamp mills crushed over 600 tons. The average tonnage of 60 mills of less than five stamps was 173 tons, whilst the average tonnage of the two-stamps and Tremains, taken together, was less than 150 tons.

Discounting rubble properties, which, as Mr. Murdoch points out in his contribution to the discussion, usually produce a large percentage of very low-grade slime, the quantity of slime obtained daily from the majority of mills of five stamps and less would vary from about 6 tons down to 2 or 1½ tons. To erect a plant for the current treatment of such a tonnage offers no great inducement to the small owner.

FURTHER NOTES ON RAND MINING.

(Read at February Meeting, 1910.)

By Tom Johnson (Member).

DISCUSSION.

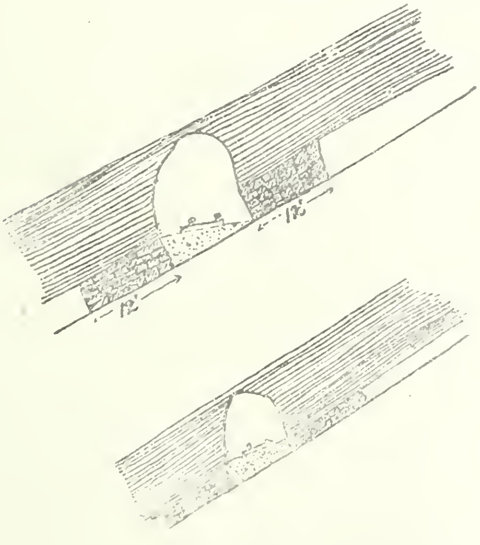
The Secretary read the following contribution :—

Mr. Rowland Gascoyne *(Member)*: It is somewhat disappointing that such an excellent paper as that read by Mr. Tom Johnson should be allowed to pass without adequate discussion. It may be due to the opinion that, on the whole, it is above criticism, but Mr. Johnson has not failed to criticise Rand methods where he considers them faulty, and I do not think he desires his paper to pass unnoticed, and on this account I venture to offer a few remarks.

Mr. Johnson commences by stating that "in some cases the roof shows weight over large areas crushing the pillars in the stopes, the drive ribs, and sometimes even the large safety pillars of the shafts." This is a somewhat wholesale condemnation of Rand mining methods, and can only be excused where an unusual condition of affairs due to bad ground prevails, and it is obvious that the pillars left to support the hanging wall are far too small, and providing it is a deep level mine it is better not to have left any stope and drive pillars at all than to have left them too small and weak to fulfil their requirements. With regard to the safety pillars of the shafts, there is much difference of opinion. Some engineers in England being of the opinion that in deep mines, where crushing of the shaft pillars are unavoidable on account of depth, it would be better not to leave any shaft pillars at all, but no one as yet has had the pluck to carry out this opinion. Naturally in coal mining this difficulty presents itself at much shallower depths than on the Rand, for beyond a depth of 2,000 ft. in coal mining a shaft safety pillar, no matter what its dimensions may be, becomes a nuisance rather than a help, and soon a source of considerable expense. The writer has managed the deepest mines in England, and in a flat mine where the shaft was 2,200 ft. deep a solid safety pillar, equal in circumference to the depth of the shaft, gave endless trouble. In another mine, where the writer was manager, with a shaft over 3,000 ft. deep, and the seam dipping 1 in 3, with the whole of the rise side of the shaft solid ground, and the safety pillar on the dip side extending for more than half a mile, there was still trouble in maintaining the shaft intact so that there is no wonder that the suggestion to leave no safety pillar whatever for shafts at these depths in coal mining should receive support in many quarters, but, as before remarked, no engineer as yet has been sufficiently brave to support his opinion by practice.

Mr. Johnson says that personally he likes to see incline shafts and drives out of danger. It is an easy matter to have all deep level drives out of danger, but the writer would be glad to hear how Mr. Johnson purposes to place his shafts out of danger say at a depth of 10,000 ft. from the surface.

With regard to the drives illustrated by Mr. Johnson, in Fig. I. the writer quite agrees with the remarks and the steps to be taken under such circumstances, but when mining has reached such a depth to produce such conditions in coal mining, it is found best not to leave any drive pillars at all but make the drive as shown in the following figure, and when the hanging wall has settled and practically met the footwall the drive will present a similar appearance to that shown in Fig. II. The reason why it is considered better to have the permanent road above seam rather than under is to ensure the complete burying of the packs thus adding to their solidity and capability of regulating and steadying the settlement of the hanging wall.



When a drive once reaches the stage shown in Fig. II. there is little trouble, as a rule, afterwards.

It is quite true, as stated by Mr. Johnson, that with a tender hanging wall it is of the greatest advantage to move the face on at the greatest speed so as not to allow the hanging wall to break over, and the member who objects to increasing the number of machines as evidenced in the last discussion of Mr. Johnson's paper, on account of their probable bad effect on the hanging wall, takes too narrow a view, and fails to grasp the situation. It is clear, from Mr. Johnson's remarks, about building a pack with irregular shaped rock, that he has had practical experience of pack building, as only a theoretical miner would ever dream of building one otherwise. Packs should be built so as to offer the greatest resistance to the subsidence of the hanging wall, but anyone who thinks that once the hanging wall begins to subside that it can be stopped by packing is making a huge mistake. This can, at best, only produce a steadying effect.

The object in mines of such a depth as where, for instance, the weight of the superincumbent hanging wall is equal to the resisting power of the reef to prevent crushing, should be to advance the workings as rapidly as possible and get the hanging wall settled in the footwall as quickly as possible, for until that object is attained there will always be trouble.

With regard to the system of breast driving, suggested by Mr. Johnson, I would point out in some mines on the Rand, where conditions are more favourable, it might be more convenient to construct the drive in the footwall rather than in the hanging wall, especially when mining at such moderate depths as are now generally practised on the Rand, but this will, of course, depend upon the underground conditions of the mine. This suggestion of Mr. Johnson to use breast drives is undoubtedly a valuable one for shallow mines, and besides the numerous advantages, pointed out by Mr. Johnson, it has the advantage of evolving a system of working approaching that of longwall, about which so much has been heard lately on the Rand.

With regard to the remarks of Mr. Parry as to keeping the hanging in its proper position only adequate pillars can do so, but depths will soon be reached on the Rand when pillars can no longer sustain the hanging wall, and if mining has to be carried on at great depths a moving hanging wall will have to be contended with, and the difficulties will then be overcome only either by longwall working or by sand filling, or perhaps by the adoption of both these methods when the underground conditions need them.

Mr. M. H. Coombe (*Member*) : I had intended writing a small contribution on Mr. Johnson's paper, but owing to pressure of work I have been unable to do so. I think, however, I may say a few words bearing on the sketch in his paper of a combined heading and stope, and also his assumption that 500ft. a month could be driven by this method. I cross swords with him here and say it is ridiculous to assume anything of the sort, as I think there are many factors militating against even normal driving in a "breast" of this description. In the first place the limit of his speed in "breasting" would be the limit of the speed made in the heading, and the work in the heading would be seriously interfered with by the extra amount of ground broken from the breast on the foot-wall side which would have to be loaded from the rail level. I say nothing of difficulties and cost of shovelling the broken ground *up* to the rail level from the low or hanging wall side. Mr. Johnson evidently overlooks his water troubles from this side also. The drilling and breaking of the ground in a connecting drive is not what takes the time, it is the handling of the broken rock that is the greatest problem in this work, and it appears to me that Mr. Johnson would add to, rather than diminish, this difficulty in this scheme of a combined "heading and breast." His scheme of building a rock pack on the top side of the drive also appears to me to possess several defects. With a reef dipping up to 20° it might be feasible, even then difficult and time wasting, but over 25° he would find it necessary to put in a "stull" to support his "pack," and he would often find a goodly section on his rails blocking his work after blasting in the face. As for bringing rock for this "pack" from other parts of the mine, this proposal seems too ridiculous to be seriously entertained. If he is going to drive 200ft. a month he will find he has his work cut out to keep his roads to the face clear for his "full" and "empties." I think the question of ventilation in these drives can be easily solved by carrying the shaft brattice through the station and cross cut into the drive. For speed and general utility I would carry a connecting drive 10ft. wide and in the centre I would put in light section pole struts, 6ft. centres hitched in front and roof 2ft. apart, lagged with light poles and filled in between with rock broken from the face. This kept to within 50ft. or 60ft. of the face would ensure air circulation and my "tracks" would be laid each side of this brattice, one road for "fulls" the other for "empties." I tried this on a small scale in Rhodesia and found it worked excellently, and I hope to have the opportunity of trying it on a larger scale here at some future time. If I do so and it is successful I will explain it more fully to this Society.

Mr. A. Richardson (*Member of Council*): Mr. Tom Johnson on mining is always interesting because his views are his own, and therefore not those of other mining engineers, and though some of his arguments are, as he admits, old, the subject is still young enough to support a much greater weight of discussion

With his remark that incline shafts should be sunk in footwall country no one who favours safe mining will cavil, but that drives should occupy a similar position is a suggestion with which mining economists are not likely to agree. The information afforded by winzes 700-800 ft. apart, with some box-holing and cross-cutting added, would not be sufficiently comprehensive, as regards values and structural geology, to enable one to draw up a safe estimate upon which to base the future policy of the mine in matters of finance, development, equipment, and so on. There certainly would be no great objection to the driving of an occasional straight haulage level under the reef, but even the driving of this should be deferred until the levels, suitably spaced, immediately above and below it had been driven far enough to furnish reliable data for its correct interpolation.

If the lower Fig. I. is a true reflection of the state of the hanging "where a large area is open" then instead of putting packs in, it might be advisable to take the men out. The maxim that a roof is never safe until it is down is worth more than a passing consideration, and, in any case, unless the keeping open of a level is a vital matter the subsequent cost of re-opening it will be counterbalanced by the crushed ore from the ribs and the absence of an elaborate system of packing. Nor can much reliance be placed on the elasticity of our strong quartzite hanging walls, an elasticity which does not approach that of the softer strata overlying coal measures. Again the rapid advance adopted with tender roofs in long-wall colliery working is made not only for the purpose of minimising danger but very largely for the purpose of roof control in its action in breaking down the coal. The latter consideration does not affect us here, so, as the cases are different, the same remedy is of doubtful application.

The author makes some sound remarks regarding pigsties, but as far as one can judge from the illustrations he always uses corner posts in building them. These posts are only necessary with steep dips, when the material might fall down the stope, and even then one post in the middle of the lower side is often sufficient. Packwalls make excellent supports, given suitable material with which to build them, but if irregular lumps of waste have to be used, as is generally the case here, proper bonding or bedding is impossible and the result is most unsatisfactory. Generally speaking pigsties are preferable.

The author must have administered a rude shock to the self-complacency of our record breakers with his estimate of 500 ft. per month advance of development faces, and it is a great pity that he has omitted all the humdrum details upon which his startling statement is based. The confined space (see Fig. VI.) would very soon lead to the face being choked up with broken rock, very little of which, on account of the high percentage of fines, would be available for packing; the waste for packs would have to be trammed in from outside, and this would assist in further burdening the already congested traffic; the construction work, if kept as close to the face, as shown, would certainly require frequent renewal; the general hurry-scurry would inevitably mean that now and then some of the reef would be left behind; and the running of these drive-stope faces on lines, while Nature runs her reefs on curves, would mean that now and then all of it would be left behind; finally, as far as I can ascertain, the development record for one month stands at 360 ft.

Mr. E. M. Weston (*Member*): The author is to be congratulated on the production of such a useful and practical paper. It is sad to see, however, that he still clings to some of his old heresies. He still wished to place 8 or 10 machines on a single stope face, but he has a new excuse for the practice this time, and it is that the roof will stand up better. The objections to such a practice are so self-evident that they need not be repeated.

It is slowly being brought home to the mining community of the Rand that they have, regardless of the warnings given by the complete wreckage of mines in other mining fields which worked on the stope and pillar theory until disaster overtook them, been skirting on the edge of a precipice, and that the complete filling of worked out stopes must be adopted. The complete loss of the Atlantic mine in the Lake Superior copper region, despite apparently large pillars, may yet have its counterpart on the Rand. Accidents due to the collapse of pillars such as recently occurred on the Cinderella Deep show that deep mining must be carried on under some new system, which will include partial and complete stope filling. I am of opinion that it will be found economical to introduce sand filling into deep mines by the aid of low pressure compressed air, afterwards damping the sand in the stopes. In this way many of the objections to the water-borne sand filling scheme will be avoided. I hope, if opportunity is offered, to go into this matter more fully later on.

Setting Props.—The theory of setting props a little up from the position, vertical to the dip, is based on the fact that in the roof of the stope the vertical force of gravity dragging the rock composing the hanging wall towards the centre of the earth is resolved into two forces, one acting perpendicular to the dip and one parallel to the dip. The force parallel to the dip increases with the angle of dip, and props are set further and further up hill from the true vertical to the dip to allow of a considerable drag of the hanging wall down the stope before the prop is displaced, otherwise it might be displaced before it takes the weight.

The author's remarks on air costs and drilling speeds are, of course, correct, and the need of them at this period only shows how slow the industry is to appreciate principles of true economy which have been rubbed in for years.

That wonderful discoveries that have been made by watching the small stope drills on trial at work underground is true. That stoping with small machines is cheaper with efficient labour than hand drilling, that the type of drill that was available years ago merely required proper air pressure, and to be made of good material to do this work, that training schools underground should be started and that 6 ft. holes should be used in small stopes, all these some of us discovered and preached to deaf ears years ago in the columns of this *Journal*. Stope drives and breast drives have been used for many years in the Lake Superior mines. When the reef is over 5 ft. wide they are economical; but not, I think, with smaller reefs.

THE TUBE-MILL CIRCUIT AND CLASSIFICATION.

(*Read at February Meeting, 1909.*)

By G. O. Smart (Member of Council).

DISCUSSION.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution:—

Mr. C. O. Schmitt (*Member*): The author has so fully and ably dealt with the main features of working tube mills that I do not think I can add anything worthy of note. There is, however, one point I would like to shortly discuss. The tube mill classifier shown by Mr. Smart (Fig. II.) has a depth of 9 ft. which is so much beyond that found in most plants on the Rand that its adoption in existing plants would, if desired, only be possible in a few instances. The total distance from centre of tube mill to top of classifier is about 10 ft. 6 in., and this height cannot always be obtained without considerable changes to the pulp elevating plant. To overcome the difficulty it will be necessary either to reduce the depth of the cone classifier or to lower the point of feeding the pulp and pebbles to the tube mill. By reducing the depth of the cone the efficiency is liable to be impaired, to feed below the centre line of the tube mill has not been possible with the feeder shown nor with any other device in use on the Rand as far as I am aware. I have designed a feeder for both pebbles and pulp which will receive these materials at any desired point below the centre line of the tube mill, and thus meet the difficulty. The illustration (Fig. I.) herewith clearly shows the application in a case where the outlet of the classifier is below the centre line of the tube mill.

Figs. II., III. and IV. show an enlarged detail of the new feeder. It consists of two chambers with a neck or trunnion for attaching to the end of the tube mill. The outer, or receiving chamber, is circular and communicates with the inner or delivery chamber, by means of a hole of a size large enough to pass the biggest pebbles. The delivery chamber is really a scoop which does not work in the ordinary way but which is fed laterally so that the wear experienced with an ordinary scoop feeder is avoided. The pulp and pebbles are delivered into the inner chamber by means of a vane set at an angle sufficiently steep to effect the ready delivery of any material. Once the pebbles and pulp are in the inner chamber they must inevitably enter the tube mill, unless the pebbles are too large to pass through the hole in the tube mill trunnion. One of the features of the feeder is that pebbles may be fed indiscriminately—which is not possible with other feeders—although the rate of feed is limited by the capacity of the tube mill inlet hole The tube mills now in use generally have an inlet hole of 8 in. diameter, which limits the size of pebbles to 6 in. and further limits the rate of feed to one pebble per revolution. The rate of feeding pebbles could, with the new feeder, be considerably increased if the tube mill inlet hole were larger, as this would permit of continuing the lifting plane of the delivery chamber through the tube mill inlet in the shape of a spiral. There seems no objection to this as the tube mill outlet has a hole 18 in. diameter and more, while 15 in. would be ample for the inlet.

Two feeders of the new type are in use at the Knights Deep, Ltd., and have, I understand, been quite satisfactory.

I may say, in conclusion, that the new feeder has been provisionally protected.

Mr. Fraser Alexander (*Member of Council*): Mr. Smart has opened up a most interesting subject, more particularly so on account of the

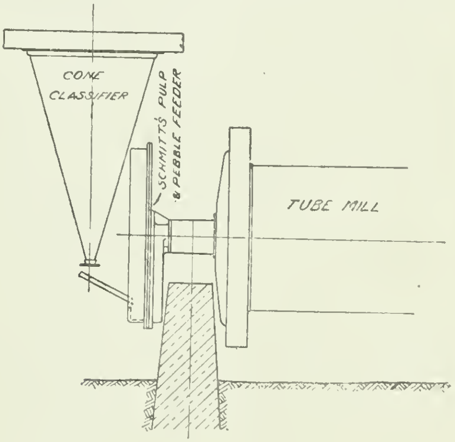


FIG. I.

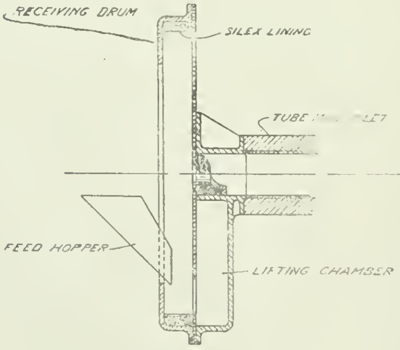


FIG 2

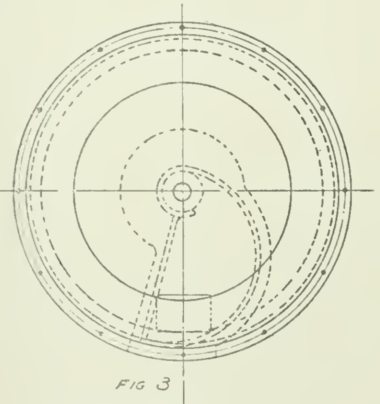


FIG 3

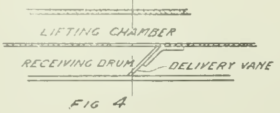


FIG 4

growing tendency to coarse crushing and reliance on tube mills to reduce the coarse particles of ore to a suitable degree of fineness. Now this reduction of the coarse particles is not the only function of the tube mill, and in dealing with classification of the pulp requiring regrinding we are compelled to take into consideration that portion of finer pyrite responsible for locking up as much or more gold than a relatively large proportion of quartz particles. This comparatively fine pyrite is difficult of classification in a large volume of flowing pulp, and, in my opinion, is more liable to be lost in single stage classification than double.

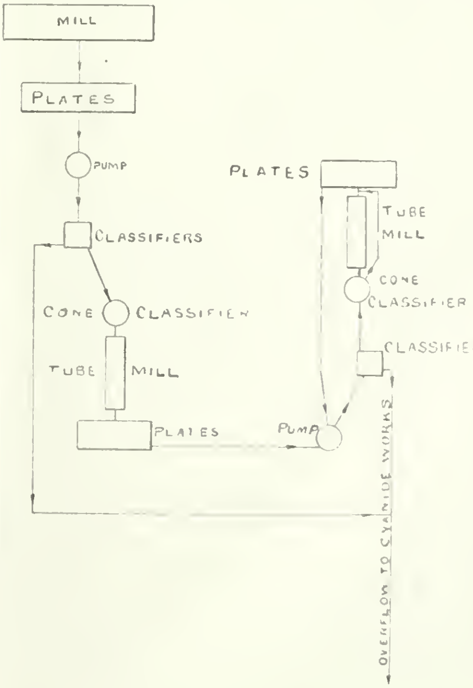
Assuming 100 stamps crushing 800 tons per day with about 8 tons of water to one of rock we have about 7,000 tons by volume to deal with in single classification in place of 60%=4,200 at the dewaterer in double classification, which must allow of more perfect concentration where volume is so much a governing factor in hydraulic classifiers. If classification and tube milling aimed only at regrinding the coarse material (say) +60, then single stage classification would amply remove by gravitation all of the coarser material, but on these fields we require to pass through and recrush, if possible, all of the +90 pyrite in order to recover the maximum gold contents, and these fine particles do not readily settle in so large a volume as must necessarily be dealt with in a single cone separator. These machines, like the old spitzlutten, are of limited capacity within fixed quantities by volume, and to force or overtax their capacity is to court disaster and loss of gold in the final residue.

The following analyses will assist to explain the advantages to be gained from reducing from 8 to 1 to, say, 3 to 1 over the shaking plates and then, reclassifying. This allows first, more intimate contact with the amalgamated copper plates, and secondly, considerably less volume and consequently better concentration throughout the whole circuit. These analyses are from a large sample taken from 400 ton collector tanks, after the tube milling and before and after the usual treatment with cyanide. The separation was carefully performed with a prospecting pan, whilst the screens used were approximately of standard aperture, namely, 60 mesh, 0·010 in., 90 mesh, 0·006 in., 150 mesh, 0·004 in. and 200 mesh, 0·003 in.

It is well known that sampling from a hose filled tank is extremely unreliable and the above figures prove this, but the figures serve their purpose in this case. It is seen from the above figures, that the finer particles of pyrite or concentrate do not readily come in contact with the amalgamated copper plates, and the reason is probably caused by the rush of too large a volume of water and sand over them. On the other hand, the solution and recovery of gold from this finely ground product is assured in the cyanide works.

Before Treatment with Cyanide. Assay Value =5·30 dwt.			After Treatment with Cyanide. Assay Value =0·74 dwt.		
Grading.	Per cent.	Dwt. per Ton.	Grading.	Per cent.	Dwt. per Ton.
+60	=13	4·40	+60	=10·3	0·84
Sand	97·30	3·97	Sand	99·3	0·80
Pyrite	2·70	20·00	Pyrite	0·7	6·6
+90	32·5	4·47	+90	30·0	0·70
Sand	97·0	3·60	Sand	98·0	0·64
Pyrite	3·0	32·60	Pyrite	2·0	3·60
+150	23·0	6·35	+150	8·7	1·20
Sand	91·2	3·07	Sand	90·3	0·60
Pyrite	8·8	40·40	Pyrite	9·7	6·80
+200	7·2	6·40	+200	7·2	0·97
Sand	87·8	2·87	Sand	90·0	0·57
Pyrite	12·2	31·75	Pyrite	10·0	4·50
−200	24·0	6·75	−200	43·2	0·5
Sand	97·0	4·33	Sand	97·0	0·44
Pyrite	3·0	85·00	Pyrite	3·0	2·50

I feel it would not be out of place to describe here the tube mill circuit now in course of construction at one of the crushing units of the Crown Mines, which, as you will see, differs very considerably from Mr. Smart's, and in my opinion



will make for high efficiency in classification, tube milling, and amalgamation. By this means we propose to separate all of that material in which the major portion of the precious metal is contained and without returning this concentrate into the raging torrent obtained from 300 stamps, to finish the operation of fine grinding by careful classification of the pulp requiring treatment in tube mills. Mr. Smart only briefly refers to the importance of bringing all of the metallic gold in contact with the plates, and I wish here to emphasise the fact, that with such coarse crushing as ⅛ in. aperture it is merely a waste of time and mercury to attempt the recovery of gold by amalgamation at the lip of the mortar box as at present practised. At the same time, it is absolutely essential to good all round recovery that all coarse particles of gold should be won before entering the cyanide works, and if this is admitted, then single stage classification is not a sufficiently safe practice. I know we have much to learn about the huge machinery now in use on these fields, and particularly in connection with the above subject. Mr. Smart's paper opens up one of many crucial points in the metallurgy of this, our staple industry, which, from the very nature of the work (wet and dirty), is not always performed as it theoretically should be.

Mr. G. A. Robertson (*Member*): I am rather inclined to think that the author unduly criticises the "ordinary wooden pyramidal or V-shaped classifiers," or otherwise exaggerates the shortcomings of this older method of classification, and at the same time gives undue praise to the method adopted on the Simmer & Jack. Having spent some time on each method of classification for tube milling perhaps a few notes, by the way, on the working of both systems may be of some interest.

The writer was engaged at the starting up of the tube mills on the Knights Deep mine about five years ago. The method of classification consisted of the V-shaped type, and there were in all 12 underflow openings, three abreast. The bulk of the coarse sands from 100 stamps gravitated through the first three nozzles with, of course, the use of the usual water service generally adopted. The coarseness of the grades of pulp gradually decreased towards the back row of classifiers, which row, as a rule, in the writer's time, was seldom running. In regulating the tonnage to the tube mills there were various factors to consider as our knowledge of tube mills in those days was not as good as it is now. The usual ¼ in. mesh screen in the tailings launder arrested the bulk of small pebbles, etc., from entering the classifiers, and there were also affixed the usual hose and wooden mouthpieces, the final outflow openings varying from ¼ in. upwards. Choking of these nozzles did take place but, as a rule, in the first row of classifiers. The coarser product falling into the choked classifier overflowed to the next row and though naturally a coarser pulp did finally overflow, it can be readily seen that the results would have been quite different had we been depending upon one single unit per tube for classification purposes. There were times when the necessary water for the spitzlutten was not available with the consequent deranging of the classification system. To overcome these difficulties Mr. J. C. McMillan, the then battery manager, had holes cut in the last row of classifiers, just in front of the baffles, from which turbid water was available, in quantity regulated by the inserted wooden plugs bored to different diameters. It was then possible by discarding hoses, mouthpieces, etc., and affixing larger nozzles for the underflow from the classifiers to keep a supply of classified pulp running to the tube mills until a clear water service was available. What is known as the dewaterer was of the pyramidal shape, the inflow from the launder delivering below a baffle about 4 in. from one side of the pyramid and on the same angle. Various devices of reducing the moisture entering the tubes were tried, one of which deserves particular mention, viz., the rod designed by Mr. W. R. Dowling. This device consisted of an iron rod about 5 ft. long with a circular handle, the bottom part of the rod finishing off in a cone 6 in. long, the diameter of the top part being 3½ in. A guide bracket was fixed near the bottom of the dewaterer so as to insure an accurate fit for the reception of the cone in the vertical pipe which took the underflow, the pulp afterwards delivering to the tube through a horizontal pipe. The device referred to was used both for the shutting off of pulp and regulating the proportion of moisture delivering to the tube. As a dewaterer, this method could reduce the moisture considerably below what was required for satisfactory tube milling at that time, but the dewaterer was nevertheless liable to choke and the device was finally discarded. Its possibilities however appeared to be considerable if applied to the conical dewaterer with vertical inflow afterwards introduced on several of the mines of the Rand mines group. The extra water used for the running of the classified pulp from the spitzlutten to the dewaterer (which is in reality another classifier) was all required for the shaking tables and flushing of launders.

The author's method of classification is a single cone with diaphragm for each tube and, as he points out, it is necessary to get an equal grading from each overflow so as to obtain maximum results. Not so easy a matter as it might at first appear, that is, if we are to consider the grinding capacity of each tube which varies in accordance with the life of the liners and the type of liner used, and it is much easier to divide an already classified pulp than it is to divide the main stream gravitating from its source of elevation. The distribution of a classified pulp is also not of so vital importance as it is all again elevated to the stage system or central classification station which is a constant for the full crushing capacity of the stamp mill.

The cones available for classification by the author's method depend upon the tube mills running so that if it were possible that two tubes were hung up out of six a much larger tonnage has got to be delivered to the other four cones because stamps are not hung up when tubes are under repairs but several screens are changed to a reduced mesh, the difference in prime crushing not being equal to the tons passed through two tubes on the Simmer and Jack, viz., about 140* tons per tube per 24 hours. As to whether the author's method automatically adjusts itself to the changed conditions with correspondly good results in classification, well, it is also possible that there is a variation of opinion in this respect.

As regards the recovery of gold by amalgamation, the writer might be permitted to state that on the Knights Central plant both the tube outlets and overflows from the classifiers (which are of the type as illustrated by the author) are passed over amalgamated plates, the overflow being responsible for 28·48 of the actual recovery. This figure represents what is collected from the stationary plates for the past three months and arrived at from the mine books, which, by request, were kindly placed at the writer's disposal. The average grading analysis of final pulp, as far as the writer could ascertain at the time was rather better than the figures given by the author, "1st to 8th January, 1910."

A few other points from observation in the working of both methods of classification at different mines may be summed up as follows:—The advantages of the diaphragm in the new method of classification are obvious. The cones are not liable to choke, but they are not choke proof, even with extreme precautions.

In order to get the best results from the new method it is essential to have a steady delivery of pulp from the elevator to the cones, but if the elevator has got "temporary convulsions" a sudden rush may carry coarse sand over the side of the cone and be lost for further classification. The final effect is not so appreciable in the old method due to the series of baffles which becalm the stream. The chief faults of the old method of classification can be compared to a pipe 6 in. long $\frac{3}{4}$ in. diameter. That this pipe is more liable to choke than a pipe of the same diameter but only $\frac{3}{4}$ in. long requires no explanation, and with such effective devices as the Bayly nozzle, etc., applied to the old method considerable improvements in operation and classification are possible. Several arguments have been brought forward by the author in support of his claims for the new method of classification, some of which are apparent under working conditions, others are, however, open to question, but it is only necessary to state that the new method is useful for an additional installation of a tube mill on some types of plant. Considering the conditions of some of our earlier milling plants it would appear to be a costly item to install these classifiers and whether the benefits (if any) on such type of plants would be justified either by way of an increased extraction or lower working costs a few figures on prime cost, etc., of both methods might be useful.

By what the author says, one would gather that the old method of classification, when on the Simmer and Jack, was in anything but "apple-pie" order when, "owing to the less labour and attention required by the classifiers two men were transferred to other work." Be this as it may, the author's paper will go a long way towards making others try to improve upon their existing methods of classification.

Mr. B. V. Blundun (*Member*): There is one point in Mr. Smart's paper which seems to me open to comment.

He says, "in order to obtain full efficiency from tube milling, it is necessary that the tube mill outflow should contain a fairly large proportion of coarse grains, such coarse grains being elevated to return again into the tube mill.' Would not this tend to choking of the tube mill circuit? If the supply from the classifier is not completely ground in one operation, it is obvious that the quantity fed is in excess of the capacity of the tube mill. Whatever amount of unground sand is returned in the first cycle of the pulp will be returned in the second cycle plus a like amount, the amount by which the mill falls short of complete grinding. This accumulation will go on until the amount of sand in the circuit is in excess of the cone capacity, when a coarse cone overflow is inevitable. I fail to see how this can give the greatest efficiency. It seems to me that the greatest efficiency would be obtained if the cone overflow contained no grains larger than required for the final product, and the tube outflow contained none larger than the cone overflow, that is, the grinding complete in one operation. This state of things may possibly be hard to obtain in practice, but the nearest approach to it should give the greatest grinding efficiency that we can obtain.

See *Journal South African Association of Engineers*, Dec., 1908.

THE ASSAY OF TIN ORES.

(*Read at March Meeting, 1910.*)

By Jas. Gray, F.I.C (Member).

DISCUSSION.

Mr. E. H. Croghan (*Member of Council*): Before making any remarks on Mr. Gray's brief *resumé* of the various methods, past and present, of assaying tin ores I desire to express my thanks to him. It is not my intention to discuss the processes mentioned since after experimenting with most of them, that the method usually known as the Pearce-Low is perfectly satisfactory for rapid work and technical accuracy. This method I have adopted for over two years, and about eighteen months ago when some discussion arose in the public papers about serious discrepancies in tin assaying I advocated this process.

Parry is a very strong advocate of the hydrogen reduction method, and appears to be prejudiced against the Pearce-Low. He has recently stated that the latter is satisfactory for low-grade ores, but fails (through giving lower results) on rich stuff. I have proved that the method is perfectly reliable right up to 77% tin in a concentrate, and wish to state, as one who has a very large experience in tin assaying, that Parry is absolutely wrong. If he had operated the method properly he could not have come to such a peculiar conclusion. In regard to the gravimetric fusion method with sodium carbonate and sulphur, with all due respect to Parry and others, though the process may seem tedious and dirty, yet carried out correctly it is extremely accurate. Naturally it is not a method for daily use, but in any special work it deserves consideration. The author's remark that frothing over occurs with caustic soda fusion requires the qualification of the shape and size of iron crucibles employed. At one time I had such trouble, but this was not the fault of the caustic soda. To remedy this I had made here for experimenting a certain size of iron crucible shaped like a platinum crucible, and have since imported similar ones from home. These are doubtless now in general use here.

The author's reference to Simpson's statement that it takes 45 to 60 minutes to obtain solution of the tin ore is not in accordance with my experience. The fusion is easily accomplished by caustic soda within 10 minutes which with a further 20 minutes for cooling solution, etc., involves a total period not exceeding 30 minutes. At this stage I wish to question Low's excessive ratio of caustic soda to ore, namely, 16 to 1; I find that a maximum 8 to 1 is an excellent working proportion. Sodium peroxide I employ in special cases such as where chromite is present, otherwise caustic soda is quite satisfactory.

In the *Chemical News* of August, 1908, a series of experiments by Walton and Scholz are given on "The decomposition of certain minerals and industrial products by means of sodium peroxide and metallic sulphides." No data are given for tin ores, and it would be interesting to carry out some trials. The principle is fusion with chiefly sodium peroxide mixed with smaller amounts of zinc sulphide, iron pyrites, or potassium persulphate. It is stated that "The chief advantage of the method is the quickness with which decomposition can be accomplished. The entire operation, including mixing, igniting (by a piece of twine about an inch long, soaked in alcohol or by magnesium ribbon), cooling and dissolving the melt, takes approximately five minutes." Another advantage is that the fusions are made in nickel crucibles, not platinum, and only a slight loss in weight occurs when the hot oxidising nature of the fusions is considered.

With regard to the experiments made by the author on the influence of certain impurities on the titration with iodine, in view of the mode of procedure, and the divergency of the results shown, they can only be considered as approximations. On a sample rich in tin (as indicated by the author's mixtures for experiments) these impurities might be considered negligible, but the supposed negligible quantities assume a different aspect when it is a matter of reporting definitely on the presence or absence of tin in ore, I find that reduction with iron nails affects the iodine titration.

In the following experiments, which illustrate this effect, the respective quantities of mixtures were fused with NaOH, the melt, etc., dissolved in sufficient HCl leaving a slight excess, 12 (3 in.) iron nails added and reduction carried on for 20 minutes after disappearance of $FeCl_3$ colour. A clear solution was obtained by filtration through glass wool, an atmosphere of CO_2 being maintained in the receiving vessel by $HNaCO_3$. Titration with starch indicator and iodine was proceeded with as usual.

The index value of the iodine was 1 c.c. = ·0052 gram tin.

(*a*) 1 gm. of a mixture of 75% Fe_2O_3 + 25% SiO_2 required 0·25 c.c. of iodine.

(*b*) Similarly a mixture of 50% Fe_2O_3 + 50% SiO_2 gave the same result.

(*c*) 0·5 gm. quantities of the preceding showed 0·1 to 0·15 c.c. iodine.

(*d*) 0·5 gm. Fe_2O_3 dissolved in HCl, reduced by iron nails, etc., as above, absorbed 0·1 c.c. iodine.

(*e*) A blank experiment with 12 (3 in.) iron wire nails heated with dilute HCl for about 30 minutes, filtered, etc., took up 0·1 c.c. iodine.

The first four experiments show an interference due to iron equivalent to practically 0·1 % tin.

Low in his book on technical methods of ore analysis takes a very roundabout way of making sodium arsenite for standardizing the iodine. It is much simpler and perfectly accurate to effect solution by sodium carbonate according to Sutton.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

(*Read at April Meeting, 1910.*)

PART II.—FURNACE EXPERIMENTS.

By Prof. G. H. STANLEY (Member of Council).

DISCUSSION.

Mr. W. Lawrie Hamilton (*Member*): I think we are all indebted to Prof. Stanley for the work he did with, shall I say, disgraceful appliances. I think the Government and those behind his back have not treated him in the way—considering the importance of the matter—he ought to have been treated. I think if he had a little money behind him and a little more time he would have shown better results. If he could have dealt with the ore in the way he would have dealt with it given the proper facilities, I thing he would have found really successful results. I would like to say that the experiments seem to show that he was not backed up in the way he might have been. The whole experiments seem to me to have been based on the practice of a foundry for reducing smelted iron instead of smelting such a complex iron ore. Another thing was the difficulty he had to get his blast in. That, of course, anyone could see in the diagram. He could not expect to have anything but "bears" as he calls them. Another thing, a foundry cupola furnace for smelting iron is altogether the wrong kind of thing in dealing with iron ore. It leaves this impression on the mind that either we know nothing about it, or the ore is so bad that it cannot possibly be dealt with; but I am confident that if the Government will spend time and money and give him a blasting furnace on proper lines he would obtain results which would be of real value to the country. I think we should urge upon the Government to try and stop this finniking way of doing things. The Professor complains of the time he had for the reduction of the ore. The furnace was so little he could not possibly get the time necessary for this reduction. In my experience I have found that to calcine the ore before putting it into a low furnace is an advantage. Had he calcined the ore and then put it in he would have had much longer time to bring it from its raw state. He might have had also a little heating stove and tried to heat the blast with it. I admire the way the Professor carried out his work, but I do not admire the tools and appliances he had.

THE TREATMENT OF ACCUMULATED SLIME, AND THE USE OF FILTER PRESSES FOR CLARIFYING SLIME SOLUTION AND BY-PRODUCTS.

(*Read at November Meeting, 1909.*)

By JOHN D. O'HARA (*Member*).

DISCUSSION.

Mr. J. H. Johnson (*Member*): Contributions to Mr. O'Hara's valuable paper on the above subject can hardly be taken as criticism so much as additions, and I think that any member treating accumulated slime should come forward and add his experiences to this paper.

In July, 1909, the Princess Estate & G. M. Co., Ltd., started treating accumulated slime in conjunction with the current slime. The plant consists of two 45 ft. diameter by 10 ft. deep collecting vats with 3 ft. cones, four 50 ft. diameter by 10 ft. deep treatment vats with 3 ft. cones, with necessary pumps, etc., and one puddler 10 ft. diameter for accumulated slime The capacity of plant when erected was estimated at 4,800 tons per month. We are now treating in the neighbourhood of 6,000 tons.

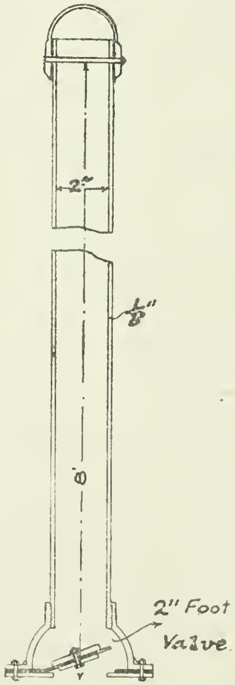
The accumulated slime is transported from storage dams by mechanical haulage, dumped into the puddler, whence it passes through spitzkasten for separation of sand, which sand is delivered above sand settlers by means of a 3 in. Morris pump. In fact, fully 75 % of accumulated slime is passed to the head of sand settlers where it combines with the current product.

When we first started treating we had difficulty in getting good settlements in the collecting vats, not only during collection but after collection ceased, it often taking 20 to 24 hours to decant off the supernatant water. We were then treating from 40 to 60 tons of accumulated slime per charge, and 120 to 140 tons of current slime (charges running from 24 to 28 hours). This naturally affected the settlement in the first treatment vats and also caused us to have too large charges to obtain good extraction (average value of charge 1.65 dwt., residues 0·44 dwt., extraction = 73·33%). Summer months coming in, settlement improved, but the treatment tanks were still delayed.

In February last several improvements were installed, viz., snifter valves, a jet of air into the puddler, and the temperature of solutions was brought up to 90° F. (by Mr. Salkinson's method). The snifters improved the dissolving of gold, the air jet improved the settlement in the collectors, and the increased heat of solutions improved the settlement in treatment vats It was now possible to obtain a charge in shorter time, and also to reduce size of charge. A charge being obtained in 15 hours containing nearly 50% accumulated slime (month of April, 1910 = 48·2%), this enabled the haulage to be worked both day and night, and the accumulated slime to be handled more regularly. Previous to February, a day-shift only was working, which meant crowding all the accumulated slime handled The cold weather coming on in April again caused trouble with settlement in collectors, and it was found necessary to increase temperature of pulp (especially the accumulated slime pulp) and maintain it at 75° to 80° F. A high alkalinity has also to be maintained = ·02% NaOH. The cyanide consumption for accumulated slime since January, 1910, has been 0·638 lb. per ton treated, and lime 19 lb. per ton treated. Costs of treatment of accumulated slime have come down from 5s. 3d. per ton in July, 1909, to 4s. 3d. per ton in January, 1910, and 3s. 3d. per ton in April, 1910. With the author's contention that a transfer from one vat to another and back again being worth all the time wasted in circulating I quite agree, although it would be interesting to hear the advocates of the two methods on the subject, as I believe there is a divergence of opinion respecting this.

Might I ask the author what his method of sampling accumulated slime is? It is stated that the accumulated slime is pumped from agitators to tailings-wheel (where, presumably, it joins pulp from mill) and then sand present is separated. Where is the sample taken? Before entering wheel or after separation of sand from slime? If in the former case, what allowance is made for the auriferous sand present, or if after separation of sand how are the products, current and accumulated slime, sampled separately so as to obtain a true value of each? If the two products are approximately the same value it would not be of any great concern, but if one product is of much higher value than the other sample taken before separation of sand (the residue sample of the two products having to be taken together) might cause fictitious extraction to be obtained, as the richness of the one product would be possibly due to the presence of gold contained in the sand.

The author speaks of a dip sample being taken of residues. How is his sampler constructed? The pipes at the bottom of an Adair-Usher vat would probably prevent a true dip sample being taken, especially as the continuous washing would naturally throw the heavier products to the bottom of tank and between the pipes. A sample taken during the discharging of a vat is apt to be lower than the true value of residues owing to the dilution of dissolved gold by the water used for discharging. The sampler used by me is an 8 ft. length of 2 in. pipe with a leather clack valve (foot valve) and so constructed that it will come as near to the bottom of tank as possible. A sketch of sampler is submitted herewith.



ENDLESS ROPE HAULAGE.

(*Read at December Meeting, 1909.*)

By H. G. Kay (Associate).

DISCUSSION.

Mr. Tom Johnson (*Member*): At the March meeting I mentioned (on information received from the engineer) that the pegs in the jockeys at the Rose Deep had not been turned since the jockeys were made. Some of the members seemed somewhat doubtful, and stated that their experience was to the effect that the pegs lasted three months. What I stated then I find was not strictly correct, on personal enquiry I found that some had been changed to allow of the use of a smaller rope, and the changed pegs had been put in jockeys in use in another haulage using a suitable sized rope. There are still in use on the tailings dumps some of the jockeys with the original pegs that I had put in in 1905, which were made from scrap drill steel. This point should be of interest to those whose pegs wear out in three months, and it might pay the doubters to try pegs of the same material; it may save a few pence.

NOTES ON THE PRECIPITATING EFFECTS OF SUBSTANCES CONTAINING VARIOUS FORMS OF CARBON AND CELLULOSE ON CYANIDE SOLUTIONS CONTAINING GOLD AND SILVER.

(Read at January Meeting, 1910.)

By A. J. Clark (Associate) and W. J. Sharwood (Corresponding Member of Council).

DISCUSSION.

Mr. F. Wartenweiler (*Associate*): In this paper the authors state that instances have been reported where "graphitic" ore (usually slate or schist) has caused precipitation. I would like to report such an instance which came under my observation.

In the course of a metallurgical examination of a slate ore containing a small amount of carbonaceous matter, in Eastern Oregon, U.S.A., in 1905, I obtained the following results:—

The original ore value was 1 oz. gold, and treated with a 0·25% KCN solution in a percolator, gave a maximum extraction of 66·8%. Fine grinding did not increase the extraction. Percolation of the ore after roasting, with a ·10% KCN solution gave an extraction of 91·1%. When the treatment of the raw ore was continued with solution carrying considerable gold in solution, there was a reprecipitation, and two tests where made to determine the amount. In each test the tailings from the above percolator tests of the raw ore were taken and agitated for 24 hours with mill KCN solution of known strength and gold content.

Test No 1.—

Percolator tailings before agitation, gold $6.61 per ton.

Percolator tailings after agitation, gold $9.09 per ton.

Solution before agitation with tailings, gold $2.32 per ton, KCN 15%.

Solution after agitation with tailings, gold $1.24 per ton, KCN ·12%.

Indicating a precipitation of 46·5% of the gold in the solution.

Test No 2. (On a different lot of percolator tailings).—

Solution before agitation with tailings. $4.75 per ton KCN ·07%.

Solution after agitation with tailings, $4.12 per ton, KCN ·05%.

Indicating precipitation of 13·2% of gold values in the solution. My notes do not give the relative volumes of solution, but contain a note that a large solution volume had a tendency to keep the gold in solution.

The meeting then closed.

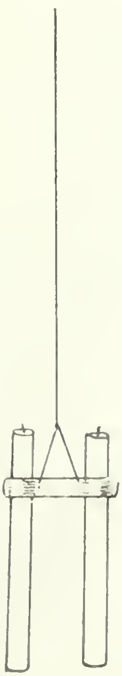
STOPE MEASUREMENTS.

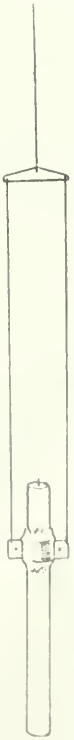
(Read at May Meeting, 1909.)

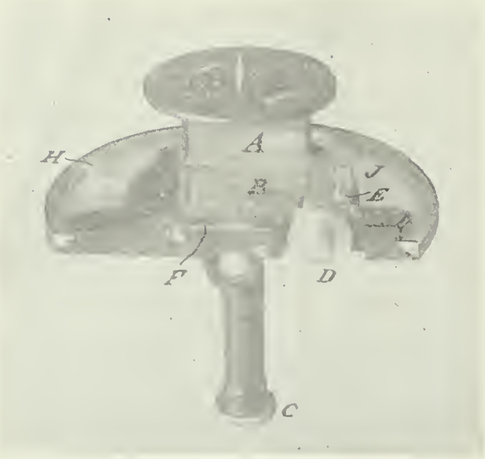
By Otto S. Tonnesen (Member).

REPLY TO DISCUSSION.

Inadvertently the illustrations referred to by Mr. Tonnesen in his reply to the discussion (see







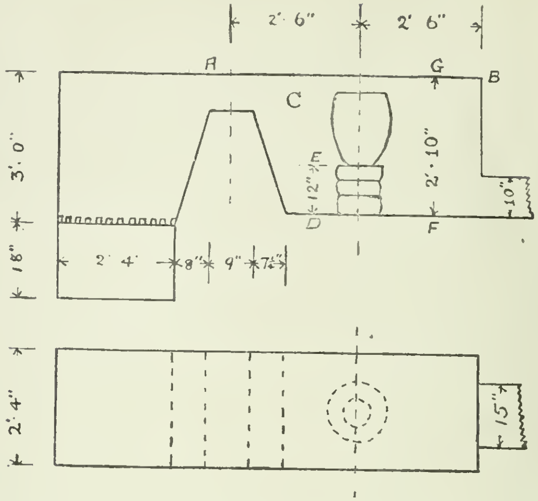
April *Journal*, pp. 369-373) were omitted from the text. They are now reproduced for the information of members.

Contributions and Correspondence.

SMELTING GOLD AND SILVER SLIME.

On reading Mr. F. J. Thomas' paper on "The Small Cyanide Plant as Erected in Rhodesia," I am prompted to send a sketch and description of a method of smelting gold and silver slime which I used in the Hill Country of Chihuahua some years ago. Our practice at that time in the Hill Country was to smelt in the standard type of furnace using charcoal or coke as fuel, the cost of transport (about 20 days by mule back) and the loss in weight of the coke in wear during transportation made its price almost prohibitive. The disadvantages of charcoal as fuel and its cost need not be pointed out, but in addition to this we found ourselves caught in the midst of the wet season without charcoal and unable to get supplied at any price.

I conceived the idea of a reverberatory wood furnace, and though the idea may not be new, I had never heard of its application to this use. We had on hand a sufficient supply of adobes (sun-dried bricks used for houses, etc.). The accompanying sketch is self-explanatory, and is one I sent to a friend who had heard of our success with it at La Republica M. Co., and wished to install one. Its success was instantaneous. We obtained a very hot fire in a short time, and did quick, clean and cheap work. Fuel used was "white pine" or a white scrub oak locally called "Enceno Blanco." I believe this method has been adopted under similar conditions in several good sized mines in Chihuahua with decided success. We were shipping about 45,000 oz. of fine silver per month when I installed this furnace. Possibly there may be places in Rhodesia where this could be made a money saver.



Yours very truly,

A. L. Blomfield (Associate).

Golden Cycle Mining Co.,
Colorado Springs, Colo., U.S.A.,
March 16, 1910.

Notices and Abstracts of Articles and Papers.

METALLURGY.

Permanent Expansion of Cast Iron under Moderate Temperature.—"In May, 1909, Crane Co. made two straight bars of cast iron, $\frac{1}{4} \times \frac{1}{2}$ in., accurately machined The following results were obtained :—

	Inch.	Inch.
Original lengths	18·336	23·382
Gain in five weeks at 325 ...	·003	·008
Gain in thirty days at 600°	·145	·177
Gain in thirty days at 600	·014	·028
Final lengths	18·498	23·595

The permanent expansion at a temperature of 325° (equivalent to 82 lb. steam pressure) was very small, but at 600° it equalled in sixty days ·104 in. per foot on both bars, which is an extraordinary amount.

The measurements were taken with the greatest care and carefully checked. Further experiments along this line of investigation are being conducted." —*The Valve World*, Nov., 1909, p. 426. (H. A. W.)

1 Ton = 2,000 Lbs. = 32 Cub. Ft. of Solution.

THE CAPACITY OF CIRCULAR VATS PER FOOT OF DEPTH.

By W. A. Caldecott.

Sp. Gr. of Dry Slime = 2·

Diameter of Vat in Feet.	Circumference of Vat in Feet.	Cubic Feet per Foot Depth.	Tons of Dry Sand per Foot Depth.			Tons of Solution per Foot Depth.	Slime Pulp.															
			Collected.	Leached.	Transferred.		Percentage of Moisture.															
							30%	32·5%	35%	37·5%	40%	42·5%	45%	50%	1 to 1½	1 to 2	1 to 2½	1 to 3	1 to 3½	1 to 4	1 to 4½	1 to 5
							1·788	1·739	1·693	1·649	1·607	1·567	1·530	1·459	1·337	1·266	1·219	1·187	1·163	1·144	1·129	1·117
		Area of Vat in Square Feet.	Cubic Feet per Ton.				25·56	27·27	29·08	31·05	33·19	35·35	38·01	43·89	59·79	75·75	92·00	107·75	123·62	139·91	156·18	172·20
			21·5	24	26	32	Tons of Dry Slime per Foot Depth of Pulp.															
5	15·71	19·63	913	·818	·755	·613	·768	·720	·675	·632	·591	·552	·516	·447	·328	·259	·213	·182	·159	·140	·126	·114
6	18·83	28·27	1·31	1·18	1·09	·883	1·106	1·037	·972	910	·852	·710	·744	·644	·472	·373	·307	·262	·228	·202	·181	·164
7	21·99	38·48	1·79	1·60	1·48	1·20	1·51	1·411	1·32	1·240	1·16	1·083	1·01	·876	·643	·508	·418	·357	·311	·275	·246	·223
8	25·13	50·27	2·34	2·09	1·93	1·57	1·96	1·843	1·73	1·620	1·51	1·415	1·32	1·15	·841	·664	·546	·466	·407	·359	·322	·292
9	28·27	63·63	2·96	2·65	2·45	1·98	2·49	2·333	2·19	2·049	1·92	1·791	1·67	1·45	1·06	·840	·691	·590	·515	·455	·407	·369
10	31·42	78·54	3·65	3·27	3·02	2·45	3·07	2·880	2·70	2·529	2·36	2·211	2·06	1·79	1·31	1·03	·855	·728	·633	·561	·503	·456
11	34·56	95·03	4·42	3·96	3·66	2·97	3·72	3·485	3·27	3·061	2·86	2·675	2·50	2·17	1·59	1·25	1·03	882	·769	·679	·608	·552
12	37·70	113·10	5·26	4·71	4·35	3·53	4·42	4·147	3·89	3·643	3·41	3·183	2·98	2·58	1·89	1·49	1·23	1·64	·915	·808	·724	·657
13	40·84	132·73	6·17	5·53	5·11	4·15	5·19	4·867	4·56	4·275	4·00	3·736	3·49	3·02	2·22	1·75	1·44	1·23	1·07	·949	·850	·771
14	43·98	153·94	7·16	6·41	5·92	4·81	6·02	5·645	5·29	4·958	4·64	4·333	4·05	3·51	2·57	2·03	1·67	1·43	1·25	1·10	·986	·894
15	47·12	176·72	8·22	7·36	6·80	5·52	6·91	6·480	6·07	5·691	5·32	4·974	4·65	4·03	2·95	2·33	1·92	1·64	1·43	1·26	1·13	1·03
16	50·27	201·06	9·35	8·38	7·73	6·28	7·87	7·373	6·91	6·475	6·06	5·659	5·29	4·58	3·36	2·65	2·19	1·87	1·63	1·44	1·29	1·17
17	53·41	226·98	10·55	9·46	8·73	7·09	8·88	8·323	7·81	7·310	6·84	6·388	5·97	5·17	3·80	3·00	2·47	2·11	1·84	1·62	1·45	1·32
18	56·55	254·47	11·84	10·60	9·78	7·95	9·96	9·331	8·75	8·195	7·67	7·162	6·69	5·80	4·26	3·36	2·77	2·36	2·06	1·82	1·63	1·48
19	59·69	283·53	13·19	11·81	10·91	8·86	11·09	10·397	9·75	9·131	8·54	7·980	7·46	6·46	4·74	3·74	3·08	2·63	2·29	2·03	1·82	1·65
20	62·83	314·16	14·61	13·09	12·08	9·82	12·28	11·520	10·80	10·118	9·47	8·842	8·26	7·16	5·25	4·13	3·42	2·91	2·54	2·24	2·01	1·83
21	65·97	346·36	16·11	14·43	13·32	10·82	13·55	12·701	11·91	11·155	10·44	9·748	9·11	7·89	5·79	4·57	3·76	3·21	2·80	2·48	2·22	2·01
22	69·12	380·13	17·68	15·84	14·62	11·88	14·87	13·939	13·07	12·243	11·45	10·699	10·00	8·66	6·36	5·02	4·13	3·53	3·07	2·72	2·43	2·21
23	72·26	415·48	19·32	17·31	15·98	12·98	16·26	15·235	14·29	13·381	12·52	11·694	10·93	9·47	6·95	5·48	4·52	3·86	3·36	2·97	2·66	2·41
24	75·40	452·39	21·04	18·85	17·40	14·14	17·70	16·589	15·56	14·570	13·63	12·733	11·90	10·31	7·57	5·97	4·92	4·20	3·66	3·23	2·90	2·63
25	78·54	490·87	22·83	20·45	18·88	15·34	19·19	18·000	16·88	15·809	14·79	13·816	12·90	11·19	8·21	6·17	5·35	4·55	3·96	3·51	3·15	2·86
26	81·68	530·93	24·69	22·12	20·42	16·59	20·77	19·469	18·26	17·099	16·00	14·943	13·97	12·10	8·88	7·01	5·77	4·93	4·29	3·79	3·40	3·08
27	84·82	572·56	26·63	23·86	22·02	17·89	22·40	20·996	19·59	18·440	17·25	16·115	15·06	13·05	9·58	7·56	6·22	5·31	4·63	4·09	3·67	3·32
28	87·97	615·75	28·64	25·66	23·68	19·24	24·03	22·580	21·17	19·831	18·55	17·330	16·20	14·03	10·30	8·13	6·69	5·71	4·98	4·40	3·94	3·58
29	91·11	660·52	30·72	27·52	25·40	20·64	25·84	24·221	22·71	21·273	19·90	18·590	17·38	15·05	11·05	8·72	7·18	6·13	5·34	4·72	4·23	3·84
30	94·25	706·86	32·87	29·45	27·19	22·09	27·64	25·921	24·30	22·765	21·30	19·895	18·62	16·12	11·81	9·31	7·70	6·56	5·70	5·05	4·53	4·11
32·5	102·10	829·58	38·59	34·57	31·91	25·92	32·46	30·421	28·53	26·718	24·99	23·349	21·83	18·90	13·87	10·95	9·02	7·70	6·71	5·93	5·31	4·82
35	109·96	962·11	44·74	40·09	37·00	30·07	37·62	35·281	33·09	30·986	28·99	27·079	25·28	21·94	16·09	12·68	10·48	8·92	7·76	6·88	6·17	5·60
37·5	117·80	1104·47	51·37	46·02	42·48	34·51	43·21	40·501	37·98	35·571	33·28	31·086	29·06	25·16	18·47	14·58	12·01	10·25	8·93	7·89	7·07	6·41
40	125·66	1256·64	58·41	52·36	48·33	39·27	48·13	46·081	43·20	40·471	37·86	35·368	33·02	28·65	21·00	16·56	13·69	11·65	10·14	8·98	8·05	7·31
42·5	133·26	1413·06	65·72	58·88	54·35	44·16	55·28	51·817	48·59	45·509	42·57	39·771	37·18	32·20	23·63	18·65	15·36	13·11	11·43	10·10	9·05	8·21
45	141·37	1590·43	73·97	66·27	61·17	49·70	62·19	58·322	54·69	51·222	48·24	44·763	41·80	36·26	26·58	20·95	17·32	14·75	12·83	11·37	10·19	9·25
47·5	149·23	1772·05	82·42	73·84	68·16	55·38	69·33	64·982	60·91	57·071	53·39	49·875	46·62	40·37	29·61	23·39	19·26	16·45	14·33	12·67	11·35	10·29
50	157·08	1963·50	91·32	81·81	75·52	61·36	76·77	72·002	67·52	63·237	59·16	55·263	51·60	44·77	32·81	25·87	21·39	18·21	15·81	14·04	12·58	11·42
52·5	164·93	2164·75	100·69	90·20	83·26	67·65	84·69	79·382	74·44	69·718	65·22	60·927	56·95	49·32	36·21	28·58	23·53	20·09	17·51	15·47	13·86	12·57
55	172·79	2375·83	110·50	99·00	91·38	74·24	92·90	87·122	81·70	76·516	71·58	66·868	62·44	54·17	39·71	31·30	25·88	22·03	19·17	16·99	15·23	13·82
57·5	180·61	2596·72	120·78	108·20	99·87	81·15	101·59	95·223	89·30	83·630	78·24	73·085	68·32	59·16	43·43	34·28	28·23	24·10	21·01	18·56	16·63	15·08
60	188·50	2827·43	131·50	117·81	108·75	88·36	110·56	103·683	97·20	91·061	85·19	79·579	74·31	64·47	47·25	37·25	30·80	26·22	22·82	20·22	18·42	16·45
65	204·20	3318·31	154·34	138·26	127·63	103·70	129·82	121·684	114·11	106·870	99·98	93·395	87·30	75·61	55·50	43·81	36·07	30·80	26·84	23·72	21·25	19·27
70	219·91	3848·45	178·99	160·35	148·02	120·26	150·49	141·124	132·36	123·944	115·95	108·316	101·14	87·74	64·31	50·70	41·92	35·69	31·05	27·52	24·46	22·39
75	235·6	4417·86	205·48	184·08	169·91	138·06	172·84	162·004	151·92	142·282	133·11	124·342	116·23	100·66	73·89	58·32	48·02	41·00	35·71	31·58	28·29	25·66
80	251·33	5026·55	233·79	209·43	193·32	157·07	196·55	184·325	172·80	161·886	151·29	141·473	132·20	114·10	83·94	66·35	54·29	46·24	40·21	35·69	32·17	29·15
85	267·04	5674·50	263·93	236·14	218·25	177·33	222·01	205·088	195·13	182·754	170·97	159·710	149·29	129·29	94·91	71·91	61·68	52·66	45·90	40·56	36·33	32·95
90	282·74	6361·73	295·89	265·07	244·68	198·80	248·76	233·287	218·79	204·887	191·48	179·052	167·31	144·41	106·23	83·97	68·70	58·53	50·89	45·17	40·71	36·90
95	298·45	7088·22	329·68	295·34	272·62	221·51	277·32	259·927	243·75	228·284	213·56	199·500	186·48	161·50	118·55	93·57	77·05	65·78	57·31	50·66	45·38	41·16
100	314·16	7853·98	365·30	327·25	302·07	245·43	307·11	288·008	270·08	252·946	236·40	221·052	206·56	178·92	131·16	103·67	84·82	72·26	62·83	55·76	50·27	45·55

RAND METALLURGICAL PRACTICE.—"At the time it was decided to proceed with the construction of the Benoni Consolidated Reduction Plant, the principle of double stage crushing by means of the heavy-weight stamp plus tube mill combination introduced by Dr. Caldecott had received almost universal confirmation from all the leading engineers on the Rand. And this was not merely academic confirmation only, but, what was of far greater importance, confirmation of the practical sort involving the alteration, wherever possible, of existing reduction plants from single-stage to double-stage crushing and the designing of all new reduction plants for the new method of treatment.

I must confess that I was at first, to some extent, led away by this universal enthusiasm for the new method, especially as it had been brought forward and vouched for by a metallurgist of Dr. Caldecott's eminence and experience after a long series of experiments especially devised and superintended by Dr. Caldecott himself to test the truth of the principle involved. As the new method had then, moreover, received the practical confirmation of every engineer of any standing on the Rand, I accepted Dr. Caldecott's conclusions without reserve and consented to the adoption of the principle of double-stage crushing on the Benoni Gold Mine.

The plant was designed on the lines that the investigations of the Mines Trials Committee have conclusively proved to result in the attainment of the, in the light of our present knowledge, highest obtainable mechanical efficiency of both stamp and tube mill.

These investigations have shown that the low cost of crushing with light-weight stamps through fine mesh screening is more nearly approached in double-stage crushing as the weight of the stamp approaches the maximum and the coarseness of the battery screening increases. And that it is with intermediate weight stamps and screens of medium mesh that the mechanical efficiency of the double-stage crushing unit falls so far below that of the light-weight stamp, and that the cost of crushing by the double-stage method compares most unfavourably with the cost of single-stage crushing.

Fully realising this, I advised the erection of stamps of 2,000 lb. weight and the adoption of the 4-mesh screening, and, of course, calculated the number of tube mills necessary to deal with the estimated output. Dr. Caldecott's remark that the number of stamps and tube mills shows a 'ratio of fewer stamps per tube than are in operation in any other plant on the Rand,' with its implied sarcasm is surely uncalled for, as no one can know better than Dr. Caldecott himself that the ratio of stamps to tube mill is not a fixed one, but is dependent on the duty of the stamp, the coarseness of the battery screening, and the desired fineness of the final pulp.

A 2,000 lb. stamp crushing through 4-mesh screening has a duty of 20 tons, the pulp containing about 75% of +90 product, and as a standard 22 ft. tube mill has an output of about 250 tons per day grinding to a final pulp containing not more than 10% of +60 it can be easily calculated that the ratio decided on is the right one.

Although I advised the adoption of the principle of double-stage crushing at the Benoni Consolidated, I at the same time kept a very close watch on the reduction costs of those mines that had adopted this new method of reduction, and as month by month I saw no decrease in these figures that could be put down as due to this new method of treatment, and as I saw, moreover, that the working costs at New Kleinfontein, which still adhered to the old method of single-stage crushing, compared most favourably with the working costs on the mines that had followed where Dr. Caldecott had led, it became daily more and more apparent to me that the claims put forward by the advocates of the tube mill were not being fulfilled in actual practice.

To confirm this supposition I had very accurate estimates made of the comparative capital expenditure involved in the two methods, and had, at the same time, a careful analysis made of all available figures bearing on the running and maintenance costs of the two types of plant, with the result that I was reluctantly (reluctantly, as I had hoped that we were on the eve of great economic improvements in milling) compelled to come to the conclusions stated in my contribution to the discussion on Dr. Caldecott's paper on 'The Development of Heavy Gravitation Stamps,' which conclusions have been subsequently confirmed by the results so far arrived at in the investigations at present being carried out at New Kleinfontein.

Unfortunately the design and construction of the Benoni Consolidated plant is so far advanced that it is practically impossible, without heavy financial loss, and great delay, to revert to the old method of single-stage crushing.

I do not, however, anticipate that the adoption of double-stage crushing at Benoni will seriously increase the working costs on that mine as compared with what the working costs might have been expected to have been had the old method of single-stage crushing been adhered to, as the plant has been designed, as I have already explained, to allow of the maximum known efficiency of both the stamps and tube mills being obtained."—E. J. WAY.—*South African Mining Journal*, April 30, 1910, p. 255. (A. McA. J.)

TREATMENT OF SLIME.—"The complexities attending the treatment of slime all hinge upon that peculiar property by which finely divided solid particles in suspension acquire a virtual specific gravity that is less than their normal specific gravity, and as a result of which they will not readily pass out of suspension. The meaning of this property is dynamically just as unproven as is the nature of the all-pervading link between particles of matter, and a branch of metallurgy so intimately connected with a subject so little understood may well be approached with hesitation.

Lodge tells us that the great force in nature is the *vis a tergo*. Let us start with this assumption and consider our subject from the point of view of an impelling force retarded. To borrow an example from the same authority, the front end of a steel bar must be pulled, before the *vis a tergo* impels the hind end to follow. Evidently there is an obstacle to the free settlement of slime, and it is reasonable to assume that it would be better to remove this obstacle rather than adopt methods of separation that postulate continued suspension. To such engineers as might consider that the complexities attending the practice of vacuum-filtration are unduly great, or that the method leaves something to be desired, a description of a plant recently erected at the Queen mine, in the Sheep Creek district of British Columbia, will be of interest.

This plant is the outcome, on a working scale, of experiments the aim of which was to apply the principle known as acceleration of settlement, due to the removal of fine slime as it settles. This acceleration was first noticed by me when making attempts to withdraw settled slime by means of a conveyor from the bottom of a tank in which a charge had been collected, and the effect upon the rate of settlement was so surprising as to encourage the idea of a continuous process. This object has now been achieved, for the plant herewith illustrated treated pulp at the rate of 100 tons of dry slime per 24 hours.

The main feature of the process is the agglomeration of slime upon the belt. If the settling slime be withdrawn without interfering with the progress of settlement, the increased tendency of solids to pass out of suspension is so marked that this agglomeration permits the withdrawal of the solid particles out of the pulp in a compact form. The device required to facilitate the operation is found in a conveyor travelling upward at an angle of about 18° at the rate of 4 to 6 ft. per minute. The inclination and speed vary with the character of the slime handled, and may be adjusted in accordance with the ultimate purpose of the separation.

Before proceeding further it may be well to consider for a moment the explanation of this accelerated settlement. It may be taken as an axiom that the greater the concentration of solid matter in suspension in a liquid medium, the slower is the rate of settlement; therefore, apart from any other factors affecting settlement, the suspended solids constitute in themselves an element of retardation, or, in other words, retardation of settlement is largely due to the mutually repellant action of the solid particles. Accepting this fact, let us consider it in connection with three imaginary layers, A, B, and C, in a partly settled pulp. As between these three layers there is an upward force acting from A to B, and again from B to C, and downward forces acting from C to B and B to A; so that as far as B is concerned, apart from the force of gravity, it is in equilibrium. If the layer A is removed the upward force A to B is removed and the layer B is free to settle, or is 'free-settling.' The essential factor is the removal of the bottom layer and the withdrawal of each *lowermost solid particle*. Thus a movable bottom to the settling-tank is implied. There is a wide difference between the physical state of the settled particle and that of the one still partly in suspension above it; thus, a scraper working over the bottom of a tank and dragging out thickened pulp is performing a different function from that of a conveyor removing settled slime.

Many different classes of slime have been treated, from finely ground hard quartz to pure clay, but a discharge that is lumpy can be obtained in all cases. The tank is essentially a well or trough set at an angle of 18°, in and out of which a conveyor-belt 2 ft. wide, having flanged sides, travels over 14 in. rollers. The sides of this trough are vertical and are 24 in. high; over them is built a long settling-tank, which at its wide and deep end (over the lower roller) extends to a height of 6 ft. above the trough. This settling-tank is divided into partitions, and nowhere are the sides at a less angle than 65°. The lower roller is set in the well or trough, which is closed at the rear end, and revolves in bearings with stuffing-boxes and removable bushings. The head roller is set on brackets at such a height above the horizontal top of the trough as to allow of an upward travel of 8 ft. above the surface of the pulp. An arrangement of sand-filters set in the tank itself allows the solution to be removed from near the surface (where the pulp is naturally thinnest) to be delivered clarified by means of a vacuum pump. The sand-filter is set in circular trays of 2 ft. diameter, with 6 in. sides, the top and bottom being made of fine screening, which is placed upon the top of an inverted cone, to the bottom of which the vacuum-hose is attached. A simple mechanism attached to the driving-shaft permits a reversal of the vacuum once every two minutes, when a slight flow of solution is delivered under low pressure, cleansing the sand of the slime that has been sucked down into it. Thus continuous operation is rendered possible. These internal filters are not, however, essential to the process. The solution may be drawn off unclarified from the top of the settling-tank and passed through ordinary sand-filters, which can be periodically cleaned without the use of a vacuum at all.

The pulp is delivered at the rear end of the tank, and the solid slime settling out of suspension deposits itself upon the belt and is thereby drawn out of the pulp. The slime falls and is washed off the belt by a spray into a hopper, through which it is discharged into a small air-lift agitator, to emerge as a continuous stream of thoroughly disintegrated pulp. This settling action underlies the practical application of the method. At the rate of 100 tons of dry material per day a ribbon of thick slime 1¼ in. deep is removed on the belt, which, by the time it reaches the head roller, contains no more than 27% moisture.

The feed for this plant was taken from the tailing of a 20-stamp mill after passing through a hydraulic classifier. The whole overflow passed 100 mesh, and 70% passed 200 mesh. This overflow was elevated by a centrifugal pump to two Callow thickening tanks, from which the discharge (thickened to 15% solid) was run direct to the settling-tank.

In order to obtain a discharge sufficient to demonstrate the capacity and mechanical efficiency of the plant, the washed pulp from the air-lift disintegrator at the head of the belt was also returned to the centrifugal pump and was elevated by it to the thickening-tanks. In this way any desired flow was obtainable, and in order to give the plant a thorough trial, a continuous run was made for 30 hours, handling at the rate of 140 tons of dry slime per 24 hours. In this case the machine was overloaded and a certain amount of the slime layer being discharged by the belt, slipped off the sides and back into the tank. This was done purposely in order to try the efficiency of the means adopted for keeping the tank free of accumulating slime. The method is simple. Two 2 in. pipes are run through the cover of the trough at the rear end, just behind the lower roller, to within three inches of the bottom; the top of these 2 in. pipes extends into a small box built upon the top of the wide end of the tank, through which a hole is bored. The top of the pipes is below the surface of the pulp in the tank. A little air admitted at the bottom of the 2 in. pipes by two ½ in pipes is sufficient to cause a slight continuous upward current, which has the effect of preventing any accumulation of slime on the bottom of the tank without interfering with the deposition upon the belt, and it was found after the 30 hours' trial that the bottom of the tank was perfectly free from slime.

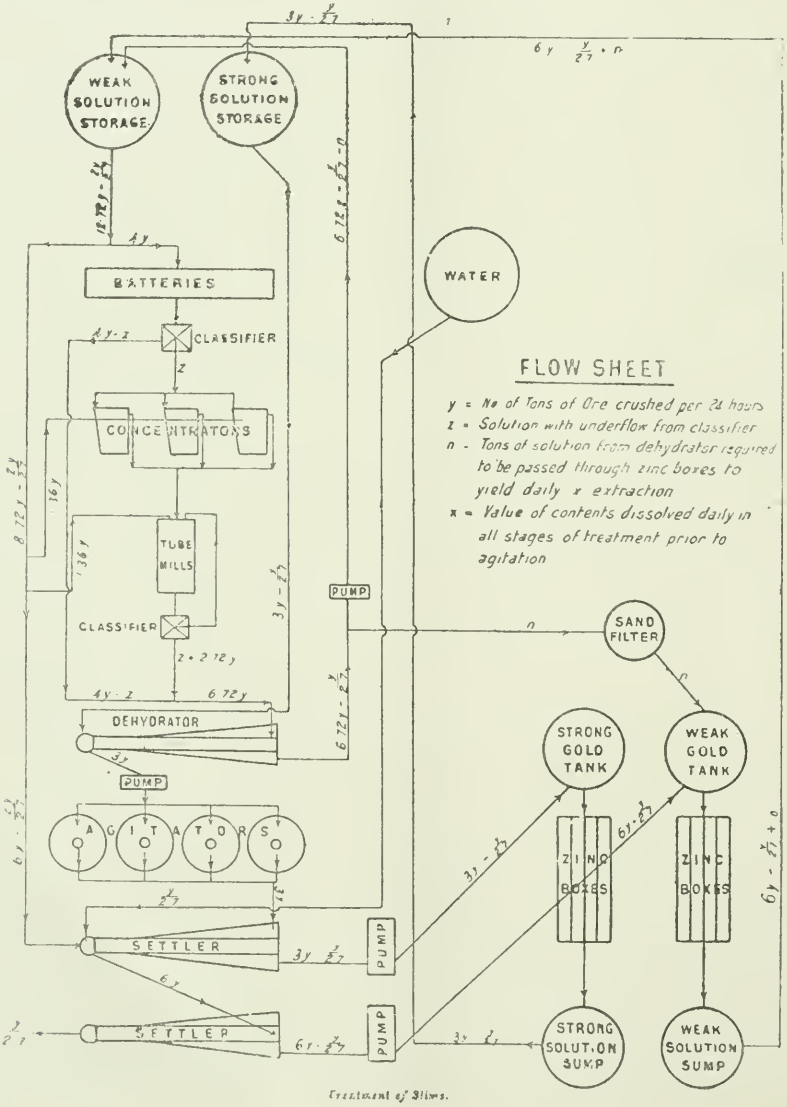
The sides of the trough were made of wood 3 in. thick, and all the rest was 2 in. thick; the bearings for the rollers were castings bolted down to the supporting 6 by 8 in. timbers, upon which the tank rests. The belt was a 4-ply rubber belt, 2 ft. wide, having flanges 1¾ in. high. These flanges are not necessary, but were adopted in the first plant as being more satisfactory. The cost of this plant, apart from the power and vacuum pump, did not exceed £150. As may be judged, the power consumed, and applied to the roller by worm gearing, was slight; 2 h.p. was sufficient to run all the machinery.

The plant was used only as a dehydrator, and to demonstrate the handling of a sufficient and continuous flow of pulp, but the disintegrating and washing air-lift agitator was also used, looking to the applicability of the process to cyanidation. This leads to a consideration of the relative merits of displacement and dilution washing. Aside from any doubt as to the perfection of washing obtainable by known displacement methods, the main feature of decantation practice is the dependance upon calculations as to the extent to which washing by dilution is effected. And in the case of a continuous process, this advantage is more pronounced. The accompanying flow sheet will indicate the manner in which the method may be applied in the case of an ore crushed in cyanide solution.

The process of treatment suggested in connection with washing the slime off the belt may also be gathered from the following example of an ore crushed in water, re-ground, and passed to Pachuca agitators. Between the crushing or grinding machinery and the agitators there would be a slime-settler to act as a simple dehydrator, and the discharge would be washed off the belt with strong cyanide solution and elevated to the agitators.

In such a case there may be 100 tons of ore and 600 tons of water, or 700 tons of pulp going to the first settler for dehydration. The belt in this settler discharges pulp having 27% moisture. Therefore, from this first dehydration there would be 100 tons of slime and 37 tons of water, or 137 tons of pulp going to agitators.

Up to this point 700 − 137 = 563 tons of water has been returned to the batteries. The 137 tons of pulp going to the agitators is made up to a consistency of 25% solids by adding 300 − 37 = 263 tons of strong solution. Thus 263 tons of strong solution, plus 137 tons of pulp, or 400 tons of cyanide pulp, eventually



Treatment of Slimes.

go to the first separator. From this first separation, discharged 100 tons of slime and 37 tons of gold, cyanide solution, making 137 tons delivered from the separator.

Up to this point, 400 − 137 = 263 tons of strong solution of a value, say, of 3 dwt. per ton, containing altogether 789 dwt., has been recovered and passed through zinc-boxes, and thence to the strong cyanide vat. The discharge from this first separator, namely, 100 tons slime and 37 tons of gold cyanide solution, containing 3 dwt. per ton equal to 111 dwt., is washed off the belt with barren precipitated solution, producing a ratio of solution to solids of 6 : 1. The amount of barren solution added then is 600 − 37 = 563 tons of weak solution, producing eventually 600 containing 111 dwt. gold, equal to 0·185 dwt. per ton.

We thus have 700 tons of pulp with weak solution going to the second separator. From this separator the discharge is again 100 tons slime and 37 tons of weak solution at 0·185 dwt. per ton. Thus 6·85 dwt. gold is going to waste.

Up to this point, 700 − 137 = 563 tons of weak solution of a value of 0·185 dwt. per ton, or a total of 104·15 dwt., has been recovered and returned to the weak zinc-boxes and thence to the weak cyanide storage-vat. Assuming now that the original ore was of a value of 10 dwt. per ton, and that 90% extraction was obtained from the solid slime, the total loss from 100 tons is 100 dwt not extracted, plus 6·85 dwt. not dissolved, or 106·85 dwt., which gives a total loss of 10·69%, or a total extraction of 89·31%. Of the 900 dwt.-gold in solution we have then a total loss of 6·85 dwt. or about ¾%.

It will be seen that there is no accumulation of solution."—H. G. Nichols.—*Mining Magazine*, Nov. 1909, p. 221. (J. A. W.)

MINING.

Ventilating System at the Comstock Mines, Nevada.—"This paper by the Professor of Mining and Metallurgy at the University of Nevada deals in extenso with this subject, under the following heads:—1. Temperatures (surface, underground, mine workings. 2. Ventilating system (*a*) in general, (*b*) in detail. 3. Measurement of ventilating currents, (*a*) temperature (*b*) velocity. 4. Cost and operating expense. 5. Underground humidity. 6. Control of humidity and heat. 7. Physiological effect of working in high temperature and humidity. 8. Appendix by Professor Frandsen.

The ventilation of metalliferous mines has received but little attention, and comparatively few data are available. The literature of coal-mining, on the other hand, is replete with data of mine-ventilation, the topic being one of considerable interest even at the present time. In coal mines, ventilation has for its principal object the removal of explosive gases. To accomplish this, fans are generally employed to force into or exhaust from a mine large volumes of air. In metal-mining, while occurrences of gases are not infrequent, the volume of gas encountered is usually so small as to be unimportant, and it is generally non-explosive. In these cases the purpose of the ventilating-current is to supply pure air to the miners in the different working-places, and to remove the gaseous products of blasting. In but few instances has the ventilating current to perform the additionol function of cooling the mine-workings. In most metal-mines ventilating-currents are established by the use of two shafts, one an upcast, the other a down-cast. The natural elevation of the ground-temperature, due to increased depth, is relied upon to warm the air sufficiently to start an upward current. As depth is attained, the average temperature of the ascending current is raised and the ventilating-efficiency of an upcast shaft increased. Fans and air-pipes are used for the ventilation of pits, shafts, and dead-ends, but are seldom used for the general ventilation of a large mine. The Comstock mines furnish an excellent example of the use of natural ventilation for providing a current of sufficient volume partly to overcome abnormal conditions. That natural ventilation is insufficient and not controllable enough, is evidenced by the fact that at the present time one large fan is in operation and another is being erected. Ventilation by small fans is also very largely used to supplement the effect of the shafts.

It is impossible to give an abstract of the measuring instruments and measurements made, and for these the original paper must be consulted. The physiological effect of working in high temperatures and humidity is discussed fully, the opinions of Le Neve Foster, Eliot Lord, Haldane, and Thomas, and Church, being quoted, the main facts brought out by these observers being that high temperatures and humidities cause, under some conditions, a notable rise in body-temperature; that all the bodily organs with the exception of the stomach are stimulated; that prolonged exposure to such conditions undoubtly lowers vitality; that intermittent exposure produces no permanent ill-effects; that these conditions cannot be considered an important cause of lung-diseases; and, lastly, that under abnormal conditions loss of mental control and serious bodily disturbances result.

Local physicians state that the average life of the Comstock miner approximates 25 years, and that the miners do not show any greater susceptibity to any particular disease than the residents of the town. The author states he is acquainted with miners who have worked more or less continuously underground for 30 years, and who are still capable of doing a good day's work. Compared with miners of other districts the Virginia City miners may be said to be just as healthful, if not more so. The result is due in a large measure to the fact that the Virginia City miner observes certain precautions, and that the mine-managements provide the necessary facilities. Miners are careful not to expose themselves to cold drafts, since they work stripped to the waist. On passing from a hot to a cold place a heavy coat is used to protect the heated body. Wet clothes are either removed before going to the surface, or trousers and coat worn over them. All miners take hot and cold showers after coming off shift. Frequent drinking, and the bathing of the hands, wrists, arms, and head in ice-water are resorted to in all hot workings, and undoubtedly serve to keep down the body-temperature, while temporarily refreshing. Frequent rests are taken in special cooling-rooms, so placed in the workings as to receive freshest and coolest air. In exceedingly hot workings cold water from a hose is turned upon the miner while at work.

The effect of the underground conditions upon the blood was made the subject of a preliminary study by Prof. P. Frandsen at the author's request. His results are of importance, and are given in full in an accompanying note. His conclusion, given tentatively, is that 'the conditions in the deep workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. From a comparison of a series of experiments made it appears that the main permanent effect is an increased haemoglobin-content of the individual red blood-corpuscles.'

Conditions are not favourable for a study of the comparative efficiency of miners working under normal temperatures and under those in the lower levels of the Comstock. The following conclusions are the results of the author's experience: Moderately high temperatures, from 95° to 105° F., with moderate humidities, from 50 to 70% relative humidity, and with air-currents of velocities from 200 to 300 ft. per min., do not prevent efficient work nor are they particularly uncomfortable. A higher temperature, from 110° to 115°, together with the same conditions as above, decreases efficiency to a considerable extent. A higher temperature, from 110° to 115°, with high humidity and moderate velocity air-currents, very greatly impairs the miners' efficiency; and a still higher air-velocity, under the same conditions, renders workings more bearable, but miners cannot work very long at one time. A moderately high temperature, from 95° to 105°, in a saturated atmosphere with no current, becomes very trying. Prolonged exposure with much exertion is dangerous. A moderate temperature, from 90° to 98°, and saturated air-currents of a velocity of from 400 to 500 ft. per min., with more or less vitiated air, are conditions which are very trying and give a low labour efficiency. Vitiated air will impair labour efficiency to a greater extent than a high temperature."—Prof. G. J. YOUNG.—*Trans. Amer. Inst. Min. Eng.*, Nov., 1909, p. 955. (J. A. W.)

ESTIMATED COST OF SHAFT SINKING.—"*General Statement.*—The following figures and computations have been made upon the assumption that the proposed shaft will be 6 ft. × 16 ft. within timbers and reach a vertical depth of 1,000 ft. For a vertical shaft, its total length would be 1,000 ft.; if inclined at an angle of 45° to follow the dip of the formation, its total length would be 1,400 ft. to reach a total vertical depth of 1,000 ft. A contract price of $40 per ft. for sinking would apply only if the shaft was put down in the jasper formation. If diorite is the material to be dealt with the contract price for sinking alone would be between $50 and $55 per ft., while all the other items would remain the same.

A slight difference in the cost of timber would appear in the amount of timber used in a vertical or inclined shaft, as the latter requires 10 in. × 10 in. stringers, while the former would take 6 in. × 8 in. skip runners, but this has not been taken into account.

The maximum flow of water to be handled (800 gallons per minute) is probably somewhat high.

A shaft put down to a depth of 1,000 ft., at the end of the ore formation would make available approximately 3,000,000 tons of ore now in sight, at the shafts now in operation, at the present stage of development of the mine. About one-third of this tonnage must now be left as a pillar to maintain the shaft.

If the new shaft costs $150,000, this would make a cost of 5 cents per ton on 3,000,000 tons of ore.

Ditails of Sinking Contract, Shafts, etc.

	Maximum	Minimum
Contract price for sinking, $40 per ft., includes drilling, blasting, powder, caps, fuse, etc. ..		
For a vertical shaft, 1,000 ft. ...		$40,000
For an inclined shaft, 1,400 ft. ...	$56,000	
Computing all work for a shaft 6 ft. × 16 ft. within timbers, three compartments, shaft sets 5 ft. centers; and assuming 40 ft. per month as average sinking		
1,000 ft. would take 25 months, allow 26 months		
1,400 ft. would take 35 months, allow 36 months		
Allowing 25 working days per month, 300 per year		
1,000 ft. of sinking would take 650 days		
1,400 ft. of sinking would take 900 days		
Sinking contract would be worked on three 8-hour shifts. All other labour, one or two 10-hour shifts		
Cutting pump station 10 ft. × 15 ft. × 20 ft. at a depth of 500 ft. ...	350	350
A maximum flow of 800 gal. per min. when bottom of shaft is approached is used as the basis of pumping expenses		
Total	$56,350	$40,350

Details of Labour.

	Maximum	Minimum
Blacksmith, at $2.25 per day; helper at $1.65 per day; $3.90 per day for 900 days	$3,510	
$3.90 per day for 650 days ...		$2,535
Two landers at $1.70 per day for 900 days	3,060	
Two landers at $1.70 per day for 650 days		2,210
Two timbermen at $1.75 per day. Assuming one set of timber can be cut and framed in 1 day by two men		
1,400 ft. of shaft, 280 sets, 280 days	980	
1,000 ft. of shaft, 200 sets, 200 days		700
Two brakemen at $2.20 per day, for 900 days	3,960	
Two brakemen at $2.20 per day, for 650 days		2,860
Two firemen at $1.70 per day, for 900 days	3,060	
Two firemen at $1.70 per day, for 650 days		2,210
Allow one-fourth of mining captain's time, $25 per month, for 36 months	900	
Allow one-fourth of mining captain's time, $25 per month, for 26 months		650
Surveyor and helpers, allow $20 per month, for 36 months	720	
Surveyor and helpers, allow $20 per month, for 26 months		520
Total	$16,190	$11,685

Details of Timber.

	Board Measure ft.	Max.	Min.
2 plates, 12 in. × 12 in. × 18 ft. contain	432		
2 end pieces, 12 in. × 12 in. × 6 ft. contain	144		
4 corner posts, 12 in. × 12 in. × 4 ft. contain	192		
2 dividings, 10 in. × 12 in. × 6 ft. 4 in contain	126⅔		
4 Stringers, 10 in. × 10 in. × 5 ft. contain	166⅔		
4 centre posts, 10 in. × 10 in. × 4 ft. contain	133⅓		
Sheathing, 3 in. × 5 ft. × 44 ft. contain	660		
Boards, 1 in. × 5 ft. × 6 ft. 4 in. contain	31⅔		
Total amount of timber for 1 set	1,886⅓		

Total amount of timber in 280 sets	535.173		
Total amount of timber in 200 sets	377.267		
At $14 per thousand for hemlock timber:			
For 280 sets, $7,492.42, allow...		7.500	
For 200 sets, $5,181.74. allow...			5,200
Ladders, 17½ cents per foot, 1,400 feet		245	
Ladders, 17½ cents per foot, 1,000 feet			175
Total		$7,745	$5,375

Details of Rails, Pipe, Tie-Ropes, etc.

	Maximum	Minimum
45-lb. rails for double skip road, allow 1,500 ft., making 6,000 linear feet or 2,000 yd., 45 tons at $25...	$1,125	
200 pairs fish-plates, at 21 cents ...	42	
500 lb. rail spikes, at 3 cents ...	15	
1,120 1⅛ in. × 6 ft. 6 in. round iron tie-ropes, at 2 cents, $487.46 allow	500	
2,240 nuts and washers, $278.60, allow	300	
All of the above computed for 1,400 ft. of shaft. For a vertical shaft of 1,000 ft. depth, rails, fish-plates and spikes would not be used		
Tie-rods, nuts, and washers for 1,000 ft. of shaft allow		550
1,400 ft. of 10-in. water-column pipe	1,820	
1,000 ft of 10-in. water-column pipe		1,300
Drill steel	40	30
2-in. steam pipe; allow 250 ft. in excess of length of shaft. For 1,650 ft.	150	
For 1,250 ft.		110
4-in. air pipe: allow 250 ft. in excess of length of shaft. For 1,650 ft.	750	
For 1,250 ft.		575
Allow a maximum of 5 tons of coal per day for 26 months		2,280
For 36 months...	3,285	
3,000 lb. 60d. spikes at 10 cents per pound for 1,400 ft.	300	
200 lb. 10d. nails at 10 cents per pound, for 1,400 ft.	20	
2,160 lb. 60d. spikes at 10 cents per pound, for 1,000 ft.		216
145 lb. 10d. nails at 10 cents per pound, for 1,000 ft.		15
Total	$8,347	$5,081

Details of Plant.

	Maximum	Minimum
Shaft house and pockets	$14,800	$6,750
Engine house	9,625	5,000
Boiler house	4,200	3,500
Powder house	250	100
Coal trestle	3,000	2,500
Boilers (4)	11,660	10,500
Hoisting engine	15,000	12,000
Compressor	11,500	10,625
Skips (2)	1,000	300
Two No. 3 Rand drills, complete ...	375	375
Auxiliary pump at 500-ft. depth ...	6,000	5,000
Sinking pump	1,000	900
Temporary equipment at start, small hoist, bucket, rope, tripod, etc., allow...	1,500	1,000
Hoisting cable, 1⅜-in. diameter ...	900	700
Incidentals—Teaming, pipe fittings, air hose, picks, shovels, hammers, wrenches, timber cutter's tools, axes, saws, oil, waste, candles, temporary bell signal system, etc.	5,000	4,500
Total	$85,810	$63,750

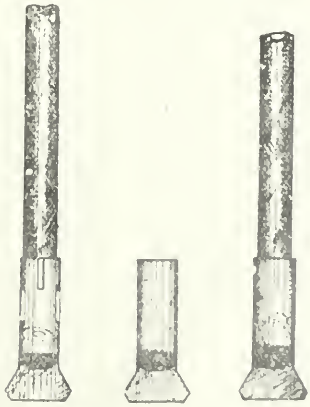
Recapitulation.

	Inclined shaft 1,400ft	Vertical shaft 1,000ft
Sinking contract	$56,350	$40,350
Blacksmithing	3,510	2,535
Landers...	3,600	2,210
Timber cutters	980	700
Brakemen	3,960	2,860
Firemen	3,060	2,210
Captain and surveyors	1,620	1,170
Timber and ladders	7,745	5,375
Rails, fish-plates, spikes	1,182	
Air and steam pipes and water column	2,720	1,985
Tie-rods, nuts and washers ...	800	555
Nails and spikes...	320	231
Coal	3,285	2,280
Drill steel	40	30
	$88,632	$62,491
Total plant, maximum		$85,810
Total plant, minimum		$63,750
Inclined shaft with minimum plant ...		$152,382
Inclined shaft with maximum plant ...		$174,442
Vertical shaft with minimum plant ...		$126,241
Vertical shaft with maximum plant ...		$148,300

—*Mines and Minerals*, Nov., 1909, p. 256. (A. R.)

A SIMPLE, SAFE FUSE LIGHTER.—"The illustration herewith readily illustrates the Eagle fuse lighter, a perfectly safe, simple and sure method of dealing with the various conditions under which unexploded charges of dynamite, often unsuspected, result in death and ruin, which the miner has long dreaded.

The igniter is of a highly inflammable material, forming a short cylindrical cap having the diameter of the standard makes of fuse, and so closed at one end as to leave a short tab for convenience for lighting. In the cap thus formed is placed a composition inflammable, but not highly explosive, by which a sufficiently quick and strong fire is assured.



The end of the fuse is rolled smooth between the thumb and finger and pressed into the cap or igniter. The application of a flame or incandescent heat to the igniter then assures ignition of a fuse that will burn to the detonator at its firing end. Both the cap and the composition are water-proof, and it never misses fire.

The system eliminates the uncertainties of blasting, and its use is economical in point of time, money and life."—*Mining World*, Nov. 13, 1909, p. 988 (C.B.S.)

LIMITS OF DEEP MINING.—"The trial borings of northern Westphalia have been pushed considerably beyond 1,400 metres; at Paruchowitz, near Rybnik, in upper Silesia, a boring went down 2,003·34 m. (6,572·6 ft.). We do not, however, think it advisable for the present to search for coal measures beginning at greater depths below the surface than 1,100 to 1,200 metres, as such very deep deposits cannot as yet be considered as workable. An initial depth of 1,200 metres is already very considerable indeed, and infers actual mining down to a depth of at least 1,500 metres below the surface. It is very questionable whether working a coal mine at such a depth would be remunerative or even possible.

It is quite true that far more considerable depths have been attained in the Tamarack copper mines of North America, and that, in Alpine tunnelling, work has been done successfully at depths even exceeding 2,000 metres below the surface, but, nevertheless, I am of the opinion that in strata of the nature of the coal measures such depths are at present not suited to practical mining, considered as a paying enterprise. A depth of 1,500 metres seems to me to be the extreme limit.

Apart from the difficulties presented by the nature of overlying strata and by inflammable gas, which vary exceedingly according to localities, the following facts are obstacles to very deep working:—high temperature, excessive crush, and cost.

Temperature.—The temperature of the earth's crust increases as we descend in a proportion which, though not the same for all localities, on the whole seems pretty constant. This high temperature becomes a decided obstacle at slightly varying depths, somewhere below 1,000 metres under the surface. Somewhere about this point the earth's temperature may be considered to average 36° C.—equal to the heat of the blood.

The temperature of the rocks, however, may be much higher than that of the air in the workings near by, as the latter may be greatly reduced by ventilation and cooled more or less in proportion to the quantities of cold air forced through the mine. The rate at which the fresh air is warmed by absorption of heat from the warm walls of the rock, also seems to depend in a large degree on the system of working. In a cellular, viz., the pillar and stall system, the air seems to acquire almost immediately the full temperature of the coal; but under the longwall system the air appears to retain much longer a considerably lower temperature in sweeping past the working face.

This cooling effect of ventilation, however, is greatly affected by the degree of humidity of the workings, as the water draining from the rocks has at least the temperature of the earth, and by its great specific heat warms the air much more considerably than the strata, which being bad conductors, are very soon isolated by a thin cooled crust, whilst in a wet mine the supply of warm water is continually kept up.

If strong springs are met with, often of still higher temperature than the rocks, the difficulties increase very much, and soon become insurmountable. In damp air heat also becomes much sooner unbearable than in the dry atmosphere, as active evaporation of the perspiration is a great help in keeping down the temperature of the body. Too dry air, however, causes new inconveniences in coal mines, by the nuisance and even grave dangers caused by coal dust.

Apart from very large faults, experience seems to show that in generally soft strata, like coal measure shales, the percolation of water soon stops at some depth, the fissures and pores in the rocks being closed by the pressure. Hard and thick beds of coarse sandstone, however, may remain conductors of water for great distances and more considerable depths. Generally, however, the workings are quite dry in very deep coal mines.

There is a serious obstacle which, in some coal pits counteracts the effects of ventilation in keeping down the temperature, viz., the oxidation of the coal as a source of heat. This oxidation is specially active with some qualities of coal, and with fine dust, and very often the heat generated is great enough to cause spontaneous combustion. This property of some coals will probably prove an insurmountable obstacle to mining them at very great depths.

Effect of Depth upon Crush.—The effect of the accumulative weight of several hundred metres of strata cannot be spoken of with any certainty, nor does it show itself in the same manner in different localities, because it is highly influenced by the physical properties of the rocks. It remains a generally observed fact, however, that pressure considerably increases at great depths. Strata of the nature of coal measure shales act like a very stiff and plastic mass under very severe pressure, and gradually but irresistibly fill up all cavities made in them. At very considerable depth no kind of support can withstand this crush, and the galleries have to be re-widened continually, as all walls slowly creep inwards. These effects moreover depend greatly on the system of working and specially on the care with which old workings are filled up, preventing as much as possible the strata from becoming loosened.

Massive sandstones above the coal seams also help the support and take off a great deal of the immediate pressure, causing it to come on more gradually, and becoming active only after the coal has been taken out.

However, at considerably greater depths than 1,000 metres the crush becomes a serious nuisance in a great many coal mines, and may gradually increase to such extent as to prevent deeper progress.

Effect of Depth upon Cost.—In several cases, though it may be technically possible to work at certain depths, costs may become too great, the work ceasing to pay. The profit to be made on mining coal is dependent on so many circumstances that no special depth may be said to limit mining enterprise for reason of cost. Besides what is unprofitable now may soon become remunerative, after the supplies of shallower mines have been exhausted and competition ceases from that quarter, or better methods of haulage, mechanical cutting, or ventilation have been invented.

That cost increases with depth can easily be understood, as high temperature affects the efficiency of the miners, at the same time reducing the hours the workmen can stay in the mine without being inconvenienced. Winding and ventilation become more costly, likewise the expenses of timbering and of maintaining air passages and haulage roads increase. Nevertheless, this increase of costs is not yet greatly felt in the deeper mines of Westphalia.

As a rule, therefore, the richer the deposit is in coals of good quality, the longer deep working will pay. The British Coal Commission of 1871 put down the maximum depth at 4,000 ft., and the minimum thickness of workable seams at 12 ins., but at very great depths it seems advisable to consider only seams of 2 ft. or more in thickness as workable; where the deposit is very regular and the roof good enough to allow the use of electric coal cutting machinery even at great depths, or the coal is of exceptional quality, thinner seams may possibly be worked with advantage.

From all this it will be clear that it seems quite sufficient for the present to extend the explorations to an initial depth of 1,200 metres (4,000 ft.) below the surface."—W. A. J. M. VAN DER GRACHT, Memoirs Geological Exploration of the Netherlands. *London Mining Journal.*—Nov. 20, '09, p. 278. (A. R.)

CONCRETE FOR SHAFT TRACKS.—"The extended use of reinforced concrete for mine work on the Rand is a noteworthy feature at the present time, and especially is this true of new equipments. Concrete now seems to be utilised wherever practicable. Its extensive employment in the metallurgical works of the City Deep plant has already been dealt with, and we are now able to give particulars of the use it is being put to at the Bantjes Consolidated Mines at Florida.

At this property it is being utilised in a new direction, new as far as the Rand is concerned at any rate, with the exception of some similar work done at the Angelo Deep two or three years ago. But as this Angelo Deep work does not appear to have attracted much attention, the employment of concrete in two shafts of the Bantjes mine is a most interesting matter. The central incline has been stripped and enlarged, and the shaft is now 32 ft. 4 in. wide 'over all,' instead of 17 ft. as formerly. Five compartments are now prepared, and there will be four hauling ways instead of two only in the old shaft. Concrete blocks are now being carried down the shaft, which dips at an angle of 35°. The skip rails run on these, and the gauge has been increased from 3 ft. to 4 ft. 2 in. The central concrete blocks, which carry two rails, are 2 ft. 9 in. in width at the top, whilst the outside blocks, on each of which only one rail rests, are 12 in. wide at the top. Only one divider is being put in between the two pairs of skip roads. Sixty pound rails are used, and strong steel supports are employed in the shaft, which will contain very little timber; in fact, no timber at all. The shaft is indeed an excellent piece of work, and it should be possible to haul in it at the rate of a mile a minute with a minimum of risk.

The advantages claimed for this method of supporting skip rails are: (1) Higher rate of hoisting; (2) greater safety; (3) a cleaner shaft; (4) much more permanent work; (5) less wear and tear on the rope. The cost is estimated at less than that of a wooden road, and the work of laying the concrete proceeds very rapidly. At a little depth in the shaft, where solid rock exists, the concrete is mixed on the floor of the shaft and put in the skip road direct."—*The South African Mining Journal*, Jan. 15, 1910, p. 494. (A. R.)

MISCELLANEOUS.

APPLICATIONS OF CONCRETE IN THE METAL-MINING INDUSTRY.—"Concrete, used for the mere purpose of supporting unsafe ground, is gaining but slowly in favour, being confined for the present to an occasional tunnel portal, to lining shafts through surface soil down to solid rock, and to a very limited use of 'stulls.' In the coal-mining industry much greater use of concrete for all purposes is made.

The chief advantage of concrete for lining shafts through the surface soil and down to bed-rock, outside of its durability, is its efficiency in keeping out water. The upper part of the lining is usually so shaped and extended as to form the foundations for the hoisting machinery. Arrangement for supporting the ordinary shaft equipment of cage-guides, ladders, pipes, etc., must be well thought out beforehand in order to avoid having to cut into the concrete after it has set. Shaft linings are comparatively simple in vertical shafts, but inclined shafts do not lend themselves quite so easily to placing the forms and ramming the concrete, but not even these require any very high order of ingenuity. In this latter case the upper part had perhaps better be made arching and, if possible, reinforced either with specially purchased iron or with whatever old material may be available on the spot, such as old rail or pipe. A neat method for shaft lining is to make slabs of concrete three or four inches thick by two or three feet square, which when properly hard are placed and supported against the sides of the shaft, the space between them and the soil or rock being filled with concrete well rammed in; these slabs are left permanently in position.

It is as a means of controlling water underground that concrete is growing in favour rapidly. It is a much better stroke of business to keep water out of a mine than to pump it out. As an instance, at the Candelaria mine, in Mexico, the great influx of water was found to be at a point midway between the first and second levels, about 160 ft. from the surface. A portion of this water was very ineffectually trapped on the second level by means of wooden troughs and pumped thence to the surface, but the great bulk of the water found its way to the bottom of the shaft at 700 ft. Raising the water from this depth to the surface necessitated a continuous effort at the pump to overcome some 630,000 foot pounds. What with condensation in such a long steam line, pipe friction, etc., this amounted to something over 30 horse-power at the boiler. It was therefore decided to arch over the second level from the shaft completely to the end. This being done, the water, thus completely confined above the arch, was pumped only 200 ft. to the surface. The pump at the bottom of the mine was then able in a few hours work per day to take care of all the water seeping into the mine below this level. The saving in reality was found to be more in money than was represented by the steam saved, the men being much more efficient in the more agreeable circumstances, besides the heavy bills for repairs to the pump being much diminished. The forms used in this work consisted of two segments from an old steam boiler, lapped over to one another to a greater or less degree to accommodate the variable width of the level, the smaller irregularities being closed by scraps or boards or sheet iron or anything else that came handy. As elegance and finish are not considered necessary underground the marks left by these were not smoothed off. The preliminary work, having for its object the preventing of the water from washing over the wet concrete, was much more difficult and engaged much more attention than the mere construction of the arch.

At the San Blas mine, also in Mexico, in sinking a shaft 8 ft. by 12 ft. in area, a heavy flow of water was encountered at 90 ft. from the surface, through a fissure which crossed the shaft at about a 25° angle. The flow of water was such that in spite of the most energetic efforts only 8 ft. of progress was made in a month. The shaft was enlarged in an irregular outline, for two or three feet on each of its four sides for a belt of about three or four feet above the fissure and about the same distance below. After the water had risen to its natural level in the shaft, some 250 tons of concrete were dumped in and allowed several weeks to set. The water was then pumped out and the shaft re-excavated through the concrete and carried down to 400 ft. without further trouble. Had a central hollow core been left in the concrete the work of sinking through it would have been very much diminished. The total cost of this work did not exceed 9s., including the re-excavation of the shaft, the cement costing, delivered at the spot, 18s. per barrel.

It may be as well to indicate the precautions necessary to secure a successful issue to a piece of work of the above sort. First, the water should be without movement, or the flow be at least imperceptible. For this reason the shaft was allowed to fill up to its natural water level. Second, the concrete should not be dumped in and allowed to fall to the bottom, causing a separation of the heavier from the lighter ingredients. In the case cited the concrete was lowered in a wedge-shaped bucket, one side of which was hinged at the top and fastened at the bottom by a latch to which was attached a wire leading to the surface. When the bucket was lowered and placed as nearly as possible in position the wire was pulled, releasing the concrete without giving the cement much opportunity to separate from the other ingredients. Third, the re-excavation must be made without the aid of explosives. Fourth, ample time must be allowed for the concrete to solidify

At another mine, water entered at a point where the vein crossed a considerable stream. On occasions in summer time the bed of the stream ran dry. At such periods the mine pump was almost idle. During one dry season the bed of the stream was cleaned of all boulders and sand and other loose material for about 10 ft. up-stream and the same distance downstream above and below the vein crossing. This space was then covered with a pavement 6 in. thick, effectually and permanently doing away with nearly all the pumping expenses.

A good instance of how not to do a job of this sort was at the Herrarias mine in the South of Spain. The river Almanzora runs through the property, and in following a small vein of rich silver ore the river gained entrance into the workings. A ship load of cement was dumped into the river, being thrown at random over the ship's side over the place where the leak was supposed to be—but, of course, absolutely without effect on the mine pumps. The final result was that a valuable mine was abandoned for a number of years.

For 'timbering' underground, concrete 'timbers' are attracting a certain amount of attention. The average life of a pine timber or stull in a mine may not be counted on to exceed four years, while the life of a concrete 'timber' is indefinite; it costs more in the first instance and, owing to its weight, it is more costly to place in position. Concrete timbers are easily made in long wooden troughs, or in split tubes of sheet iron, with the ends adjusted so that they can be changed to suit the desired length and angle of cut. If it is possible to do so, the form should be set close to where the stull is to be finally placed. The conspicuous disadvantage of the concrete stulls is the difficulty in placing them at the proper angle with regard to the line of thrust of the ground to be supported. Frequently when the ground being supported settles to a small extent the angle of thrust is greatly changed; in such a case a pine timber is easily wedged up, or the ends cut to a new angle, or the position of the whole thing changed to suit the new conditions; once a concrete timber is in place any changes are very difficult. At the present moment, the greater use of concrete stulls and timbers is in replacing rotted ones. In this case, the ground having already settled and the wooden stulls having been subjected to observation for a considerable period of time, the concrete props can be placed with more intelligence, not only as to the direction of the lines of force but also as to the amount of strain they are to be called upon to resist. Both of these factors are usually only partially known in the first instance.

As a general proposition, the use of reinforcement, either of scrap material or of some one of the many special shapes of iron, is not very usual for underground work in the metal-mining industry. For surface structures, however, all the usual rules of such constructions hold good. Portions of the various structures, in which the stresses are exclusively in compression do not require reinforcement.

These statements suppose the use of cement of good quality, which, in out-of-the-way places, is not always obtainable. It is of very little use to attempt to test the cement for the purpose of comparing the results obtained with the figures obtained by the manufacturers. So many factors are not uniform with the conditions under which the experiments are made at the factory that the final results are quite different. The age of the cement, difference of temperature and humidity of the air, of the water or of the cement itself, and even the nature and amount of the various mineral substances dissolved in the water, altogether introduce such differences as to make it a waste of time to make any other tests than the actual mixing and laying of a small preliminary batch of concrete. About the only exterior indication of defective quality is the swelling of the barrels when they have been too long in stock in a humid climate or have accidentally been wetted. It seems a pity that no way can be found to stamp each barrel reliably with the date it left the factory, like the photographic supplies.

As a general rule there is nothing at any mine in the way of surface structures that cannot be made, and better made, of concrete than of any other material—head frames, ore bins, floors, buildings, electric-wire, poles, trestles, water pipes for low pressure, boiler settings, chimney stacks, tanks of all kinds, and even boats and lighters if the mine have use for these latter."—HENRY W. EDWARDS.—*Engineering Magazine*, Nov., 1909, p. 175. (J. A. W.)

SNAKE BITES should be treated by ligaturing the bitten part at once with a tourniquet of cloth or rubber, and injecting locally a 1% solution of potassium permanganate. Alcoholic stimulants and ammonia are absolutely harmful in critical cases. They are of value as restoratives only in cases where the person has not been fatally bitten.—*Mining and Scientific Press*, Dec., 1909. (C. B. S.)

CASE HARDENING OF STEEL WITH GASES.—"This paper gives the results of experiments on case hardening of soft Norwegian iron with various gases, carried on at the Polytechnic Institute. It was shown that in most cases nitrogen in the form of ammonia facilitates the action of carbonising gases. Nitrogen in the free state has no influence on the carbonising action. It was shown that the ammonia does not react with the carbon gases to form cyanogen. Carbon monoxide seems to act almost as well without as with ammonia. The depth to which the carbon penetrates and the percentage of carbon in the case were determined by analysis. The chemical reactions taking place during the carbonisation were determined as follows:—

With carbon monoxide $2CO = C + CO_2$
With methane $CH_4 = C + 2H_2$.

With acetylene two reactions seem to take place, as follows:—

$$2C_2H_2 = CH_4 + 3C$$

and

$$C_2H_2 = C_2 + H_2.$$

The carbonising ability of the gases studied ranges according to the following order: Carbon monoxide, acetylene, illuminating gas, methane. Carbon monoxide is by far the best gas for this purpose."—OLSEN and WEIFFENBACH.—*Industrial and Engineering Chemistry*, Aug., 1909, p 624. (A. McA. J.)

TEMPERATURE CONVERSION TABLE—F, C.—CENTIGRADE—FAHRENHEIT.—The following table, which is self explanatory, covers the whole temperature range from the temperature of absolute zero to that of the electric arc.

C°	0	10	20	30	40	50	60	70	80	90
	F	F	F	F	F	F	F	F	F	F
− 200	− 328	− 346	− 364	− 382	− 400	− 418	− 436	− 454		
− 100	− 148	− 166	− 184	− 202	− 220	− 238	− 256	− 274	− 292	− 310
− 0	+ 32	+ 14	− 4	− 22	− 40	− 58	− 76	− 94	− 112	− 130
0	32	50	68	86	104	122	140	158	176	194
100	212	230	248	266	284	302	320	338	356	374
200	392	410	428	446	464	482	500	518	536	554
300	572	590	608	626	644	662	680	698	716	734
400	752	770	788	806	824	842	860	878	896	914
500	932	950	968	986	1004	1022	1040	1058	1076	1094
600	1112	1130	1148	1166	1184	1202	1220	1238	1256	1274
700	1292	1310	1328	1346	1364	1382	1400	1418	1436	1454
800	1472	1490	1508	1526	1544	1562	1580	1598	1616	1634
900	1652	1670	1688	1706	1724	1742	1760	1778	1796	1814
1000	1832	1850	1868	1886	1904	1922	1940	1958	1976	1994
1100	2012	2030	2048	2066	2084	2102	2120	2138	2156	2174
1200	2192	2210	2228	2246	2264	2282	2300	2318	2336	2354
1300	2372	2390	2408	2426	2444	2462	2480	2498	2516	2534
1400	2552	2570	2588	2606	2624	2642	2660	2678	2696	2714
1500	2732	2750	2768	2786	2804	2822	2840	2858	2876	2894
1600	2912	2930	2948	2966	2984	3002	3020	3038	3056	3074
1700	3092	3110	3128	3146	3164	3182	3200	3218	3236	3254
1800	3272	3290	3308	3326	3344	3362	3380	3398	3416	3434
1900	3452	3470	3488	3506	3524	3542	3560	3578	3596	3614
2000	3632	3650	3668	3686	3704	3722	3740	3758	3776	3794
2100	3812	3830	3848	3866	3884	3902	3920	3938	3956	3974
2200	3992	4010	4028	4046	4064	4082	4100	4118	4136	4154
2300	4172	4190	4208	4226	4244	4262	4280	4298	4316	4334
2400	4352	4370	4388	4406	4424	4442	4460	4478	4496	4514
2500	4532	4550	4568	4586	4604	4622	4640	4658	4676	4694
2600	4712	4730	4748	4766	4784	4802	4820	4838	4856	4874
2700	4892	4910	4928	4946	4964	4982	5000	5018	5036	5054
2800	5072	5090	5108	5126	5144	5162	5180	5198	5216	5234
2900	5252	5270	5288	5306	5324	5342	5360	5378	5396	5414
3000	5432	5450	5468	5486	5504	5522	5540	5558	5576	5594
3100	5612	5630	5648	5666	5684	5702	5720	5738	5756	5774
3200	5792	5810	5828	5846	5864	5882	5900	5918	5936	5954
3300	5972	5990	6008	6026	6044	6062	6080	6098	6116	6134
3400	6152	6170	6188	6206	6224	6242	6260	6278	6296	6314
3500	6332	6350	6368	6386	6404	6422	6440	6458	6476	6494
3600	6512	6530	6548	6566	6584	6602	6620	6638	6656	6674
3700	6692	6710	6728	6746	6764	6782	6800	6818	6836	6854
3800	6872	6890	6908	6926	6944	6962	6980	6998	7016	7034
3900	7052	7070	7088	7106	7124	7142	7160	7178	7196	7214
C°	0	10	20	30	40	50	60	70	80	90

C°	F°
1	1·8
2	3·6
3	5·4
4	7·2
5	9·0
6	10·8
7	12·6
8	14·4
9	16·2
10	18·0

F°	C°
1	0·56
2	1·11
3	1·67
4	2·22
5	2·78
6	3·33
7	3·89
8	4·44
9	5·00
10	5·56
11	6·11
12	6·67
13	7·22
14	7·78
15	8·33
16	8·89
17	9·44
18	10·00

Examples: 1347° C = 2444° F + 12·6 F = 2456·6 F : 3367° F = 1850 C + 2·78 C = 1852·78 C.
—Dr. LEONARD WALDO.—*Metallurgical and Chemical Engineering*, March, 1910, viii., 3, 123. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

JOURNAL OF THE INSTITUTE OF METALS. Vol. II., 1909. Edited by G. SHAW SCOTT, Secretary. Cloth. 8vo., pp. 250. Ill. (London: The Institute of Metals, Caxton House, S.W. Price One Guinea.

"The second volume of the transactions of the Institute includes the papers read at the meeting held at Manchester in October last. These papers discuss the following subjects: Aluminium-copper-tin alloys; surface appearance of solders; causes of corrosion of copper and brass; study of copper-zinc alloys; elastic breakdown of non-ferrous metals; production of zinc by the Brand process; welding of aluminium; and the technical assay of zinc.

These volumes are to be published half-yearly and we recommend that members and others should peruse them carefully, instead of relying on each paper as presented at the meetings and such outline of the discussion as may appear in the current Press. There is a vast amount of additional information in the volume, in the way of full reports of the discussion, written contributions, and authors' replies and addenda. The Institute does not get quite satisfactory treatment by the reporters that attend the meetings on behalf of the Press. It is no matter of surprise that this should be so, for the subjects are highly specialised, and a reporter, even though well versed in general technology, finds it difficult to do justice to the discussion. It is a pity that the Institute's volumes cannot be published at an earlier date after the meetings, for interest in the discussions is apt to drag. The American Institute of Mining Engineers, which holds two meetings a year on the same plan as the Institute of Metals, found it the best policy to issue a monthly bulletin so as to keep the publication of papers and the discussions upon them up to date This method has the additional advantage of making it possible to distribute some of the papers in advance of the meetings at which they are to be read and thus give members more preparation for the discussion. A new feature introduced into the current volume are the pages devoted to short abstracts and notices of papers relating to the non-ferrous metals that have appeared in the Press and in the transactions of other societies." — *Mining Magazine*, March, 1910, p. 238. (A. R.)

"FIRST AID" TO THE INJURED AND SICK. By F. J. WARWICK, M.B., and A. C. TUNSTALL, M.D. Sixth Edition. 1s. (Bristol: John Wright & Sons, Ltd.)

"We do not know of any other book on 'First Aid' so complete and full of useful information as this, and the fact of the sixth edition being called for within 18 months of the publicatiion of the fifth edition proves the demand for such a work. The present edition has been brought thoroughly up to date, and is abundantly illustrated, 36 new plates having been introduced.

It is important that a book of this description should be complete, concise, and easy of reference, and the authors have been very successful in fulfilling these requirements, but in some instances we think a fuller description of treatment is necessary. This is especially so with regard to the treatment of electric shock; a few illustrations (such as those published in 'First Aid' to Miners,' by W. B. Arthur) would be very useful.

The book is a handy size, strongly bound and thoroughly well produced. We can strongly recommend it to all interested in the subject."—*London Mining Journal*, March 19, 1910, p. 365. (A. R.)

GEOLOGICAL AND ARCHÆOLOGICAL NOTES ON ORANGIA. By J. P. JOHNSON. 10s. (London: Longmans, Green & Co.)

"Mr. J. P. Johnson's book, entitled 'Geological and Archæological Notes on Orangia,' is an indispensable, more or less scientific guide to the Orange River Colony, or Free State, as it will be called under the Union. The term 'Orangia' should be deprecated.

The stratigraphical outlines of the Venterskroon and New Rand auriferous areas are as detailed as could be expected in a book dealing with so many and varied subjects.

As regards the New Rand area referred to on p. 7, No. 9 borehole has been completed, Nos. 10 and 11 sunk, and No. 12 is on the point of starting, with every indication that the payable auriferous conglomerates will be finally located therein.

The information regarding the diamond mines, which is focussed from several sources, is given in concrete and useful form, and shows that the occurrence of diamonds in the Colony is widespread.

The Pre-Historic Period is one to which Mr. Johnson has personally given much attention. His descriptions of the stone implements of South Africa, and of the rock paintings by Kafirs, are particularly interesting.

His chapter on farming should be of use to intending settlers in the Colony."—*London Mining Journal*, March 12, 1910, p. 321. (A. R.)

Blackhall, R. H. Up-to-Date Air Brake Catechism. 23rd Ed. Cr. 8vo. *Spon.* Net 8s. 6d.

Chesnau, M. G. Theoretical Principles of the Methods of Analytical Chemistry. 8vo. *Macmillan.* Net 7s. 6d.

Corderio, F. J. B. The Atmosphere, its Characteristics and Dynamics. 4to. *Spon.* 10s. 6d.

Desch, Cecil H. Metallography. Cr. 8vo., pp. 442. *Longmans.* 9s.

Dodge, G. F. Diagrams for Designing Reinforced Concrete Stuctures. Folio, boards. *Spon.* Net 17s.

Goldingham, A. H. The Design and Construction of Oil Engines. 3rd Ed., Revised and Enlarged. Cr. 8vo., pp. 280. *Spon.* Net 10s. 6d.

Ibbetson, W. S. Practical Electrical Engineering. For Elementary Students. Cr. 8vo., pp. 168. *Spon.* Net 3s. 6d.

Matheson, Ewing. The Depreciation of Factories, Mines and Industrial Undertakings, and Their Valuation. 4th Ed., Revised and Enlarged. 8vo, pp. 242. *Spon.* *Net* 10s. 6d.

Mennell, F. P. An Introduction to Petrology. 2nd Ed., 8vo, pp. 212. *Gerrards.* Net 8s.

Mennell, F. P. The Miners' Guide Book. A Practical Handbook for Prospectors, Working Miners, etc. Cr. 8vo, pp. 204. *Gerrards.* Net 4s.

Nedden, Dipling F. Zur. Engineering Workshop Machines and Processes. 8vo., pp. 232. *Constable.* Net 6s.

Schwarz, E. H. L. Casual Geology. 8vo., pp. 256. *Blackie.* Net 7s. 6d.

Swann, W. F. G. On the Specific Heats of Air and Carbon Dioxide at Atmospheric Pressure by the Continuous Electric Method at 20° C. and 100° C. 4to. sd. *Dulau.* Net 2s.

Thomson, W. C. Bridge and Structural Designs. Cr. 8vo. *Constable.* Net 8s.

Tyrvell, H. G. Concrete Bridges and Culverts. 12mo, leather. *Spon.* Net 12s. 6d.

Workshop Receipts for Manufacturers and Scientific Amateurs. New and Thorough Edition. Vol. 3. Jointing Pipes, Pumps. With 257 Illustrations. Cr. 8vo, pp. vi.-528. *Spon.* Net 3s.

Abstracts of Patent Applications.

(C.) 586/09. Frank Ondra (1), Alfred Adair (2). Improvements relating to the treatment of zinc solutions, for the recovery of metals or metal compounds therefrom. 20.12.09.

This invention applies more specially to the zinc solutions produced in the treatment of zinc-gold slimes, and its object is the recovery of the sundry metals, base or otherwise contained therein.

The reagents used are ferrous and copper salts, the sulphides of zinc, iron or manganese and lime either as chloride, carbonate or hydrate.

(C.) 286/09. Ernest Slatter Day. Improvements in the continuous treatment of slime and apparatus therefor. 16.6.02.

This application refers to improvements in the treatment of slime by continuous decantation. The settlement of the collected slime and solution of the gold are performed as in the ordinary intermittent decantation process, but the gold-bearing solution is removed by a series of two continuously working separators with conical bottoms and a device for slow mechanical agitation. The slime pulp with about 50% moisture is drawn upwards from the bottoms of these tanks by means of pipes lying on the cones.

(C.) 96/10. John Dewrance (1), Herbert Ernest Williams (2). Improvements in the manufacture of alkaline cyanides. 11.3.10.

This is a process for the production of alkaline cyanide, and consists of mixing with alkaline ferrocyanide a suitable proportion of a cuprous or cupric salt and sulphuric acid, the hydrocyanic acid liberated being absorbed in an alkali. The resulting alkaline cyanide is recovered therefrom by any known method.

(C.) 72/10. A. M. Robeson. An improved method of refrigeration. 25.2.10.

The object of this invention is to make use in a refrigerator, to the greatest possible extent, of the well known method of cooling by the evaporation of water in dry or partially dry air and making use of the effect of contra current.

The apparatus consists of a system of metallic ducts with dry and wetted surfaces through which the atmospheric dry air is allowed to pass in succession and is cooled in an increasing degree by being in a metallic thermal contract with the wetted (and evaporating surface) whilst passing through the dry duct and cooling all the more when passing next through the wetted duct where the cooling by evaporation takes place.

(C.) 117/10. Hans Charles Behr. Improvements in stamp mill cams. 18.3.10.

This application is for a method of constructing the curved operating surface of cams for raising gravity stamps by substituting for the pure involute curve in ordinary use generated from a circle, a somewhat similar curve struck from points situated at varying distances from the centre of the cam shaft.

The object of the new curve is to vary the point of contact with the tappet within certain limits on each side of the stem centre instead of being on the centre line during the whole contact as in the present practice, and the inventor claims to obtain a more gradual start in lifting the tappet and a higher throw on the final contact.

(C.) 141/10. Francois Dallenagne (1), Henri Dallenagne (2). Improvements in ore concentrating and separating apparatus. 1.4.10.

This application relates to improvements in ore concentration apparatus, and consists of a modified form of device intended to actuate a grooved table, forming the subject matter of a prior patent No. 238/08 by the same applicant.

(C.) 89/10. Alfred Walton Stockett (1), John Hawthorne Wilson (2), John Whitton Marshall (3). Improvements relating to doors for mine cages. 8.3.10.

This application is in respect of a device for locking the doors of man skips or cages having two or more decks, in such a manner that one door being locked by the skipman locks the other door or doors, which cannot then be opened from the inside.

(C.) 242/09. Alexander John Arbuckle. Improvements in apparatus or plant for treating crushed ore products for the recovery of the metal contents thereof. 21.5.09.

The invention relates to apparatus or plant designed more particularly for the treatment of pulverised or crushed auriferous ores, for the recovery of the precious metal therein contained.

According to the description the apparatus is applicable to pulp, either as it is received from the battery, or for the separate products composing said pulp, *i.e.*, sand or slime.

The main units of the apparatus are a settling or dewatering tank or vessel, and an arrangement of screw conveyors to withdraw the settled pulp.

The specification deals in great length and detail with the description of the component parts of the apparatus.

(C.) 387/09. Frank Leonard Whitmore. Improvements in skip indicators for winding or hauling plants. 24.8.09.

This application relates to a device for giving variable motion to the pointer of dial depth indicators in connection with winding apparatus.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 152/10. James Thompson Fennie (1), Frederick George Fell (2). Improved tappet. 5.4.10.

(P.) 153/10. James Thompson Fennie (1), Frederick George Fell (2). Improved tappet fixing device. 5.4.10.

(P.) 154/10. Alexander Fullarton Gill. An appliance for the automatic closing of dump doors in head gear. 5.4.10.

(P.) 155/10. Henry James Shedlock Heather. Improvements in the control of induction motors of the slip-ring or equivalent type. 5.4.10.

(P.) 156/10. Hugh McMenamin. Improvements in driving means for ore feeders. 6.4.10.

(P.) 157/10. Bertram Gray (1), Robert George Campbell Pitt. Improvements in plates for the construction of chutes. 6.4.10.

(C.) 160/10 Otto Wilhelmi. Improvement in method of making beton or concrete sleepers with resilient rail supports. 8.4.10.

(C.) 161/10. Donald Barns Morison. Improvements in and connected with apparatus for condensing steam. 8.4.10.

(C.) 162/10. Harry Edgar Lever. Improvements in internal combustion engines. 12.4.10.

(P.) 163/10. Charles McGregor. Improvements in and relating to means for igniting the fuses of the like of explosive charges in blasting. 13.4.10.

(P.) 165/10. Andrew Frederick Crosse. Improvements in apparatus for the fine grinding of ores and other materials. 13.4.10.

(P.) 166/10. Kenneth Lancaster Graham (1), Edward Arthur Turner (2). Improvements in ore feeders. 13.4.10.

(P.) 167/10. Anastasios Demstruis Marusis (1), Frederick Mitchell (2). Improvements in apparatus for raising water or other liquids. 14.4.10.

(C.) 168/10. Peter Arthur Walsh. Improvements in tube mill linings. 14.4.10.

(P.) 169/10. Frederick Johnson Knight. Improvements in the preparation of sodium cyanide. 15.4.10.

(P.) 171/10. Edward Grote. Improvements in and relating to poles for carrying electric wires and the like. 16.4.10.

(P.) 172/10. Samuel Davis. Improvements in acetylene gas lamps. 16.4.10.

(P.) 173/10. Henry James Shedlock Heather. A method of increasing the safety of the operations of hoisting and lowering with induction motors. 19.4.10.

(P.) 174/10. Arthur Wilson Ferus. Improvements in acetylene gas generators and lamps. 20.4.10.

(P.) 175/10. Arthur Wilson Ferus. Improvements in acetylene gas lamps and generators. 20.4.10.

(P.) 177/10. Ernest Henerich Thiel. Improvements in rock drilling medium chucks. 21.4.10.

(P.) 178/10. William Price, Improvements appertaining to mechanical haulage for automatically tipping and righting the trucks or vehicles. 22.4.10.

(C.) 179/10. John Graham Stewart. Improvements in the jointing of pipes and the like. 22.4.10.

(C.) 180/10. Stewarts & Lloyds, Ltd. (1), Robert Ballantyne (2). Improvements in and in the fitting of branch service connections to main pipes. 22.4.10.

(C.) 181/10. The Beven Lock Nut Co. (1), John Anstee Wilks (2). Improvements in or connected with nut locks. 22.4.10.

(C.) 182/10. C. G. Patterson (1), The Butters Patent Vacuum Filter Co. (2). Improvements in filters for metallurgical and other uses. 22.4.10.

(P.) 183/10. Alfred Charles Churchman (1), Frederick Herbert (2). A new and improved tile, sheet, slab or the like of cement or concrete reinforced with metal or otherwise to be used for the construction of roofs, ceilings, walls and the like. 22.4.10.

(C.) 184/10. Maurice Delvigne. Improvements in explosives. 23.4 10.

(P.) 185/10. Charles Frederick Eduard Oscar Schmitt. Improvements in or appertaining to the driving means for the stamps of stamp batteries. 23.4.10.

(C.) 186/10. Paul Girod. Improvements in the process of refining steel. 26.4.10.

(C.) 187/10. Paul Girod. Method of supplying electric furnaces with polyphase current. 26.4.10.

(P.) 188/10. John Wakem Hodge. Improvements in spraying devices for use in mining. 27.4.10.

(P.) 189/10. Jacob Boadle Yondale Thomson. Improvements in means for automatically locking the drums of winding and like engines when unclutched. 27.4.10.

(P.) 190/10. Edward Grote. Improvements in direct producing continuous current dynamos, and motors. 27.4.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Adams, E. L., *l/o* Johannesburg; New Heriot G. M. Co., Ltd., P. O. Box 10, Cleveland.
Adams, H. A., *l/o* Village Main Reef G. M. Co., Ltd.; Robinson G. M. Co., Ltd., Johannesburg.
Atkinson, C. E., *l/o* Bradford, Yorkshire; Sheba Bongola Mine, Selukwe, Rhodesia.
Bagshaw, J. F., *l/o* Magnesite Mines, Ltd.; Transvaal Bischoff, Ltd., Warmbaths.
Bisset, A. C., *l/o* East Rand; Kilnshade, Macduff, Banffshire, Scotland.
Butters, C., *l/o* San Francisco; College Avenue and 29th Street, Oakland, California, U.S.A.
Coles, W. Alison, *l/o* Selukwe; 23, Carnarvon Road, Bertrams.
Coombe, M. H., *l/o* Langlaagte; Randfontein Deep, P. O. Box 45, Randfontein.
Eaton, J. M., *l/o* Etna Mine, Hartley; Gaika G. M. Co., Queque, Rhodesia.
Gosney, P., *l/o* Essexvale, Rhodesia; Hill House, Southall, London.
Graham, H. W., *l/o* Johannesburg; c/o Bank of Africa, 113, Cannon Street, London.
Gray, J. W., *l/o* Waihi, New Zealand; 7, Ashbrook Terrace, Leeson Park, Dublin, Ireland.
Hesom, W. W., *l/o* Gordon Road, Johannesburg; New West Bonanza G. M. Co., Klerksdorp.
Hill, J. W., *l/o* Salisbury; 26, Newark Drive, Pollokshields, Gasgow, Scotland.
Holloway, G. T., *l/o* Chancery Lane; 9-13, Emmett Street, Limehouse, London E.
Hutton, C. E., *l/o* Orange Grove, Johannesburg; Gwelo Club, Gwelo, Rhodesia.
Ladds, L. H., *l/o* Queque; Colleen Bawn Mine, Gwanda, Rhodesia.
Leslie, C. D., *to* Consolidated Gold Fields of S.A., Ltd., P. O. Box 1167, Johannesburg.
Macfarlane, T. J. M., *l/o* Johannesburg; P. O. Box 258, Salisbury, Rhodesia.
Mackintosh, H. G., *l/o* Dunraven Hotel; Surprise G. M. Co., Selukwe, Rhodesia.
Rascher, L. H., *l/o* Knights; P. O. Box 55, Winburg, O.R.C.
Staver, W. H., *l/o* Illinois; Telluride, Colorado, U.S.A.
Thomas, G. G., *l/o* Sabie; East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand.
Weston, R. O. *l/o* Queque; Gwelo, Rhodesia.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. X. JUNE, 1910. No. 12.

Proceedings
AT
Annual General Meeting, June 18, 1910.

The Annual General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, June 18th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

76 Members: Dr. J. Moir, Messrs. C. B. Saner, E. H. Croghan, E. J. Laschinger, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, A. F. Crosse, W. A. Caldecott, E. H. Johnson, R. G. Bevington, H. A. Adams, R. Allen, W. T. Anderson, A. J. R. Atkin, S. G. Bartlett, S. Beaton, W. Beaver, H. D. Bell, B. V. Blundun, A. J. Bowness, G. L. Burnett, J. M. Campbell, A. L. Chambers, T. Donaldson, J. F. Ferguson, G. Goodwin, J. Gray, F. G. Guthrie, R. E. Hall, E. Hollis, W. H. Jane, J. H. Johnson, T. Johnson, T. S. Jolly, T. W. Jordan, G. A. Lawson, H. Lea, J. Lea, C. D. Leslie, W. D. Lloyd, Dr. J. McCrafe, H. Meyer, P. Morrisby, S. Morison, S. Newton, W. Nicklin, F. D. Phillips, E. T. Rand, L. J. Robinson, O. D. Ross, A. H Scarf, G. H. Smith, T. F. Smith, B. C. T. Solly, K. K. H. Sartorius, A. R. Stacpoole, H. Taylor, J. A. Taylor, A. Thomas, J. Thorburn, J. R. Thurlow, K. Tonnesen, O. Tonnesen C. Toombs, M. Torrente, W. G. Urquhart, J. F. Walker, F. W. Watson, F. Wells, A. Wilkinson, and L. J. Wilmoth.

19 Associates and Students: Messrs. A. R. Adams, J. G. V. Clarence, C. E. Deakin, C. L. Dewar, H. S. H. Donald, W. J. N. Dunnachie, D. G. Esplin, W. J. R. Hunter, R. L. Impey, A. King, W. Ninow, H. B. Powter, H. Rusden, H. Stadler, J. B. Sutton, D. J. Thomas, J. Thorlund, W. Webb and L. A. Womble.

21 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the May *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Jas. Gray and H. A. White were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Abel, William James, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Fitter.

Aspland, C. Hatton, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider. (*Transfer from Associate Roll.*)

Aubourg Louis, Simmer Deep. Ltd., P. O. Box 178, Germiston. Amalgamator.

Brown, John Henry, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Battery Foreman.

Byth, Alexander Robertson, Simmer Deep, Ltd, P. O. Box 178, Germiston. Mill Assistant.

Gibson, James Monro, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

Inglis, William George, Simmer Deep, Ltd , P. O. Box 178, Germiston. Mill Shift Boss.

Jane, Walter Hooper, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Foreman. (*Transfer from Associate Roll*).

Macgregor, H. S., Simmer Deep, Ltd., P. O. Box 178, Germiston. Assistant Manager.

Mackenzie, Kenneth Russell, East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Cyanider.

McCoy, Frederick Austin, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

Murphy, Sewell John, Simmer Deep Ltd., P. O. Box 178, Germiston. Mill Shift Boss.

Ritchie, Henry Douglas, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

Roberts, Frank Gowyne Alfred, Knight's Central, Ltd., P. O. Box 91, Germiston. Mine Manager. (*Transfer from Associate Roll*).

Simpson, John, Witwatersrand G.M. Co., Ltd., P. O. Box 1, Knights. Colliery Manager.

Smith, Thomas Francis, Crown Mines, Ltd., P. O. Box 158, Fordsburg. Cyanider. (*Transfer from Associate Roll*).

Tatham, Basil St. John, F.S.S., Randfontein Deep, Ltd., P. O. Box 45, Randfontein.

Thorpe, William Edward, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Assayer and Chemist. (*Transfer from Associate Roll.*

TRELOAR, AMOS, Rooiberg Minerals Development Co., Ltd., P.O. Rooiberg *via* Warmbaths. Reduction Officer.

URQUHART, WILLIAM GRAHAM, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator. (*Transfer from Associate Roll.*)

WAKEFIELD, WALTER PERCY, Simmer Deep, Ltd., P. O. Box 178. Mill Shift Boss.

WILSON, J. GORDON, M.A., B.Sc., 87 Wolmarans Street, Johannesburg. Analytical Chemist.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

BERNSTEIN, MILTON HENRY, P. O. Box 258, Benoni. Assistant Amalgamator.

CHARTERS, ALFRED GRAHAM, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

DASHWOOD, GEORGE LEWIN, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tube Miller.

DUDGEON ARTHUR FREDERICK, P. O. Box 17, Penhalonga, Rhodesia. Mine Owner.

FRANKLIN, FRANCIS ROMAN, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

GILBERTSON, ALEXANDER, Rooiberg Minerals Development Co., Ltd., P. O. Rooiberg *via* Warmbaths. Mine Captain.

HEATLY, WALTER BRUCE, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Assistant.

HOLMAN, LERNARD WHELPTON, Consolidated Goldfields of S. A. Ltd., Laboratory, P. O. Box 108, Germiston. Assayer.

HUMAN, WILLIAM, Geldenhuis Deep, Ltd. (North), P. O. Box 5, Cleveland. Mill Shift Leader.

MARE, FREDERICK KARSTEN, Geldenhuis Deep, Ltd. (North), P. O. Box 14, Cleveland. Mill Shift Leader.

MACKAY, HUGH SINCLAIR, Simmer Deep, Ltd., P. O. Box 178, Germiston. Tube Miller.

METHVEN, ALEXANDER MARSHALL, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Learner.

PATTLE, HAROLD ALFRED, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

WESTFIELD, FREDERICK ARTHUR, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

As Students.—

ARNOT, NIGEL STANLEY, Consolidated Gold Fields of S. A., Ltd., Laboratory, P. O. Box 108, Germiston.

SNEEZUM, EDWARD ALFRED, Geldenhuis Deep, Ltd. (North), P. O. Box 5, Cleveland. Learner.

The President: I have much pleasure in stating that this brings our total membership to over 1,200.

ANNUAL REPORT.

The Secretary submitted the Annual Report of the Council and the Financial Statement for the past year as follows:—

Your Council has great pleasure in placing before you its report upon the work of the past year, together with the usual Statement of Accounts. With reference to the latter we regret that it has not been found possible, owing to pressure of work on account of the postponement of the Annual Dinner and illness of the Secretary to place before you at this meeting an audited statement. This we hope to accomplish before publication in the *Journal*, and it is not anticipated there will be any material difference in the statement from that submitted to you this evening.

The Revenue shows an increase over that of last year of £222 6s. 9d., each item of revenue contributing its quota to this result.

On the other hand, the Expenditure has increased in every respect excepting rent.

The cost of the *Journal* has also increased, owing to the increase in the number of pages as compared with the previous year. And secondly, the total cost of the extra copies kept in stock has been written off, in place of the former custom of debiting the cost to the volumes of Transactions.

Your Council has also decided to write down considerably the cost of former volumes now in stock, owing to the fact that so far no depreciation in value has been allowed in previous years on this account, and further this item shows a considerable increase annually without possibility of its redemption except under extraordinary circumstances. To meet this a sum of approximately £235 has been set apart. After full allowance for the above, there remains a surplus of revenue beyond the necessary expenditure of about £44.

Turning to the Balance Sheet, it will be noticed that the Cash Assets here and in London exceed our immediate liabilities by about £226, in addition to which there is over £450 due to us by sundry debtors, chiefly for *Journal* advertisements, which will be collected during the course of the next month.

In this connection it may be stated that in accordance with former resolutions of your Council those members, whose subscriptions are in arrear for two years, will be the subject of special consideration by the Finance Committee and should it be found that these members value the benefits received so lightly as to bestow such attentions on the requests issued by the Secretary for payment, as is usually evidenced by consignment to the waste paper basket, they will compel your Council to adopt that judicial attitude towards them, which is such a matter for regret to both parties.

During the year 118 Members, 86 Associates and 43 Students were elected and admitted, 19 of the new members having been associates who have transferred and 3 were transferred from the student roll to that of the associates. The resignation of 12 members and 7 associates have been accepted with regret, while the loss by death, has been as follows:—

Prof. H. Bauerman (Hon. Member), Messrs. W. Bradford, J. Davies, W. H. Smith, Dr. J. T. Carrick, and H. S. Stark (Members).

In expressing our sorrow to the relatives of the deceased, whose loss as members of the Society we regret, we desire to refer especially to the pathetic sadness surrounding the death of our late member, Dr. Carrick, who was a passenger by the *S.S. Waratah.*

The tables of membership, as usually submitted are given herewith, and show that a substantial increase to our already large membership has taken place during the year.

During the past year twelve Ordinary General Meetings were held, at which the high excellence of the Society's past record was maintained both in the number and in the quality of the papers submitted. No Special General Meetings were held.

The following is a list of the papers submitted during the year:—

July, 1909. The President's (Mr. A. McArthur Johnston) Inaugural Address on "The Crystallization of Iron and Steel."

"Air Lift Pumps for Underground Work" (Note), by Mr. B. C. Travers Solly.

August, 1909. "Making of Silver Amalgam" (Note), by Mr. E. H. Croghan.

"The Continuous Collection of Sand for Cyaniding," by Mr. W. A. Caldecott.

"Notes on Corrosion, with special reference to the Corrosion of Steel Winding Ropes," by Mr. M. Thornton Murray.

September, 1909.—"A Sketch of the small Cyanide Plant as erected and worked in Rhodesia," by Mr. F. J. Thomas.

"A Method for the Recovery of Zinc from solutions of Sulphate," by Messrs. W. Cullen and G. F. Ayers.

TABLE I.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.	June, 1909.	June, 1910.
Hon members	5	5	5	8	14	9	13	18	18	16	17	18	21	22
Life members		1	1	1	1	1	1	2	3	4	7	7	8	9
Members	49	81	101	197	281	291	441	639	680	680	683	618	630	731
Associates	—	—	—	180	238	243	281	338	321	314	342	357	342	402
Students	—	—			—	—		4	8	4	7	18	20	60
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224

TABLE II.

DISTRIBUTION.	MEMBERS.			ASSOCIATES.	STUDENTS.	TOTAL.
	Honorary	Life.	Ordinary.			
Transvaal (Witwatersrand)	3	1	480	175	47	706
Do. (outside districts)	1	—	55	39	5	100
Rhodesia			67	46		113
Cape Colony, O.R.C., and Natal	1		6	3	—	10
British East & West, German S.W. & Portuguese E. Africa	—	—	4	4	—	8
British Isles	12	5	38	21	3	79
Australasia ...	—	1	8	23	—	32
Dominion of Canada .	—		1	2	—	3
India and Burma	—	—	3	5	—	8
Europe		—	4	6	—	10
United States of America	5	1	22	24	1	53
Central and South America			2	2		4
Mexico .		—	7	15	—	22
Borneo ...	—	—	—	1	—	1
Dutch East Indies			17	5	—	22
China, Japan and Korea			1	3	—	4
Federated Malay States		1	1	1	—	3
Addresses Wanted		—	15	27	4	46
TOTAL	22	9	731	402	60	1,224

September, 1909. — “Analysis of Gases from burning Nitro-Glycerine Explosives,” by Messrs. W. Cullen and D. W. Greig.
“Specific Gravity of Bisulphate Solutions,” by Mr. T. Donaldson.
“The Development of Heavy Gravitation Stamps” (reprint from *I.M.M. Journal*), by Mr. W. A. Caldecott.
October, 1909.—“The Barberton Goldfields,” by Mr. Alex. Richardson.
“The Assay of Cyanide Solutions and Slime Residues carrying dissolved Gold,” by Mr. A. Whitby.
“Experiments on the Assay of Acid Washes resulting from the Cyanide ‘clean up’ by the use of Bi-sulphate,” by Mr. L. J. Wilmoth.
November, 1909.—Some Experiments on Smelting Titaniferous Iron Ore” (Part I.), by Prof. G. H. Stanley.
“Treatment of Ore Slime,” by Mr. A. F. Crosse, with addendum on “The Treatment of Black Sands,” by Mr. F. Alexander.
December, 1909. – “Endless Rope Haulage,” by Mr. H. G. Kay.
“A Rotary Extractor for Precious Metals from Solution,’ by Messrs. D’Arcy Lloyd and Ernest T. Rand.
January, 1910.—“The Precipitating Effects of Substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing Gold and Silver,” by Messrs. A. J. Clark and W. J. Sharwood.
February, 1910.—“Further Notes on Rand Mining,” by Mr. Tom Johnson.
“The Tube-Mill Circuit and Classification,” by Mr. G. O. Smart.
March, 1910.—“The Assay of Tin Ores,” by Mr. Jas. Gray.
“Mining at the Redjang-Lebong Gold and Silver Mine, Sumatra,” by Mr. H. Philp
April, 1910.—“Test for Prussic Acid” (Note), by Dr. J. Moir.
“The Treatment of Accumulated Slime, and the use of Filter Presses for Clarifying Slime Solution and By-Products,” by Mr. J. D. O’Hara.
“Some Experiments on Smelting Titaniferous Iron Ore” (Part II), by Prof. G. H Stanley.
May, 1910. Grading Analyses and their Application,” by Mr. H. Stadler.
“Experiments on the Precipitation of Gold from Cyanide Solutions by Carbon in Lime,” by Mr. E. H. Croghan.
June, 1910. “Sand Filling on the Witwatersrand,” by Mr. Edgar Pam.
“The Destruction of Cyanide,” by Dr. James Moir and Mr. Jas. Gray

There were thus submitted 25 original papers during the year, the reading of which and the discussions thereon tended to make the meetings both instructive and interesting. The result is most noticeable in the large attendance at all the meetings. The greatest number present was 124 at the November meeting, and the average for the year was 97, an advance of 29 per meeting over the previous year. We can claim that the Society was never in a more healthy state, financially, numerically and in the interest evinced by the members.

The Annual Dinner maintained its reputation as one of the important functions of the year both from a social and political point of view. The speeches of Mr. F. D. P. Chaplin, who proposed the toast of South Africa, and of Mr. P. Duncan, who replied, were listened to with great interest, since they expressed the hope of greater activity in the colonisation of South Africa.

During the year thirteen Council Meetings were held. The average attendance at these meetings was 14, which affords excellent testimony that the Society’s affairs was well looked after during the year. The attendances of Members of Council for the period were as follows:—

Mr. A. McA. Johnston*	9	Mr. E. H. Croghan -	11
,, W. R. Dowling -	10	,, M. Cullen - -	6
Dr. J. Moir - -	7	,, K. L. Graham -	7
Mr. C. B. Saner -	11	,, E. J. Laschinger	7
,, A. F. Crosse -	2	,, A. Richardson -	7
,, S. H. Pearce -	2	,, G. O. Smart -	10
Dr. W. A. Caldecott	9	Prof. G. H. Stanley -	12
Mr. W. Cullen -	2	Mr. J. E. Thomas† -	7
,, R. G. Bevington	10	,, A. Whitby -	6
,, J. Littlejohn -	9	,, H. A. White -	13
,, F. F. Alexander	10	Prof. J. A. Wilkinson	13

*Granted 3 months’ leave. †Granted 4 months’ leave.

In addition to these meetings many of the Members of Council were called upon to serve on the Editorial, the Publications, and the Finance Committees, whilst various sub-committees have been appointed during the year to investigate, and report to the Council on, matters pertaining to your interests. These committee meetings have taken a considerable amount of time and the Council is indebted to those members who have served thereon and who always ungrudgingly devoted themselves to assist the welfare of the Society.

One of the most noteworthy events in the history of the Society during the past year has been the introduction of the system of admitting apprentices, or learners, into the Society, on the same status as students. As you will have noted, this has been largely taken advantage of, though it is felt that the system may be still further extended amongst our younger workers with

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 15th June, 1910.

Dr.

1908-9. £ s. d.	EXPENDITURE.	1909-10. £ s. d.	£ s. d.
	To Charges (including Postages, Attendance, Bank Charges, etc.) ...		
(187 18 0	,, ...	212 1 8	
157 2 0	,, Printing and Stationery ...	223 2 1	
67 18 0	,, Rent and Lighting	30 6 8	
386 0 0	,, Salaries...	447 5 0	
7 8 0	,, Depreciation on Furniture ...	12 19 6	
– – –	,, Transactions	235 15 4	
20 0 0)	,, *Provision for Doubtful Debts*	—	
			1,161 10 3
826 6 0			
	,, Society's Journal—		
	Cost of Reporting, Printing		
(995 12 5	and Sundries	1,073 16 10	
409 8 3)	*less* Received for Subscriptions		
	and Advertisements ..	384 19 5	
586 4 2			688 17 5
260 2 10	,, Balance, surplus on year's working		44 12 1
£1,672 13 0			£1,894 19 9

Cr.

1908-9. £ s. d.	REVENUE.	1909-10. £ s. d.	£ s. d.
	By Subscriptions—		
(1,135 11 6	Members	1,257 18 0	
	Life Members	21 0 0	
455 19 3	Associates	489 0 9	
8 13 3)	Students	21 0 0	
			1,788 18 9
1,600 4 0			
72 9 0	,, Entrance Fees		106 1 0
£1,672 13 0			£1,894 19 9

BALANCE SHEET, as at 15th June, 1910.

Dr.

1908-9. £ s. d.		1909-10. £ s. d.	£ s. d.
(136 11 6	To Sundry Creditors	136 6 3	
27 16 6	,, Subscriptions paid in advance ...	45 3 0	
6 0 0)	,, Prize Fund	6 0 0	
	,, Subscriptions for Decennial Index	14 0 0	
			201 9 3
170 8 0			
	,, Revenue Account—		
(1,231 7 3	Balance at June 1, 1909 ...	1,491 10 1	
	Add surplus as per Revenue		
260 2 10)	Account, June 15, 1910 ...	14 12 1	
			1,536 2 2
1,491 10 1			
£1,661 18 1			£1,737 11 5

Cr.

1908-9. £ s. d.		1909-10. £ s. d.	£ s. d.
(174 15 2	By Cash at Bank—		
	Johannesburg	187 2 11	
	London	239 11 10	
91 0 3)	,, Cash in Hand	0 19 8	
			427 14 5
265 15 5			
86 0 0	,, Office Furniture		116 0 0
939 15 5	,, Transactions, Vols. I. to X.—		
	Stock in hand		707 5 0
(372 10 0	,, Sundry Debtors...	488 14 9	
2 2 9)	*less* Provision for Doubtful Debts ...	2 2 9	
370 7 3			486 12 0
£1,661 18 1			£1,737 11 5

We have examined the foregoing Statements of Account, compared them with the Books and Vouchers of the Society, and we find the statements to be true and correct, as shown thereby.

Sgd. ALEX. AIKEN and CARTER, } *Hon. Auditors*

Johannesburg, 7th July, 1910.

(Signed) J. LITTLEJOHN, *Hon. Treasurer.*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

benefit to themselves and to the industry at large. It is to these apprentices that the Society must look for its future extension and management, and it is hoped that the ensuing twelve months will note a large increase in their numbers.

A pamphlet is being drafted on the lines laid down by Drs. Macaulay and Irvine in their paper on "Safety Measures in Mining," wherein it is hoped to state in brief and convenient form many of the measures therein mentioned The publication of this is, however, held back until the report of the Mining Regulations Commission has been published, since it is anticipated that useful and authoritative data may be extracted therefrom, which will considerably enhance the value of the pamphlet. In the meantime, rules to be observed in the handling of high explosives are being drawn up at our instigation and with the assistance of the Chamber of Mines. It is hoped these will shortly be finally approved, and in due course issued to all mines wherein such explosives are used.

It will have been noted that during the year a system of abstracting particulars of some of the patents applied for in the Transvaal, which would more particularly interest members, has been adopted. This has evidently met with approval, since it assists in keeping members advised in greater detail than is possible in the ordinary list of patent applications.

During the year advantage has been taken of the arrangement made some years ago with the Institution of Mining and Metallurgy, and a paper read at one of that Institution's meetings was reprinted in our *Journal* and the joint discussion published. This met with considerable success and also enhanced the value of our proceedings. Further a paper has now been conjointly read and discussion will proceed in this as in the former case.

You will have seen that the Institution of Mining and Metallurgy have issued its sub-committee's report on the Standardization of Bullion and Assay Values. As individuals your members have given their opinions concerning these standards, but it was felt that no action should be taken by the Council as a body. As soon, however, as the final recommendations of the Institution of Mining and Metallurgy are published, it may be necessary to bring the matter before you to determine whether it would not tend to improve our systems here by following the lines adopted by that body.

We desire to mention that considerable interest has been added by the visits to the Rand of a past president of the Institution of Mining and Metallurgy, Mr. W. McDermott, and a Member of Council, Mr. F. W. Harbord.

There has been issued to members and to mining, metallurgical and chemical men throughout the world, the draft report of our sub-committee on the standardization of metallurgical and mining terms. Since uniformity in the usage of these and similar terms is much to be desired, your Council would commend this draft to the notice of all members in the hope that they will express opinions on the words contained therein and that they will assist the sub-committee in still further enlarging the scope of the enquiry by submitting other words and phrases which might well be included in the list.

The Committee on the Standardization of Materials, of which mention was made in the last annual report and to which your President and Prof. Wilkinson were appointed to act on behalf of your Society, has not met during the year, so that no record of progress is to be reported. It is felt that some action should be taken to place many of the commercial products on a basis whereby certain standards may be accepted as a guarantee of quality, and also that uniformity be adopted in making the necessary tests. If the aforementioned committee do not recognise the value of proceeding with this at once, it might be advantageous to this Society to consider the recommendation of standards both in quality of materials and in methods adopted in their testing or analyses.

A sub-committee of your Society had the honour of placing the views of your Council on Electric Power Schemes before the Power Companies Commission.

The question of Closer Union between the various technical societies on the Rand has not been further discussed by your Council.

The number of our Corresponding Members of Council has been doubled during the past twelve months. The new members appointed consist of of a past Vice-President, Mr. T. Lane Carter, and Dr. W. J. Sharwood, both of whom assist Mr. G. M. Taylor in looking after our interests in the United States of America, whilst Mr. H. M. Leslie and Mr. E. G. Banks have been kind enough to represent us in India and New Zealand respectively It was felt that Rhodesia might well be brought more directly in touch with our Council here and with this end in view, Mr. Clement Dixon and Mr. C. B. Kingston (both past Members of Council) kindly consented to further our interests amongst the mining men of that country.

We would beg to record our sincere thanks to all our Corresponding Members of Council for the encouragement which they have from time to time given us by making suggestions, forwarding the names of new members or associates and in obtaining papers. In particular we would thank our representative in Great Britain, Mr. G. T. Holloway, for the kind interest he takes in our members during visits to London.

In view of the increase in the number of Student Members, who, as has already been stated, are recruited not only from the various colleges but also from mines and works, your Council decided to give as much practical encouragement as possible to this class of members in their technical studies, by instituting a scheme of prizes the details of which will be published for your information. The scheme, drawn up by your Council was duly submitted to the Council of the Transvaal University College and accepted by that body as formulated.

A list of the Members of the Society was issued in November last. The proposal to publish a decennial Index did not meet with sufficient encouragement to warrant proceeding with this, and so the matter has been allowed to stand over for the present.

The attendance at the Social Dinner prior to the monthly meetings has been encouraging during the past year, and members are reminded that these pleasant functions will still be continued.

Our Secretary, Mr. Fred. Rowland, has been indefatigable in his efforts towards furthering the Society's interest in all directions. He has extended his hours of attendance so that should members desire to find him in his office, he will be there between the hours of 10 and 4 on Mondays and Saturdays.

In conclusion, your Council would take this opportunity of again acknowledging the indebtedness of Members to the Transvaal University College for the accommodation of the Secretarial department and for the use of the Lecture Room for the monthly meeting; to Messrs. Van Hulsteyn, Feltham & Fry, the Hon. Solicitors, for legal advice and assistance; to the Hon. Auditors, Messrs. Alex. Aiken and Carter; and to your Hon. Treasurer, Mr. James Littlejohn, in whose capable hands has so long rested the financial control of the Society. Your Council would also express the acknowledgments of the Society to the local press for the space accorded and the publicity given to the Proceedings of the Society and notices of its activities.

The President: I beg to move that the Annual Report and Statement of Accounts be formally adopted.

Dr. J. Moir (*Vice-President*): I have much pleasure in seconding the motion. I do not think there is any special reference I have to make on the Report except to the great progress we have made this year, and the increasing wideness of view and scope of the Society and its papers. It seems probable that within the next year or two we shall have to sub-divide the Society to some extent, or re-arrange our proceedings as the result of this extraordinary catholicity of view.

The motion was agreed to.

The President: Now that the Council's Report has been adopted, it behoves me to say good-bye to you as President. Fortunately in this Society the precedent demands that there shall be no valedictory address; I hope you all appreciate that as much as I do. During the President's term of office there is always a considerable amount of work devolving upon him, and I sympathise with your future President in the duties which he will be called upon to undertake. When I took over the office nearly a year ago I felt very much non plussed with regard to how I was going to manage, and because I felt there were many members in this Society much more qualified to do the work than I was. I was, however, in a very fortunate position because I had three excellent Vice-Presidents and 12 good men and true as Members of Council. In addition, your financial affairs were in the hands of a very sound member, who has been already alluded to in this Report—Mr. Littlejohn. I am sure the Society is very much indebted to Mr. Littlejohn for the good work he has done in the past. Although one of our most retiring members and although we seldom hear him in discussion, we know the good work he does, and I am sure you will show your warm approval by according him a special vote of thanks. My year of office has been very pleasureable to me, and I can only say that I retire from it with a certain amount of regret. On the year's working we have made a profit, though that profit has not been large. This accords very much with my view. The individual is different from a Society. For the individual it is necessary as the years go on that he should consider the time may come when he will not be able to continue the fight as of old. On the other hand, a sound society ought to have new blood coming in year after year, and it is new blood we have to foster and encourage. There is no necessity therefore for hoarding up funds which will keep the Society going in future years, and I believe the best possible way of making any society a success is to make it independent of itself year by year, and let the members who are paying their fees at the present time get the benefit of those fees. This we are trying to do by giving them dividends in the form of material which is printed in our *Journal* With regard to the future policy of the Society, which Dr. Moir has hinted at to-night, there ought to be a little more encouragement, to begin with at any rate, given the mining side of our Society.

This might be achieved by having every third meeting a mining night, so that this might be devoted to papers dealing with mining, and any man interested in the mining side of our Society could come here and converse on the subject, knowing he will not have to wait late before the paper he wishes to discuss comes on. In this way we make a beginning towards what may turn out to be a greater scope for our Society in the encouragement of mining subjects, and we would hope to have still more meetings devoted to mining. In conclusion, I thank you very much for the encouragement you have given me during the past year, and for the sympathy extended me during that time. I will now call upon the Secretary to read the Scrutineers' report with regard to the Officers and Council for 1910-1911.

The Secretary read the Scrutineers' report, as follows :—

Johannesburg, 18th June, 1910.

The President,

Annual General Meeting of Members.

Dear Sir, — We have received from the Secretary this day 200 ballot papers, which we have opened and examined.

Hereunder we beg to hand you the result of our scrutiny.

President :

Dr. James Moir.

Vice-Presidents :

Mr. C. B. Saner.
Mr. W. R. Dowling.
Prof. G. H. Stanley.

Members of Council :

1 Mr. K. L. Graham.
2 ,, G. O. Smart.
3 Prof. J. A. Wilkinson
4 Mr. A. Whitby.
5 ,, F. F. Alexander.
6 ,, H. A. White.
7 Mr. A. Richardson.
8 ,, E. J. Laschinger
9 ,, E. H. Croghan.
10 ,, Jas. E. Thomas.
11 ,, Tom Johnson.
12 ,, Robert Allen.

There being no opposition the Hon. Treasurer, Mr. James Littlejohn, is re-elected unanimously

Yours faithfully,

Sgd. Jas. Lea,
,, A. Thomas,
,, S. Newton, } *Scrutineers.*

The President : After hearing the report of the Scrutineers I have to congratulate you very much on having elected Dr. Moir as President of our Society. You all know Dr. Moir very well, and I am sure that a man of his high attainments will make an excellent President, and that the Society will improve greatly under his care. The work which Dr. Moir has done has not been confined to what he has said to us at our meetings, but to the getting out of our *Journal* during the whole of the year. This is a work which can only be appreciated by those who have worked with him. I am sure he will considerably enhance the reputation of the Society during the forthcoming year.

Dr. J. Moir (*Vice-President*) : I thank you very much for the signal honour you have conferred on me in appointing me your President this year, and I thank Mr. Johnston too for the kind way he has spoken of me. I say it is a signal honour because it is really the highest honour which the Rand has to confer upon any scientific man—at all events this is the largest and most influential Society on the Rand, and I feel the responsibility. The fact that you have elected such an excellent selection of Vice-Presidents and such a hard working Council—in fact you have appointed practically all the hard workers—encourages me much in my very difficult task. Unfortunately I am not much of a speechmaker, and I do not look forward very much to that part of my duty, but with the help of these gentlemen I shall do my best. Someone has written

"A little limelight now and then
Is relished by the wisest men"

or something very flattering like that. That expresses my position very well, and although I am not naturally fond of the limelight I feel it is my duty to accept the position, and I feel I shall manage with the assistance of the Vice-Presidents and Council.

Mr. Chas. B. Saner (*Vice-President*): I feel I should like to congratulate the Society upon its wise choice in electing Dr. Moir as President, and I think that the members of the Society have cause for self congratulation in recording their votes for so many of the old Council of last year. By doing so they have shown their appreciation of the hard work those Councillors have done during the past year. That work has not shown itself very much in public, but any of those who enter the Council will realise that it is hard work. Also, the right infusion of new blood is a good thing for the Society. It is always necessary that a vigorous and lively society such as ours should have new blood brought in, and that we should confer that honour every year upon one or two active members of the Society who may previously have scoffed at the honour, but who will speedily realise that it is not all beer and skittles but more skittles than beer. One thing, as a mining man, I would like to urge upon the mining members of the Society to-night, and that is not to hide their light under a bushel as they have done in the past but to come forward and read short and concise papers. We know they have not time to write long and wearisome papers, but they can bring forward short and concise notes, not necessarily giving to the members only successful results of what they have done but what they would like to have done though they failed. I am quite certain that on the Rand, with its 60 miles of reef, there are many problems of great importance daily faced and solved by those who work underground. If they will only give us the benefit of their experience we, who are less fortunate, may be able to benefit by it. I am quite certain that if the mining men will only come forward and give us papers not only the Society but the mining industry will benefit by the interchange of ideas. That is what we are here for. By the interchange of ideas we benefit ourselves and benefit the various mines. I most heartily thank the members of the Society for having elected me as a Vice-President for the coming year.

Prof. G. H. Stanley (*Member of Council*): I beg to propose a vote of thanks to the gentlemen who acted as scrutineers. It is a tedious sort of thing to have to go through some hundreds of ballot papers, and I therefore think they deserve the hearty vote of thanks of the Society. I have also to return thanks for my own election as Vice-President of the Society, which I value very highly.

Mr. J. Gray (*Member*): I should like to second the proposal. For five years I acted as scrutineer. This year I sought higher honours but have been "unplaced."

Mr. S. Newton (*Member*): On behalf of the scrutineers I thank you very much.

SAND-FILLING ON THE WITWATERSRAND.

By Edgar Pam, A.R.S.M. (Member).

The process of filling up worked out areas on mines with sand has been in use for some years in various parts of Europe; water is used in all cases as a carrying and depositing medium

Sand filling has been carried on for over a year at the Robinson Gold Mining Company's mine, but only on a small scale, and before commencing to lower large tonnages, it was decided by Messrs. H. Eckstein & Co. to secure the services of an engineer who had previous experience of this work. Mr. Waller of Kaltowitz, Silesia, accepted the appointment, and the following is an account of the system recommended by him, modified, of course, to suit local conditions:

For convenience I will follow the sand from its temporary abode on the dump to its final resting place underground. The method of transport from the dump to the shaft is purely a mechanical question and I will touch only on the main points.

Where there is a fall of 10% or more from the foot of the dump to the shaft, or where the ground can be excavated to get this fall, hydraulic sluicing is undoubtedly the best method; more usually, however, this inclination cannot be obtained without great expense, and either conveyor belts or mechanical haulages would have to be used. With either of these two methods of transport, the sand is dumped into a brick-lined bin excavated near the shaft and connected to it by a tunnel. The bins which are at present in use are 40 ft. long, 30 ft. deep and 10 ft. wide, holding from 500-600 tons.

Parallel to the long axis of the bin and divided from it by a brick wall is a tunnel 6 ft. wide and 8 ft. high. The bottom of the bin is built with a 25% fall towards the tunnel, and through the dividing wall are four openings, 10 in. high and 6 in. broad, into which the water is sprayed.

A mixture of sand and water comes out through the openings under the nozzles and falls into a launder which has a fall of 12½% to 15%.

The sluicing water which is supplied under a pressure of 50 lb. per square inch is forced through four ½ in. nozzles and washes 200 tons of sand per hour down the mine.

The mixture running into the launder is by volume equal parts of sand and water, or by weight 60 parts of sand and 40 of water.

In addition to the four nozzles above mentioned, there is one more which leads clean water direct into the launder. This is used for washing out the pipes before and after use, and is an essential precaution against choking in the pipes.

The mixture of sand and water has now reached the shaft, and as a rule it here falls through a hopper into the pipe. Ordinary 5 in. and 6 in. pipes without lining have been used so far, but owing to the combined action of sand and water, are not expected to last long. Some special 7 in. pipes, and 5/16 in. walls, which will be lined with Jarrah wood 1 in. thick have therefore been procured from Europe.

The lining will be inserted in short lengths of about 8 in., so that even should a section of the lining get loose it will not stick in the bends and cause a choke, but go right down the pipe and out with the sand.

The pipes have loose flanges and are of dead lengths so that they can be turned and replaced more easily. They will be turned at frequent intervals in order to equalise wear.

In incline shafts and stopes the pipes may be replaced by launders, which are cheaper and less liable to be choked.

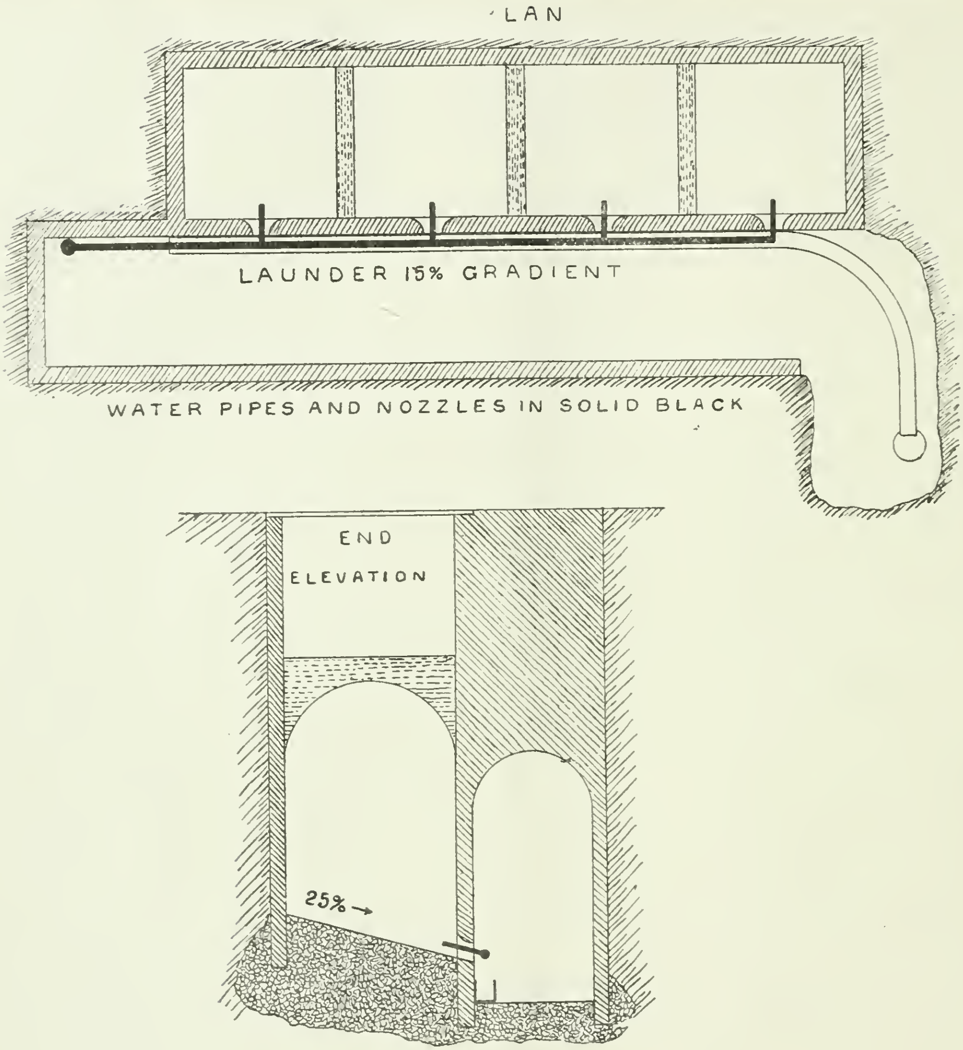


FIG. 1.—SAND BIN.

When debating the route of the pipe line there was at first considerable discussion as to whether horizontal lengths could be used, but the only objection found against them is that a greater quantity of water must be used for washing out the line. The length of a horizontal pipe line must of course bear some ratio to the pressure in the column. Mr. Waller says that the custom in Selisia is to allow 300 ft. of horizontal pipes for every 100 ft. of vertical head.

For the installation so far planned it has not been necessary to approach this limit, but 600 ft. of horizontal pipes are working well at the 2,000 ft. level.

Of upgrades we have so far no experience, they are undoubtedly possible, but it is questionable if they will prove expedient.

The only valves on the pipe line are for deviating the stream from one branch to another and one valve is always opened before the other is closed. The flow of sand can only be started or stopped from the surface, and it is therefore necessary to instal a reliable system of telephones so that instructions can be given from below.

Having traced the course of the sand as far as the top of an old stope, it will now be necessary to describe what work has been done to prepare the stope for sand filling.

All boxholes must be closed up above the pillars by means of uprights well hitched and $1\frac{1}{2}$ in. deals.

The deals are of course nailed on from inside the stope and must be fitted closely on to the pillars, footwall and hanging.

Any crevices are filled with grass which is rammed in tight and kept in place by a piece of wood.

Where possible, the sand rests against a shaft or boundary pillar on one side, but the other is usually built up similarly to the boxholes, the uprights being 3 to 4 ft. apart.

The chief point to keep in mind in filling a stope is to allow as little pressure as possible on the barricades, and for this reason the area to be filled should be as long as possible, and the water should have a free outflow.

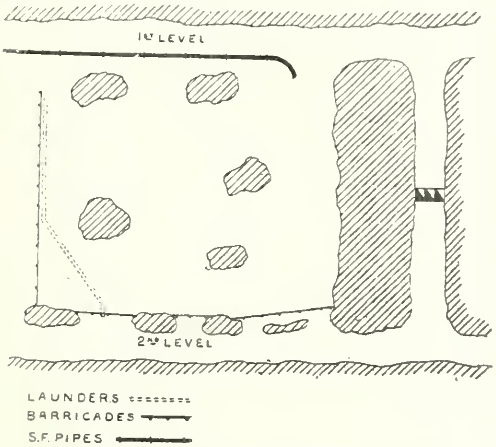


FIG. II.—STOPE READY FOR FILLING.

The outflow is usually through openings in the side barricade; these are 12 in. wide and the full height of the stope. As the sand rises the openings are closed from the bottom so that only the water lying on the sand can get away. These openings are so spaced up the barricade that water is flowing out at the bottom of one before the lower one is closed right up.

This outflow is always as far as possible from the inflow, so that as much sand as possible settles along the bottom of the stope, but even when this length is considerable (500 ft.) it is necessary to clear the water by overflow over a series of weirs and settling sumps before it is allowed to reach the pump.

Filling is at present proceeding in old stopes only but it is intended as soon as possible to fill directly behind the present stope faces. It is possible that this method of packing may induce the mines to alter their present method of stoping, as it may pay to break the ground in strips along the strike starting off from the drive. In this case a stope would be started by cutting away 50 or 60 ft. up the dip, building a timber barricade where our present drive pillars are; filling the area stoped with sand, laying rails on the sand and then stoping up again and so on to the top of the drive.

Sand-filling would be equally applicable to our present method of stoping, but the scheme I mention would have the advantage that drive pillars would be saved and shovelling and tramming cheapened. However this is a matter for the future and only experience will prove whether it is practicable or not.

I hesitate before touching on the chemical questions before such an audience, and will only mention that at present we are lowering only sand from old dumps which contain no trace of cyanide. As an additional safeguard recommended by Dr. Moir, we are sampling the dumps ahead of our work and testing for acidity, the idea being that by the time the sands have become acid all traces of cyanide will have disappeared. Our only chemical treatment at present is the addition of lime to save the pipes and pumps, but we hope in the near future, with the help of some of the members of this Society, to have a process which will enable us to use current sands with absolute safety

Our experiments in concentration and gold recovery have not yet given us results worth discussing, but we trust to pay some of our expenses by this means.

I regret that owing to the short time the work has been in progress, I can give you no details as to the probable cost of the process, but I think that, after hearing the advantages which I will summarise, you will agree with me that even if it prove expensive, sand-filling should be universally adopted.

Advantages.

1.—First and foremost, greatly increased safety for our mining population.

2.—Increased security of shafts and mine workings, which are among the biggest assets of United South Africa.

3.—Improved ventilation underground—

(*a*) Since practically the same amount of air will pass through a smaller area, its velocity must increase.

(*b*) The air can be guided into the places where men are working, and will not be dissipated in worked-out areas.

4.—Increase of ore reserves, as fewer or no pillars will be left and hence become an asset to the country and shareholders.

5.—Reclaiming of old pillars and footwalls, impossible without a sound method of supporting the hanging.

6.—Removal of dumps from the surface thus diminishing the dust nuisance, improving the aspect of the town and setting free large areas of ground for useful purposes.

7.—Possibility of mining at great depths. When getting near the maximum crushing strain of quartzite, supporting ground by pillars, waste packs and sticks is almost absurd except as a temporary measure.

Finally, I have to express my gratitude to Mr. Waller and many other technical gentlemen who have assisted me in the work which I have endeavoured to describe to you, and also to thank the firm with which I am connected for giving me permission to publish the results of experience gained on their mines.

Mr. Chas. B. Saner (*Vice-President*): In moving a vote of thanks to Mr. Pam for his able and interesting paper I will, with your permission, read the following few notes:—

I may mention that at the Village Deep we have been running sand underground for the last month, and at present can quite comfortably handle 1,200 tons per day. The pipe line goes down an old shaft at the Village Main Reef through old stopes and drives; down their main incline and along our top level; the pipe line has numerous twists, bends and curves, yet the sand flows well to its destination. The great secret is to first flow clear water through for three or four minutes *before* sending down sand, and to always run clear water through for a similar period after shutting off the sand, otherwise the sand soon settles in any horizontal portion of the pipe line and chokes it and so causes trouble.

Another essential for successful and smooth work is to have at least two separate places to fill into so as to have continuous running. In our old stopes we find great trouble in making the pillars over the drives water and sand tight, as they have taken weight and so both the foot and hanging are cracked, and I think it will be found quicker and cheaper to go up the stope 15 ft. or 20 ft. and erect the barricade along the stope continuously, ignoring all pillars. The drive pillars can be taken away, and, if found practicable, could be later taken down and used again, the sand either remaining solid and vertical, or taking the natural angle of repose without falling into the drive, this, of course, depending on the dip of the stope. The sand after 24 hours does not become solid, for as soon as water is put on again, it becomes saturated and is like a quicksand or jelly. It is surprising the quantity of hay used to fill up old cracks, etc., in the foot and hanging, making thus yet one more instance of the mining industry materially benefitting the farmers. I feel quite certain that this sand filling will benefit mining in several ways in the near future when we can fill close behind our working faces.

As the author mentions, we shall be able to guide the air to the working faces instead of its being lost in disused and worked out stopes. The hanging wall will be supported throughout. We shall cut away everything, leaving few, if any, pillars, and use much less timber. I quite foresee that we shall, before long, run the sand from the cyanide tanks after treatment by belts or other means direct to the sluicing bins.

Mr. W. T. Anderson (*Member*): I have very much pleasure indeed in seconding this vote of thanks to Mr. Pam for the interesting paper he has given us. I could have wished that the author had prefaced his paper with some remarks on the causes which have led to the commencing of sand filling on the mines of the Witwatersrand. The sand is not being sent underground in order to get rid of it or to make the face of the country more charming than it is at present. On moonlight nights the dumps may sometimes remind us of the white cliffs of Dover, but I think we must forego that pleasure if we can accomplish at the same time a purpose that will be of great benefit to the mining industry generally. Those of you who have lived over workings south of the main outcrop will have experienced the earth movements which are taking place day by day. These earth movements are not simply local, but extend over large areas of the Rand. We are striving to prevent any likelihood of subsidence in these areas. A start has been made which I hope will be the beginning of bigger things. Personally, I look forward to the time when the whole of the dumps on the Central Rand will have disappeared. The difficulties in connection with sand filling are not great, being only matters of detail, but they prevent that uninterrupted work which is necessary to carry out any project successfully. One of the objects, I think, in which members of this Society can help mining men, is to overcome any possible bad effects from sending down current sand into the workings. It would be of considerable assistance to the rapid filling of old workings if some cheap means could be discovered of destroying the cyanide in the current sand as it is taken from the tanks day by day. This would save the cost of transferring the sand to the dump, and of handling it again in a couple of years' time. The author has done the mining industry great service by reading this paper, and I hope it will promote research upon so great a subject. I have great pleasure in seconding the vote of thanks to the author.

Prof. J. A. Wilkinson (*Member of Council*): For the benefit of those members of the Society who may not know where the operations described in the paper which has just been read, are being carried out, I have been requested to ask the author if he would be good enough to place this on record as it is not stated in the paper.

Mr. E. Pam (*Member*): At the present moment sand filling is progressing on the Village Deep and on the Ferreira Deep. Several other mines will shortly be starting the work, and in a short time I believe the whole of the Central Rand will have commenced operations.

Mr. K. Sartorius (*Member*): I should like to ask how Mr. Pam is going to support his whole mass of sands against any rush on his walls and drives. Sand is a mass which has absolutely no cohesion at all as soon as dry.

Mr. Pam said he would answer the question in his reply.

Mr. E. J. Laschinger (*Member of Council*): I would like to make a few remarks in regard to this most excellent paper. From the description Mr. Pam has given as to how the operations are carried out, I am either under a misapprehension or else the operations are conducted on what I consider to be a fallacy. The author says that the surplus water is drawn off from the top of the settled sands and not from the bottom drive, in order to relieve the pressure on the barricades. The usual practice in Germany is to allow the water to percolate through the filled in sands through a barricade pervious to water but impervious to the sand. Now the whole of the filled in sand is saturated with water irrespective of the question whether the water is drawn from the top or the bottom and, therefore, the static pressure on the barricades is the same whichever method of drawing off the water is used. There is further the advantage in using the percolation method, that there would be less danger of the sand overflowing with the water and thus causing less trouble in the catch sumps and return water pumps.

It might be pointed out that our problems of sand filling on the Rand are much more complicated and present more dangers and difficulties than those in Germany. Our stopes are mostly steep, and the depths at which filling is to be carried out great. In Germany the seams are flat and at shallow depths. The dangers we have to face are that barricades may break or rushes of underground water due to rains may wash down sands into the workings. It might be suggested that Portland cement could be mixed with the first sand sent down against the barricade of a new fill, and thus after a few days or weeks at the most a solid protecting wall of artificial stone would protect the drive immediately below the fill.

Mr. Chas. B. Saner (*Vice President*): If Mr. Laschinger would come underground and have a look for five minutes rather than theorise here, it would make his remarks more valuable.

Mr. K. Sartorius: Mr. Saner, after handling a few hundred tons in the Village Deep, considers himself already an expert in the sand filling method, but I believe it would be better to theorise here than run the risk of any accidents. Furthermore, I want to learn from Mr. Saner's experience.

The vote of thanks was carried unanimously.

THE DESTRUCTION OF CYANIDE.

By James Moir D.Sc. (Vice-President) and
Jas. Gray, F.I.C. (Member).

Dr. J. Moir read the following paper:—

This paper takes its origin, on the practical side, in the necessity which has arisen of replacing in the worked-out stopes of the Rand the sand from the residue-dumps, with the view of preventing those subsidences which are nowadays swelling the accident and death-rates of the mines, and are causing alarm in the towns under the guise of supposed earthquakes. On the theoretical side, the paper is written to give an account of experiments which contradict practically everything that is to be found in the text-books on the ferro-cyanide reaction, *i.e.*, the reaction between cyanide and ferrous salts.

A preliminary survey of the possible methods of destroying cyanides showed that, although complete destruction of cyanide can be achieved by several methods (such as the action of silver and cuprous salts and of ammonium disulphide), the ferrocyanide reaction was the only one holding out any hope of rapidity of action and reasonable cost: but when we started out to find out the best conditions, we soon discovered that the text-books were "at sea" in almost every possible particular. We found (1) that when *excess* of alkali is used, the amount of ferrocyanide formed is less than when smaller quantities of alkali are present; (2) that rise of temperature above 20° C. is harmful to the reaction; (3) that the results are nearly independent of dilution; (4) that the reaction is as complete in 5 or 10 seconds as after long standing; and (5) that unless the alkalinity be most carefully adjusted to suit the amount of iron used, an excess of ferrous solution gives no better results than the theoretical quantity (that required by the equation: $6KCN + 2FeSO_4.7Aq = K_2Fe(FeCy_6) + 2K_2SO_4$, etc.

It will be seen from the above that in no case have we succeeded in obtaining entire destruction of the cyanide used, although in several cases we have obtained a reaction sufficiently complete for all industrial purposes; the cyanide having been reduced to 3 or 4 parts per million.

We have had some difficulty in establishing an analytical method for these solutions, which should be free from fallacy. To begin with, we have to point out a popular superstition of the Rand, viz., that when a solution goes turbid with one drop of silver nitrate, it is thereby proved to be free from prussic acid. This is quite a mistake. It is merely evidence of the absence of cyanide (cyanidion, to be exact), whereas any amount of hydrocyanic acid may be present and will not be titrated by silver nitrate unless the solution be made strongly alkaline.* If for example 10 c.c. of $\frac{N}{10}$ cyanide be treated with slight excess of acid (say 10·5 c.c. $\frac{N}{10}$ acid) and then titrated with $AgNO_3$, the solution will go turbid with one drop, although it contains enough prussic acid to poison the operator. If thereon excess of alkali be added the solution will titrate up to about 95 % of the original cyanide taken, the balance having gone off into the air as HCN during the acid stage. For this reason a great many of the Rand cyanide strengths are too low to the extent of from 10 to 40 %† of the figure returned, because the titration is not correct unless the "protective alkali" greatly exceeds the cyanide present.

The strongly alkaline condition of the solution before titration being thus necessary, we were faced with the difficulty that the use of alkali was bound to alter the composition of the reaction-product which we wished to investigate. For example, it breaks down $K_2Fe_2Cy_6$ into K_4FeCy_6 and $Fe(OH)_2$, and the latter would attack the free cyanide which we wished to estimate in the solution. We have got over this difficulty by two methods, viz. (1) oxidation of the reaction-product before rendering alkaline; (2) coagulating the reaction-mixture with comparatively inert substances such as magnesia, chalk, kieselguhr, charcoal, etc., so as to enable it to be filtered before adding the strong alkali.

We now proceed to describe our experimental method and the results :—

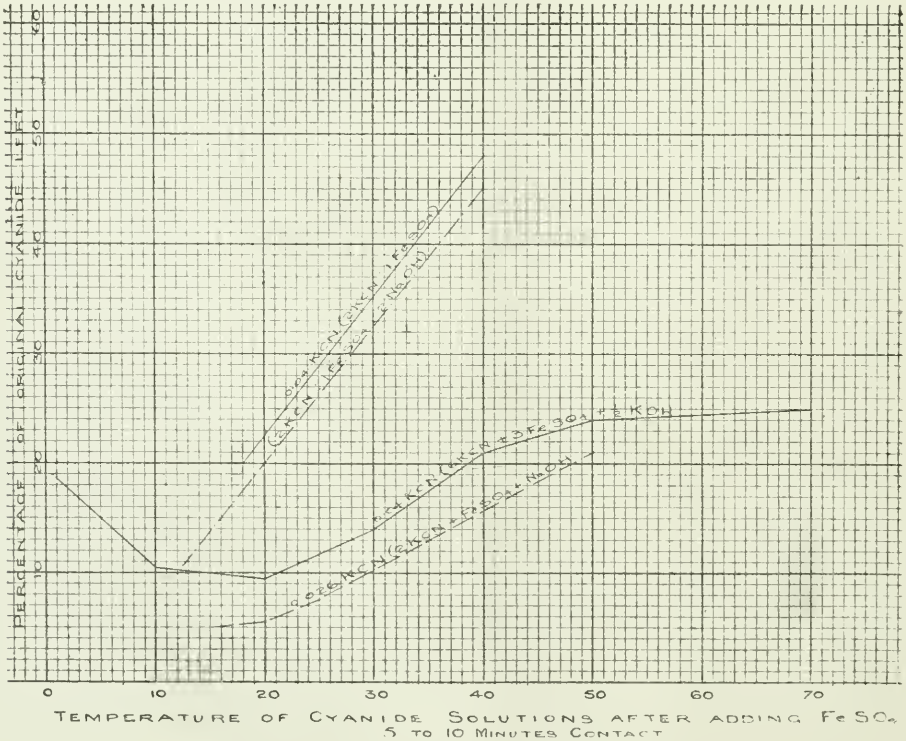
Effect of Temperature: The deleterious effect of heating was first noticed in trying to improve a bad "conversion" by heating the cyanide solution before adding the iron. About 400 c.c. of 0·01 % KCN was treated with $FeSO_4$ in the proportion 6 KCN : 3 ($FeSO_4 + 7Aq$) both dissolved in tap water, the mixture being worked

---

* See experiments in S.A. Association for Advancement of Science Report for 1904, p. 59.

† These higher errors only occur when the solution contain zinc.

Curves showing that the "Destruction" of Cyanide is best at Ordinary Temperatures.



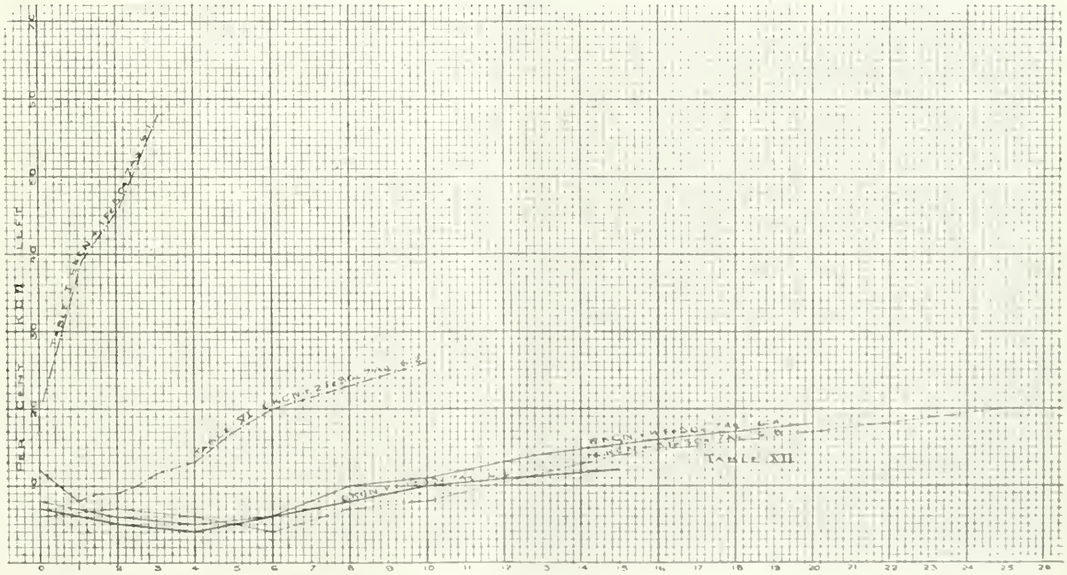
Temperatures are Centigrade. Strength of Solution shown on each Curve.

up after a minute by adding an acid solution of chromium trioxide (or of potassium bichromate), and, immediately after, an excess of caustic soda, and titrating the filtrate from the $Fe(OH)_3$ and $Cr(OH)_3$. The cyanide left was 20% of original. On heating the cyanide from 18° to 40°C., and working up as before, 48% of the cyanide was left undestroyed. [Both these results are abnormally high owing to the use of tap water containing CO_2, for it should be noted in passing that ferrous sulphate reacts only with cyanide and not at all with free hydrocyanic acid]. The above experiment was repeated with addition of alkali to the cyanide solution ($6KCN : 3FeSO_4 : 6NaOH$). The cyanide left was 10% at 12°C. and 45% at 40°C. Two systematic researches were then made. In the first the molecular ratio was $6Cy' : 3Fe^{\cdot\cdot} : 3OH'$, using 130 c.c. of 0·02% KCN solution heated (or cooled) to the required degree before adding the iron solution and after adding the alkali; the product was worked up by the chromic acid method. The cyanide left was 5% at 15°C, 5·5% at 20°C, 7·5% at 25°, 13% at 35°C and 21% at 50°C. In the second series the alkali was omitted, but an impurity of about one-half molecule was present (counted as $\frac{1}{2}K_2CO_3$) in the cyanide used, and the cyanide solution taken was roughly 0·04%. The product was worked up as above, but ten minutes contact was given with such cold solutions. The percentages of cyanide left were: at +1°C, 18·5%; at 10°, 10·5%: at 20°, 9·2%; at 30°, 14%; at 40° 20·8%: at 50° 23·6%; and at 70°, 25%. There is thus a well-marked "optimum-temperature" under these conditions, situated at about the ordinary temperature. This phenomenon is very difficult to account for, and the best explanation we have to offer is that low temperature diminishes the dissociation (and therefore availability) of ferrous hydrate, whilst high temperature polymerises (coagulates) it and throws it out of action at once. The different behaviour of the solutions on adding the $FeSO_4$ seems to support this idea, since the hot solutions give an immediate dark precipitate (? Fe_3O_4), and the cold solutions a clear dark orange coloration, which takes 5 to 10 seconds to change into the ordinary green reaction-product (? $K_2Fe_2Cy_6$).

(*b*). *Effect of Addition of Alkali Before Iron*: This portion of the work commenced in an attempt to improve bad "conversions" by adding caustic alkali, with the object of lessening the hydrolytic decomposition of the cyanide. The mathematical equation governing this action is $[HCN][OH'] = K[Cy']$ where K is about $\frac{1}{120000}$ at ordinary temperatures; hence, since cyanide present as HCN does not react with iron, we hoped by addition of alkali to diminish the HCN and so increase the conversion. Experiment, however, showed that other factors come into play, with the result that, whilst a little alkali is beneficial, an excess of alkali is most harmful.

To save space the results are tabulated. It should be noted that the chromic acid method gives somewhat irregular results, because a little HCN escapes into the air during even a momentary acid stage.

Curves showing the Effect of Gradual Increase of Alkalinity on the "Destruction" of Cyanide.



Each Curve is got with a Definite Molecular Ratio of Iron to Cyanide, varying from 1 to 8 Mols. $FeSO_4$ to every 6 of Cyanide.

TABLE I.—6KCN : $1FeSO_4 \cdot 7H_2O$, varying alkali and acid. 200 c.c. of N/100 KCN used : product treated with $KMnO_4$ till green, then NaOH.

Acid before Fe.	% KCy left.	Alkali before Fe.	% KCy left.
mols.		mols.	
5·9	86	0·1	21
3·9	51	1·1	39
2·9	37	2·1	46
1·9	33	3·1	78
0·9	24		
0·4	22		

TABLE II.—6KCN : $2FeSO_4 \cdot 7H_2O$: varying alkali before iron : contact 30 sec. : temperature about 20°.

KCN Taken.	Volume of Solution.	$FeSO_4$ 7Aq.	N/10 Alkali.		% KCN Left.
grams.	c.c.	grams.	c.c.	mols.	
0·04	203	0 057		0	12·5
,,	204	,,	1	= 0·97	9·5
,,	205	,,	1 5	1·45	6·4
,,	,,	,,	2 0	1·95	5·5
,,	,,	,,	2·5	2·4	14·3
,,	206	,,	3	2·92	23·8
,,	208	,,	5	4·9	37·5
,,	210	,,	7	6·85	41·2
,,	212	,,	9	8·8	52·2
,,	214	,,	11	10·7	65·9

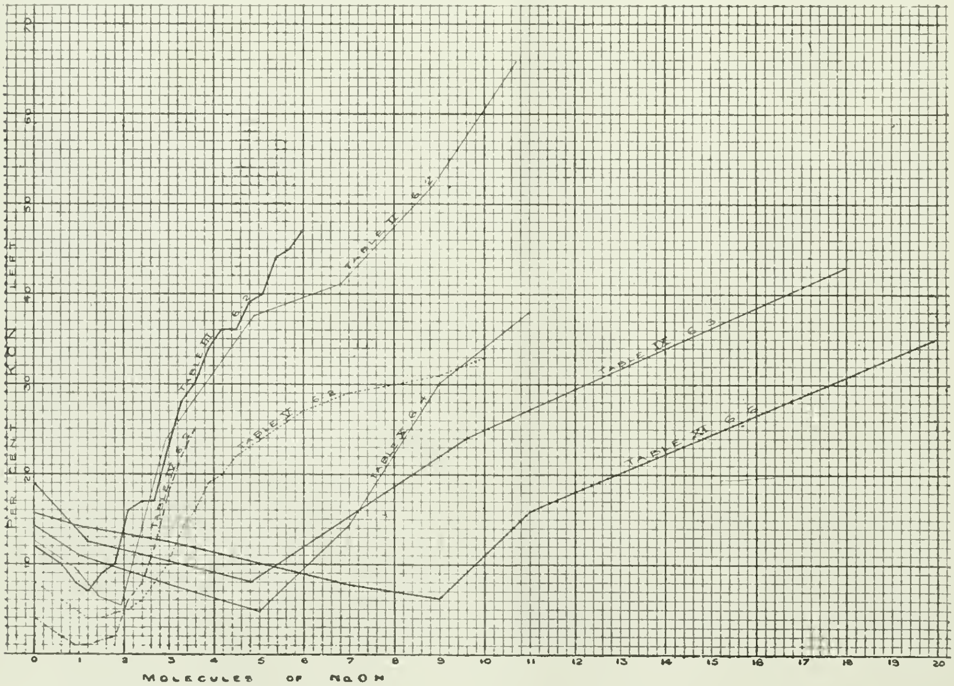
Reaction stopped with CrO_3 in H_2SO_4

TABLE III.

6KCN . $2FeSO_4 \cdot 7H_2O$ (constant volume of 100 c.c., 30 sec. contact : *excess* CrO_3.

KCN Taken.	$FeSO_4$7Aq	N/10 Alkali.		% KCN left.
grams.	grams.	c.c.	mols.	per cent.
·065	0· 927	0		12
,,	,,	½	= 0 3	11
,,	,,	1	0·6	10
,,	,,	1½	0·9	8
,,	,,	2	1·2	7
,,	,,	2½	1·5	9
,,	,,	3	1·8	10
,,	,,	3½	2·1	16
,,	,,	4	2·4	17
,,	,,	4½	2·7	17
,,	,,	5	3	23
,,	,,	5½	3·3	28
,,	,,	6	3·6	30
,,	,,	6½	3·9	34
,,	,,	7	4·2	36
,,	,,	7½	4·5	36
,,	,,	8	4·8	39
,,	,,	8½	5·1	40
,,	,,	9	5·4	44
,,	,,	9½	5·7	45
,,	,,	10	6	47

Curves showing the Effect of Varying Iron and Alkali on Cyanide : also the Varying Stability of the Product.



II. and III. give Effect of Chromic Acid, and IV. and V. that of Strong Alkali, on the Product of the Theoretical Proportions. The Others give Effect of Excess of Iron.

TABLE IV.—$6KCN : 2FeSO_4 \cdot 7H_2O$ and varying alkali *before* iron. Product coagulated by conc. alkali *after* iron, and filtered after addition of magnesia. 100 c.c. N/100 KCN taken.

Mols. Alkali.	% KCN Left.	Mols. Alkali.	% KCN Left.
0	4*	2·1	6
0·3	3	2·4	8
0·6	2	2·7	12
0·9	1	3·0	18
1·2	1	3·3	22
1·5	1·5		
1·8	2	3·6	25

* MgO *before* alkali leaves about 2%.

TABLE V.—As above, using definite amount strong alkali at end (5 c.c. of 1% NaOH).

0	8	3·3	14
0·3	7	3·6	16
0·6	6	3·9	19
0·9	5	4·2	20
1·2	4	4·5	22
1·5	4	4·8	23
1·8	4·5	5·1	24
2·1	5	5·4	25
2·4	6	5·7	26
2·7	8	6·0	27
3·0	10	7·0	29
		8·0	30
		9·0	31
		10·0	33

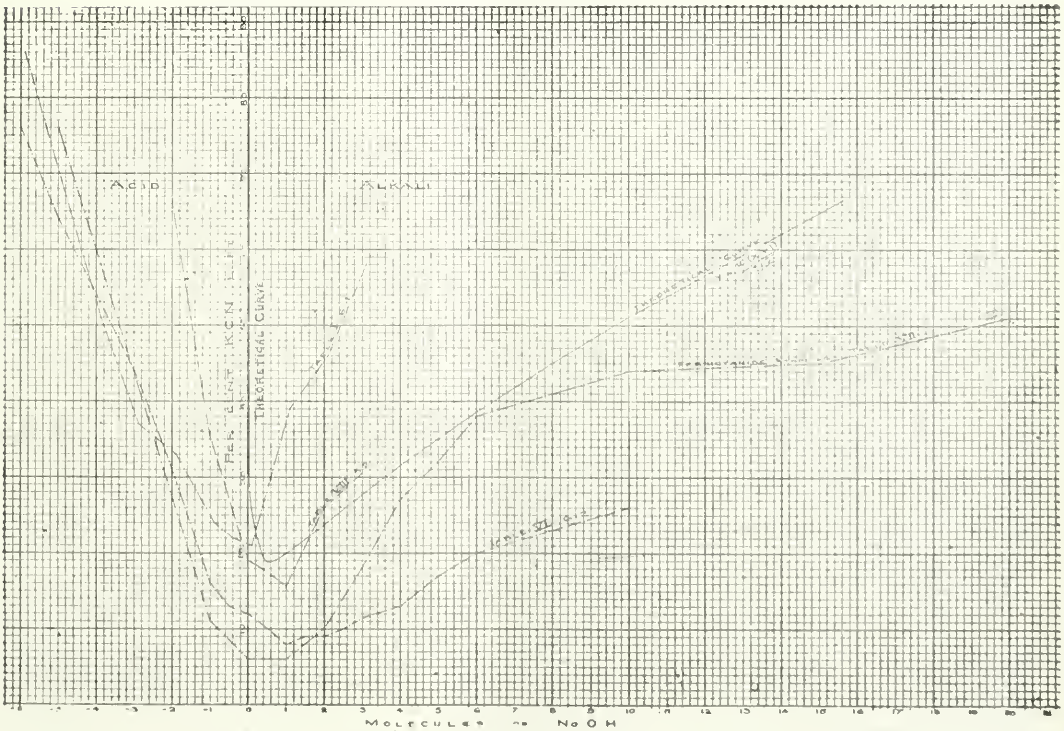
TABLE VI.

6 : 2 reaction stopped with N/10 $KMnO_4$ (4 c.c.) = equivalent of total iron.

KCN.	$FeSO_4$ 7Aq.	N/10 NaOH.	Time.	% KCN Left.
gram.	gram.	mols.	secs.	
0·065	0·0927	− 6·0*	30	76·0
,,	,,	− 5·0	,,	64·0
,,	,,	− 4·0	,,	54·0
,,	,,	− 3·0	,,	14·0
,,	,,	− 2·0	,,	31·0
,,	,,	− 1·0	,,	16·0
,,	,,	− 0·5	,,	13·0
,,	,,	+ 0·0	,,	12·0
,,	,,	+ 0·5	,,	10·0
,,	,,	+ 1·0	,,	8·0 (11% twice)
,,	,,	+ 1·5	,,	9·0 (10% once)
,,	,,	+ 2·0	,,	9·0
,,	,,	+ 2·5	,,	10·0 (12 & 8%)
,,	,,	+ 3·0	,,	11·5
,,	,,	+ 4·0	,,	13·0
,,	,,	+ 5·0	,,	17·0
,,	,,	+ 6·0	,,	20·0
,,	,,	+ 8·0	,,	23·0
,,	,,	+ 10·0	,,	26·0

* Minus alkali is HCl before Fe.

Curves showing Unbroken Continuity between Effects of Acid and Alkali on the Reaction.



Neutral Point. Neutral means Neutral to Phenolphthaleïn after adding Theoretical $AgNO_3$. Methylorange Neutral Point is at − 6.

TABLE VII

As above, varying alkali and acid, 100 c.c. N/100 KCN taken: product + K_3FeCy_6 equivalent to total Fe, then heated with NaOH and MgO.

Acid.	% KCN Left.	Alkali.	% KCN Left.
mols.		mols.	
5	76·0	0·5	6·0
3	44·0	1·0	6·0
1	11·0	2·0	10·0
0	6·0	3·0	18·0
		4·0	27·0
		5·0	32·0
		6·0	38·0
		10·0	44·0
		15·0	45·0 (av.)
		20·0	51·0

TABLE VIII.

$5KCN$* : $2FeSO_4 \cdot 7H_2O$, varying alkali and acid, 100 c.c. N/100 KCN taken: product + $KMnO_4$ equivalent to total Fe, then heated with NaOH and MgO.

Acid.	% Cy Left.	Alkali.	Cy Left.
mols.		mols.	
2	66·5	½	17·5
1	35	1	15·7
0	19	2	26
		3	30·5

*$10KCN + 4FeSO_4 + O + H_2O = 2(FeCy_2.FeCy_3) + 4K_2SO_4 + 2KOH$

TABLE IX.

$6KCN : 3FeSO_4 \cdot 7H_2O$ (alkali before iron; contact 30 sec.), temperature about 15° C.

KCN Taken.	Volume	$FeSO_4$ 7Aq.	N/10 Alkali.		KCN Left.
gram.	c.c.	gram.	c.c.	mols	
·065	217	·139	0		19
,,	,,	,,	2 =	1·2	12·5(av
,,	,,	,,	4	2·4	11
,,	,,	,,	8	4·8	8
,,	,,	,,	16	9·6	24
,,	,,	,,	30	18	43

Reaction stopped with excess acid and CrO_3.

TABLE X.—$6KCN : 4FeSO_47Aq$.

KCN Taken.	Volume of Solution.	$FeSO_4$ 7Aq.	N/10 Alkali.		% KCN Left.
grams.	c.c.	grams.	cc.	mols.	
0·04	203	0·114	0		14·3*
,,	204	,,	1 =	1(app)	11·0
,,	206	,,	3	3 ,,	7·8
,,	208	,,	5	5 ,,	1·8
,,	210	,,	7	7 ,,	14·3
,,	212	,,	9	9 ,,	30·0
,,	214	,,	11	11 ,,	38·0

Reaction stopped with CrO_3 in H_2SO_4.

TABLE XI.—$6KCN : 6FeSO_47Aq$.

grams			cc.	mols.	
0·04	203	0·171	0		15·8†
,,	204	,,	1 =	1	11·3(av.)
,,	206	,,	3	3	12·5
,,	208	,,	5	5	14·3 (?)
,,	210	,,	7	7	7·8
,,	212	,,	9	9	6·3
,,	214	,,	11	11	15·9
,,	223	,,	20	20	35

Reaction stopped with CrO_3 in H_2SO_4.

---

* 27 % with excess CrO_3.

† 26 % with excess CrO_3.

TABLE XII.—Excess iron and varying alkali. Reaction stopped by $KMnO_4$ equivalent to total Fe.

KCN	Volume of Solution.	$FeSO_4$ 7Aq.	NaOH	% KCN Left.
	c.c.	mols.	mols.	
0·065	106	4·0	0·0	8
,,	,, + 3·34	,,	2·0	6
,,	,, 6·68	,,	4·0	5
,,	,, 10·02	,,	6·0	6
,,	,, 13·36	,,	8·0	10
,,	,, 16·7	,,	10·0	11
,,	,, 21·71	,,	13·0	14
,,	,, 26·72	,,	16·0	16
,,	,, 33·4	,,	20·0	18
,,	109 +	6·0	0·0	7
,,	,, 3·34	,,	2·0	5
,,	,, 6·68	,,	4·0	4
,,	,, 10·02	,,	6·0	6
,,	,, 13·36	,,	8·0	8
,,	,, 16·7	,,	10·0	10
,,	,, 25·05	,,	15·0	12
,,	112	8·0	0·0	6
,,	112 + 3·34	,,	2·0	7
,,	,, 6·68	,,	4·0	6
,,	,, 10·02	,,	6·0	4
,,	,, 13·36	,,	8·0	7
,,	,, 16·7	,,	10·0	8
,,	,, 25·05	,,	15·0	14
,,	,, 33·4	,,	20·0	17
,,	,, 41·75	,,	25·0	20

The cyanide was heated to 19° C. and the reagents added. After the reaction was stopped the solution was filtered without warning.

Discussion of function of Alkali.—It will be seen that each ratio of iron to cyanide has an *optimum* alkali which is not directly proportional to the iron. This *optimum* is less (speaking in molecular proportions) than the iron when the latter is low, and greater than the iron when the latter is high. This relationship may be approximately expressed by the equation: $^{*}\mathrm{OH}' = 2\mathrm{Fe}^{\cdot\cdot} - 2{\cdot}5$ which gives $\mathrm{OH}' = 9{\cdot}5$ when $\mathrm{Fe} = 6$: $\mathrm{OH}' = 5{\cdot}5$ when $\mathrm{Fe} = 4$: $\mathrm{OH}' = 3{\cdot}5$ when $\mathrm{Fe} = 3$: $\mathrm{OH}' = 1{\cdot}5$ when $\mathrm{Fe} = 2$; and $\mathrm{OH}' = 0$ when $\mathrm{Fe}^{\cdot\cdot} = 1{\cdot}25$. (6 mols. KCy always implied). The reason for the depressing effect of excess of alkali is undoubtedly its action on the solubility of ferrous hydrate; but we have grave doubts as to its being a simple case of mass-action and are inclined to ascribe the effect to polymerisation and coagulation. According to the mass-action law, the amount of active iron is defined by the equation $[\mathrm{Fe}^{\cdot\cdot}]\,[\mathrm{OH}']^2 = \mathrm{K}_1$ (which is a constant only when coagulation of $\mathrm{Fe(OH)}_2$ to an abnormally insoluble form is excluded): again the other equation cannot be $[\mathrm{Fe}^{\cdot\cdot}]\,[\mathrm{Cy}']^6 = $ constant, because (1) the concentration of the cyanide seems to make very little difference, and (2) the reaction is irreversible—even strong oxidising agents like acid CrO_3 do not liberate HCN from ferrocyanide or ferricyanide. An approximation (possibly accidental however) to the form of these curves can be got, by assuming the three equations: (1) $[\mathrm{Fe}^{\cdot\cdot}]\ [\mathrm{OH}']^2 = \mathrm{K}_1$; (2) $[\mathrm{Fe}^{\cdot\cdot}]\ [\mathrm{Cy}']^3 = \mathrm{K}_2$;

$$(3)\ \underbrace{[\mathrm{HCN}] + [\mathrm{Cy}']}_{\text{Titratable as Cy}' \text{ after alkali.}} = [\mathrm{Cy}']\left[1 + \frac{\mathrm{K}}{\mathrm{OH}'}\right] = \text{say } y.$$

$$\text{From (1) and (2)}\ \frac{[\mathrm{OH}']^{\frac{2}{3}}}{[\mathrm{Cy}']} = \sqrt[3]{\frac{\mathrm{K}_1}{\mathrm{K}_2}},\ \text{hence}$$

$$y = \sqrt[3]{\frac{\mathrm{K}_2}{\mathrm{K}_1}}\left([\mathrm{OH}']^{\frac{2}{3}} + \mathrm{K}[\mathrm{OH}']^{-\frac{1}{3}}\right)$$ which is a curve possessing something like the right type of minimum.

Effect of dilution (strength of cyanide solution operated on).—It was to be expected from every known theoretical consideration, that the results obtained with very dilute cyanide solutions would be much worse than those given by stronger solutions. Experiments were therefore done, taking 0·1% KCN as the maximum strength ever likely to occur in practice, and comparing results obtained when the same amount of the reagents were added to the same weight of cyanide dissolved in varying quantities of water.

All except the last trio were worked up by the "chromic acid method." They show that dilution has almost no effect, or that such effect as there is varies inversely as the cube (or fourth) root of the concentration.

Effect of different methods of working up the reaction product.—It certainly appears from our work that the primary product of the reaction between cyanide and ferrous iron is an unstable compound, which contains cyanogen in two different forms, one permanently bound and innocuous, and the other loosely held and separable by processes of oxidation. The reaction product, according to the text books, is $\mathrm{K}_2\mathrm{Fe}_2\mathrm{Cy}_6$, generally called potassium ferrous ferrocyanide, but judging from our experiments detailed on the next page it has not only its iron in two different states of combination, but also its cyanogen. The following series of experiments, in which the original mixture was $6\mathrm{KCN} + 2\mathrm{FeSO}_4$ without alkali, will explain this.

Table showing effect of dilution only (see above).

	Final Volume.	Per cent. Cy left.	Ratio Cy : Fe.
(α) 0·092 gm. KCN and 0·066 gm. $\mathrm{FeSO}_4 . 7\mathrm{Aq}$ without alkali (neutralised tap water)	100 c.c.	26	6 : 1
	200 c.c.	24	
(β) 0·102 KCN and 0·072 FeSO_4 (distilled water)	100 c.c.	25·0 (average)	6 : 1
	200 c.c.	27·0 (average)	
(γ) 0·092 gm. KCN and 0·132 gm. $\mathrm{FeSO}_4 . 7\mathrm{Aq}$ without alkali (tap water)	100 c.c.	13·5	6 : 2
	200 c.c.	15·5	
(δ) Same as (β) but giving only 5 to 10 seconds contact	100 c.c.	24·0 (average)	6 : 1
	200 c.c.	25·8 (average)	
(ε) Same as (β) with 0·004 gm. NaOH - -	100 c.c.	18·9	6KCy : FeSO_4 : ¼NaOH
	200 c.c.	20·5	
(ζ) 0·102 KCN and 0·144 FeSO_4 - - no alkali (distilled water)	100 c.c.	6·0 (average)	6Cy′ : 2Fe··
	200 c.c.	6·5	
	600 c.c.	11·5	
(η) 0·065 KCN and FeSO_4 no alkali - - worked up with equivalent KMnO_4 and alkali	100 c.c.	19·5	5 : 2
	400 c.c.	27·0	
(θ) As above, MgO after Fe, then NaOH - -	100 c.c.	16·0	
(ι) ·065 KCN and 2 mols FeSO_4, no alkali - finished with MgCO_3 and NaOH	100 c.c.	2	6 : 2
(κ) Do. diluted before adding iron - -	400 c.c.	8	
(λ) Do. Do. - -	600 c.c.	61	

---

*i.e., Molecules alkali = twice molecules iron less 2·5.

(Generally 100 c.c. of N/100 KCN was used.) In all cases NaOH was added last before titration :—

(1) Mixture treated with excess precipitated $CaCO_3$: cyanide in filtrate practically *nil* (1% once).

(2) Mixture treated with excess $MgCO_3$: cyanide in filtrate practically *nil* (2% once).

(3) Mixture treated with excess NaOH (addition of charcoal or MgO after NaOH made no difference) : cyanide in filtrate 4% of original.*

(4) Mixture treated with excess MgO : then NaOH and re filtration : cyanide in filtrate 1 to 2% of original.

(5) Mixture treated with excess talc : then NaOH and re filtration : cyanide in filtrate 2% of original.

(6) Mixture filtered twice directly : trace colloidal prussian blue removed by NaOH : cyanide in filtrate 4% of original.

(7) Same as 5, but NaOH before talc : cyanide in filtrate 5% of original.

(8) Mixture treated with sodium peroxide solution : cyanide in filtrate 11·5% (average).†

(9) Mixture treated with ammonium persulphate, then NaOH : cyanide in filtrate 3%, 6% and 8% (according to quantity NH_4SO_4).

(10) Mixture treated with ferric sulphate, excess NH_4SO_4, then NaOH : cyanide in filtrate 22%.

(11) Mixture treated with acid chromic acid, then excess NaOH : cyanide in filtrate 13% (average of 4) + 26% (excess CrO_3).

(12) Mixture treated with $KMnO_4$ until K_2MnO_4 visible, then NaOH : cyanide in filtrate 8%.

(13) Mixture treated with $BaSO_4$ (faintly acid) and filtered twice, then NaOH and refiltration : cyanide in filtrate 1% (partly due to loss of HCN in acid stage).

(14) Mixture treated with Kieselguhr and filtered twice, then NaOH and refiltration : cyanide in filtrate 2·5%.

(15) KCN treated with sodium sulphite, then $FeSO_4$, MgO and NaOH : about ½% of KCN left.

(16) Ratio $6KCy + 2FeSO_4 + 3NaOH$ gave by direct repeated filtration 10% KCy' left.

(17) Ratio $6KCy + 2FeSO_4 + 3NaOH$ gave with excess CrO_3 method 23% KCy' left.

(18) Ratio $6KCy + 2FeSO_4 + 6NaOH$ gave by direct repeated filtration 26% KCy' left.

(19) Ratio $6KCy + 2FeSO_4 + 6NaOH$ gave with $KMnO_4$ just in excess of Fe 20% KCy' left.

Effect of acid before $FeSO_4$.—(1) ·065 gm. KCN titrated with phenolphthalein and acid used 80% of the theoretical amount corresponding to to the Cy' contained therein, and this solution, treated with $FeSO_4$ (in ratio $6KCN + 3FeSO_4$), and worked up by CrO_3 method, returned 79 of the original cyanide ; *i.e*, the free HCN totally failed to combine with iron. The 80% of the titration represents roughly the partition of the K between the two weak acids HCN and phenolphthalein (this *Journal*, vol. viii, p. 124), but the end point is indefinite. Another experiment in which $6KCN + 6HCl + 2FeSO_4 + 2KMnO_4$ was used (in that order), finishing with NaOH and magnesia, returned 81% of the cyanide taken.* Addition of acid was tried in series before the iron, but the results were rather unconvincing, as the product was worked up by magnesia and caustic soda :—

KCN mols.	Fe mols.	HCl mols.	Cy' left.
6	2	0	4
6	2	0·3	5 to 7
6	2	0·6	8 to 9
6	2	0·9	10
6	2	1·2	11
6	2	1·5	14
6	2	1·8	13

The more reliable data are given in a previous paragraph [Tables I., VI., VII. and VIII.]

Miscellaneous data.—A. "Sodium cyanide" supplied to one of the mines, when made into M/200 strength in Cy' required 5·00 c.c N/10 silver with KI indicator and was thereon *neutral* to phenolphthalein, 10·70 c.c N/10 HCl with

Experiments on variation of the proportion of iron.—

Ratio (molecular) of KCN to $FeSO_4$:		6 : 1	6 : 2	6 : 3	6 : 4	6 : 5	6 : 6		
(1) *Chromic acid stopping method :*									
(*a*) 100 c.c. of 0·1% KCN	:	59	25		9·6			Cy'% left	
(*b*) 200 c.c. of 0·02% KCN	:		12·5		14·3		16	,,	
(2) *Strong caustic soda method*	:	33	4 to 7		9		10	,,	
(3) *Ammonium persulphate* + NaOH	:	34	6	10	22	23	23	,,	
(4) *Magnesia* + NaOH *filtration*	:	23	1 to 2	0	0	0	0	,,	T = 18°C
(5) *Alkaline permanganate stopping*	:	21	12		8		7		
(6) *Do. with 2 mols. alkali before* Fe	:	45	9		6		5		

We think it possible that less than two molecules of ferrous sulphate will suffice, if excess of magnesia and practically no other soluble alkali is present.

---

* 7% once, 4% thrice and 3% once, in the five experiments.

† In titration AgO came down instead of AgI.

* HCN lost by volatilisation.

methylorange and about 9·0 with phenolphthalein for 100 c.c.; which also after adding $BaCl_2$ gave $BaCO_3$ requiring 3·2 c.c. N/10 acid with methylorange and the filtrate required 8·87 c.c. N/10 acid. Sulphate was absent. Merck's pure KCN requires about 7 c.c. N/10 acid per gram to make it neutral, *i.e.*, so that when titrated with silver in presence of phenolphthalein the colour goes just as the turbidity appears: it requires 10·55 c.c. N/10 HCl with methylorange as above. The impurity must be mainly $KHCO_3$ not K_2CO_3, on account of the smallness of the alkalinity with Ag and phenolphthalein.

B. (*a*) A mixture of equivalent weights of K_4FeCy_6 and $FeSO_4$ (1 : 1) treated with excess alkaline $KMnO_4$ did not separate any cyanide.

(*b*) Potassium ferricyanide when treated with CrO_3 and dil. H_2SO_4, then NaOH gave no cyanide.

(*c*) A mixture of KCN, $Na_2S_2O_3$ and acetic acid did not destroy the cyanide.

(*d*) Addition of ammonium polysulphide to cyanide solutions completely destroyed it.

(*e*) Addition of $FeCl_3$ spoils the conversion by acting as an acid: this is not what one would expect from the text-books.

(*f*) *Destruction by cuprous salts.*—A mixture of $CuSO_4$, Na_2SO_3 and HCl reduces the cyanide to about 2 parts per million. The following series was tried also (30″ contact, then NaOH):

$6KCN + 1CuSO_4 + Na_2SO_3 \rightarrow 54\%$ Cy′ left

,, 2 & 3 ,, ,, $\rightarrow$ both about 5% left

Higher proportions produced apparent total destruction of cyanide, but in reality it is merely rendered non-titratable, as $KCuCy_2$, unless 6 mols. Cu are used. Apparently, however in acid solution the proportion $2KCN + CuSO_4$ gives almost complete precipitation. The following equations are probable:

$$6KCy + 2CuSO_4 = 2KCuCy_2 + 2K_2SO_4 + C_2N_2$$

$$(2)\quad C_2N_2 + 2NaOH = NaCy + NaOCN + H_2O:$$

hence theoretical ratio for *complete* destruction in *alkaline* solution is $3KCy + 2CuSO_4 + 2KOH = 2CuCN + 2K_2SO_4 + KOCN + H_2O$ whereas in acid sulphite solution the ratio is 1 : 1, thus

$$2KCy + 2CuSO_4 + Na_2SO_3 + H_2O = 2CuCN + K_2SO_4 + 2NaHSO_4.$$

Both seem to be prohibitively expensive.

(*g*) 200 c.c. N/100 KCN + Br_2 until yellow, after short contact made alkaline with either NaOH or NH_4HO, warmed gently and titrated with $AgNO_3$: cyanide left—*nil.*

(*h*) Excess of alkaline permanganate also destroys cyanide completely.

(*i*) Theoretical quantity chlorine (bleaching powder and acid) converts wholly to CNCl.

Theory of the incomplete conversion.—The fundamental equation is that already given, viz.: $\frac{[HCN]}{[Cy']} = \frac{K}{[OH']}$: applying this to a 0·065% KCN solution without protective alkali (viz.: $[Cy'] = \frac{1}{100}$) we get $[HCN][OH'] = \frac{1}{12000000}$ hence in such a *pure* solution, since [HCN] and [OH′] are equivalent and numerically equal, we have $[HCN] = [OH'] = \frac{1}{2000\sqrt{3}}$: hence hydrolysis expressed as a percentage of the cyanide is $\frac{100[HCN]}{[Cy']} = \frac{5}{\sqrt{3}} = 2·9\%$.

Speaking *a priori*, therefore one would never expect a higher conversion than 97·1%: and as a matter of actual experiment, the above solution titrates with silver only to 97% of the cyanogen in it. The addition of the iron solution, however, introduces another factor, viz.: alteration of the alkalinity ([OH′]). In all probability the concentration of OH′ is now reduced to that required by the equation, $[Fe^{\cdot\cdot}][OH']^2 = C_0$ (solubility product of $FeOH)_2$, equal to about $\frac{1}{7 \times 10^{11}}$. This concentration of OH′ is roughly $\frac{M}{7000}$ and the corresponding HCN concentration is therefore $\frac{M}{1700}$, roughly 6 of the total cyanide started with. This would appear to be the theoretical limit of conversion for centinormal cyanide when no adventitious alkali is present. A small addition of alkali will improve the conversion by lessening the amount of free HCN, but at the same time it diminishes the amount of available iron according to the square of the quantity of alkali used: the final result is that the best conversion is to be got by a compromise between these contrary factors.

In cyanide destroying for stope-filling purposes, the alkalinity of the water used must therefore be low, but after the conversion has occurred, a high alkalinity is desirable so as to prevent the residual traces of HCN getting into the air. Practical experiments show that when the cyanide is reduced to 0·002%, a saturated solution of lime will retain the HCN so well that it can barely be detected in the air of a closed vessel

ADDENDUM.

Since reading this paper, it has struck us that another point connected with cyanide destruction requires discussion: this is, that there is no possible method of removing the sodium and potassium also present in these sands, and that the water used in any such cyclic process will gradually become saturated with sodium sulphate and that therefore all the chemical reactions will be interfered with unless the water is occasionally changed.

Mr. H. A. White (*Member of Council*): I wish to propose a hearty vote of thanks to Dr. Moir and Mr. Gray for this able paper.

As I had the opportunity some few days ago of reading the excellent paper we have just heard, I thought it would be convenient to prepare a contribution to its discussion in time for this evening. During the last two months I have attempted to use my spare moments in the consideration of the problem of cyanide destruction with a view to using for "sand filling" the residue immediately after completion of treatment; I do not think that there would be much cyanide (as cyanidion) to destroy after a six months exposure on the dump.

Before describing my own experiments, I propropose to give a short criticism of the paper before us, although I can only take serious exception to the statement regarding cyanide titration. I hate superstition as much as the authors, but I do not think that any cyanider worthy of the name is ignorant that HCN can only be titrated by silver nitrate in the presence of excess of alkali. (See Vol. I., p. 165 of this *Journal*, where method is given for testing HCN (free) in the presence of cyanides.) Some authors object to anything but a very slight excess (Siebold, Sutton's Vol. *Analysis*, p. 217). In the presence of zinc, the addition of large excess of NaHO is made for the purpose of estimating total simple cyanide (this *Journal*, Vol IV., p. 474). This is very different from the "free" or available cyanide, in terms of which all our records on these mines are at present kept. The authors speak of an "error of 10 to 40 %," but the usual experience is that the "total cyanide" is from two to three times as much as the "free cyanide" registered. It must be remembered that our treatment is regulated in accordance with results obtained by fairly definite and consistent analysis, and an accusation of error should not be too lightly made. I regret that the authors have not more clearly indicated what they mean by "destruction of cyanide." I know it is wise to avoid definitions where possible, but in this case a real conflict is involved. I do not think that conversion of cyanides into ferrocyanides can fairly come under this heading, and I am doubtful if their change into cyanates or thiocyanates warrants this description. To change the issue slightly, I think the authors should bring forward some direct evidence that the ferrocyanide they are content to leave in the sand will not, under the conditions to which it will be exposed, give rise to HCN in more than infinitesimal quantities. I may say I have some little doubt on the point, but I am sure Dr. Moir, with the aid of his new indicators, can quickly settle the question. Beside the cyanide in the residual solutions, the presence of sulphocyanides and ferrocyanides must also be considered. In some cases only minute amounts of these substances are present, but this is not always the case. Leaving out of the question the ferrocyanides, which are possibly harmless, the presence of sulphocyanides may occasionally require careful consideration. I have already crossed lances with the senior author on the decomposition of these salts with ferric sulphate (or chloride), which is present in mine water and will be increased as a result of oxidation of the pyrite in the sand. (This *Journal*, Vol. VI., p. 274.) That HCN is a product of the reaction can now be easily demonstrated by means of hydrocoerulignone paper, or even phenolphthal*ine*. (This *Journal*, Vol. VII., p. 375.) (By the way, the new words "phene" and "phine" should be useful in place of phenolphthal*eine* and phenolphthal*ine*, especially considering the difficulty the A.M.O.S. has with the spelling and pronunciation of the latter pair.) It may seem thankless to steal the doctor's thunder with which to slay him, but really the compliment is a subtle one; if he tries this experiment I trust he will choose favourable conditions. Before leaving this point I would like to say that again I am doubtful if under likely circumstances any real danger is going to arise from this source, especially as in the dark the action is so slow, and for certain other reasons. I think, however, the authors could relieve our minds on this matter also.

Of course it must always be borne in mind that any solution of the problems before us must give financial considerations due weight, and it will be seen that my own efforts in this direction do not err in this way; also that the practical application of the agents to be used must present no serious difficulty. In this connection the authors' reference to a saturated solution of lime seems to ignore the fact that such would be costly to maintain (at 1 solution to 1 of sand—about 1d. per ton of sand), and in practice would be partly neutralised in the mine by acid waters, and on exposed faces by the mine air, which the chemist to the Mines Department accuses of frequently containing too much CO_2. I am sure, however, he would prefer that gas to HCN. Again I do not labour this point as the authors have given no indication of the practical application of their remedy, which, however, we await in their reply. The one thing their paper has made abundantly clear is the fact that the problem is not to be solved by mere examination of the literature on the question, and their experience that applications of known principles in new directions involve independent research is painfully well known to many of us. The paper is very sound on the theoretical point of view, though no account of any effect from zinc present is taken ; and the ingenious ionic explanations are so reasonable as to offer no temptation for a more critical examination, but I hope the discussion on the points raised will be worthy of so valuable a paper.

My own view of the "destruction of cyanide" necessary for the present purpose is that it involves the unlinking of the bonds between C and N. In that case such intermediate substances as formic, oxalic, glycolic or other organic compounds of C.H & O and amonia will be formed, or ultimately carbon dioxide and nitrogen gas. A practical consideration is the fact that the reactions proposed must take place in the cold with sufficient rapidity. No rapid destruction of iron, wood or vegetable fibre can be permitted, and some advantage may be gained if the gold in solution is not precipitated but is recoverable from a possible wash. It may not be necessary to destroy ferrocyanides, it may be advantageous to destroy sulphocyanides, and it is certainly necessary to destroy practically all the simple cyanide present. Old dumps (over six months) are possibly safe enough, and in any case I do not propose to consider them, for as a reduction manager I propose to let the mine sluice away my residues from the vats, under mining cost, and if the cyanide works bear the cost of destruction of cyanide in return, the nett result will be a reduction in cost of gold extraction, with which I am more immediately concerned. Conditions will vary much on different mines, so that it is difficult to estimate the amounts of cyanide and other salts to be dealt with. Upon the May Consolidated, a ton of residue sand will be associated with about 240 lbs. of solution of a "total cyanide" value of ·02 to ·03%, alkali equal to ·01% NaOH, and insignificant quantities of ferrocyanide and sulphocyanide. On some mines, especially if no final washes of "slime" solution, or of water, are used, the ferrocyanide and the sulphocyanide may each amount to as much as half the cyanide strength. It is evident that each ton of sand contains enough poison to kill more than 100 men.

I will first of all consider those reagents which promise success, and then outline the arrangements I would propose (if trials on a large scale prove satisfactory) for their economic utilisation. The only reagents I have been able to obtain satisfactory results from are, in the order of their efficiency (1) Alkaline permanganates (2) Alkaline picrates (3) Aldehydes and Ketones. The cost in these three cases are all in the neighbourhood of one penny per ton of sand for chemicals only, but it is just possible that extra gold may be recovered in the course of their application to pay for this in some cases, and in any case I feel convinced that a reduction of cyanide used will be induced indirectly, so that ultimately no permanent increase of working cost is to be anticipated.

ALKALINE PERMANGANATES.

It is a well-known fact that permanganate of potash in acid solution has little or no action on HCN, but it is not so well known that in slightly or strongly alkaline solutions the simple cyanides are oxidised to cyanates, the ferrocyanides to ferricyanides, and the sulphocyanates to sulphates and cyanates (in acid solutions HCN only is formed). The reactions I have established, to my own satisfaction at least, are as follow :—

In slightly alkaline solution.

$$3KCNS + 8KMnO_4 = 3K_2SO_4 + 3KCNO + 8MnO_2 + K_2O \qquad (1)$$

(A reaction requiring 93·8ccN/10 permanganate gave as a result 98·0cc and 4.8cc N/10 acid against 4·7cc).
(Reaction complete at 15 C in a few hours).
(No trace of KCN in filtrate).

$$2KMnO_4 + 2KCN = MnO_3,\ MnO_2 + 2KCNO + K_2O \qquad (A)$$

(A reaction showing theoretically 38·3cc N/10 permanganate and 7·7ccN/10 alkali formed, required 40·9cc permanganate to destroy KCN
43·5 do. to show red colour
and 7·9ccN/10 acid.)

$$KMnO_4 + 3K_4Fe(CN)_6 = 3K_3Fe(CN)_6 + 2K_2O + MnO_2 \qquad (2)$$

(No further action takes place in the 24 hours than the formation of ferricyanide with a brown precipitation of manganese).

In addition to that caused by the above reactions, some further permanganate is consumed by the cyanide solution last drained from a residue. This is due to some small extent to organic complexes present, and partly to the breaking down, of the KCNO formed, to ammonia and potassium bicarbonate, the former being attacked slowly by alkaline permanganate.

$$KCNO + 2H_2O = KHCO_3 + NH_3 \qquad (3)$$

(The formation of carbonate can be demonstrated with calcium nitrate ; barium nitrate also gives precipitate soluble in HNO_3 but this might be due to cyanates).

$$2NH_3 + 2KMnO_4 = 2MnO_2 + 2H_2O + 2KHO + N_2 \qquad (4)$$

(In solution of ammonia a few drops of $KMnO_4$ slowly change colour to brown precipitate).

This hydrolysis develops further alkalinity and though the hydrated manganese precipitate in equation (A) cannot be washed free from potassium hydrate, yet an excess of alkalinity is found. In one case in accordance with equation (1) alkalinity in excess was found up to a total of 6·5ccN/10 HCl though the precipitate is difficult to wash, and the methylorange end point was rather indefinite.

In acid solution the hydrolysis of the cyanate is apparently much more rapid and so is the action of permanganate upon the ammonia salt formed, and upon such products of the decomposition of cyanide already found in the liquid. If therefore an excess of permanganate be added to a working cyanide solution and after a couple of hours the mixture be filtered, and the filtrate made strongly acid, a further decomposition will be observed. It is quite certain therefore that no harm is going to be done by small excess of permanganate if this method of destroying cyanide be adopted before using "current" residues for sand-filling.

In acid solutions the action of permangantes upon cyanides is insignificant, but an alkalinity of 0·10% NaHO (and this would generally be present) is sufficient in the dilute solutions we are considering to enable the reaction to proceed to the end so that KCN cannot be detected by the prussian blue reaction, nor by Phine (sensitive to 1 part in 400,000, I find) nor even by hydrocœrulignone. These latter tests can be applied because the cyanide is destroyed before the action on permanganates (of other bodies present) is complete. In one case the total destruction of $KMnO_4$ was 44 cc. but KCN had all disappeared when 41 cc. only were used and the filtrate gave no reaction to these highly sensitive indicators. The working solution used for experimenting (last drainings) contained protective alkalinity 0·010% NaHO, available cyanide 0·010 KCN, and total cyanide 0·029% KCN. A mere trace of sulphocyanide and no ferrocyanide was present. This required about 20% by volume of N/10 $KMnO_4$, but if 40% were added the colour in the filtrate disappeared upon the addition of dilute acid in excess if sufficient time were given. (The presence of zinc has no effect at all.)

On the other hand a synthetic solution requiring 40·9 cc. for complete cyanide destruction consumed 44 cc. N/10 permanganate, while alkaline and upon addition of H_2SO_4 to the filtrate therefrom consumed a further 7 or 8 cc., though this latter figure would be increased by further time allowance. The total destruction of ammonia formed from KCNO would require 9 or 10 cc., a total of 61 cc. in accordance with equation (B) below.

$$6KMnO_4 + 10KCNO + 14H_2SO_4 = 8K_2SO_4 + 6MnSO_4 + 5N_2 + 10CO_2 + 14H_2O \quad (B)$$

(In this case there is no precipitate formed, and minute bubbles of gas arise.)

If a very strongly alkaline solution be used the green manganate, always first formed, is permanent, and as this is the least advantageous case much more permanganate is consumed. A very large excess of alkali, as is well known (Roscoe & Schorlemmer, Vol. II. part 2, p. 19), is required to make the manganates stable in solution and I do not think this case a likely one to arise in practice. The following is the equation:—

$$2KMnO_4 + KCN + K_2O = 2K_2MnO_4 + KCNO \quad (C)$$

(A solution requiring 76·6 cc. N/10 $KMnO_4$ (theoretical) showed permanganate bands to spectroscope at 68·2 cc. at once, but 76·6 cc. showed only very faint bands after 16 hours. A slight brown precipitate formed, and the solution turned red upon addition of excess acid.)

Although the above reaction commences with great speed the end point is very slow in arriving. A similar effect may be observed with sulphocyanides in strongly alkaline solution. The extraordinary stability of $HAu(CN)_2$ referred to by Messrs. Caldecott & Johnson (Vol. IV., p. 271) is no doubt the reason that it is quite unattacked by permanganate. A solution carrying 3 dwt. gold per ton gave no gold in the precipitate with excess of permanganate, after thorough washing, and the same is the case with formaldehyde and picrates, also ferrous sulphate. As the gold in the solution associated with discharged residues amounts to about 1 part in ten millions, this effect, though possibly enabling a small further saving of gold to be made, will not otherwise be worth consideration.

In accordance with equations (A), 1 and 2 the following permanganate consumption is indicated:—

1 lb. KCN	requires	2·425 lb. of $KMnO_4$
,, KCNS	,,	4·340 ,,
,, $K_4Fe(CN)_6$	,,	0·143 ,,

Assuming that one ton of residue is associated with 200 lb. of solution carrying 0·025% total cyanide, with insignificant amounts of the other salts, the above figures would show a consumption of 0·1212 lb. of permanganate, value about 0·6d. It would not be necessary to use pure permanganate of potassium, as a cheaper base, such as soda or magnesia, would show advantage in original or transport cost.

I may add that it is not easy, because of the coloured precipitate, to follow these reactions very closely, and the only method I have found useful is to place spots on glass and examine the colour transmitted through a direct vision spectroscope. In this way it may be shown that half an hour to two hours is required for equations (A) 1 and (2). It requires another 12 hours before the precipitate can be filtered off to enable the particles to grow from the size of colloidal suspension.

It will be observed that the final effect is complete destruction of KCN with formation of CO_2 and NH_3 or N_2. The intermediate KCNO slowly decomposes while the solution remains alkaline and speedily when acid, and can cause no danger in either case. To test this action of KCNO I attempted to prepare some cyanate, but the product I obtained showed only 6% NaCNO to silver nitrate and about 38% of Na_2CO_3. In alcoholic solution it gave a very good reaction with cobalt nitrate, but dilution with water speedily destroyed the fine blue colour. A small quantity recrystallised from boiling alcohol showed no carbonate and refused the prussian blue test. The impure salt, though quite free from NaCN, gave the prussian blue reaction in consequence of reduction by ferrous sulphate, and gave a good colour to "phine" after this reduction (that is in the filtrate from excess NaHO) though *directly* no trace of action was visible. This explains one case where I found a trace of green in the prussian blue test after application of small excess of $KMnO_4$ to a working solution. The amount was quite insignificant, but as the work was done on a cold day, I presume some cyanate (in the suitable modification) remained. In this same working solution mixture, as in all other cases tried with excess of permanganate (after 16 hours), no trace of reaction on "phine" paper could be observed when the prepared paper was suspended over the strongly acidified solution in a closed vessel. If such a mixture be tested within five minutes of adding the excess permanganate, a faint action can be observed in ten minutes more but if the test be delayed two hours no HCN can be observed on rendering acid—the alkaline solution gives off no HCN when mixed with permanganate in excess, though a fair reaction is observed with the weak working solution by itself, much intensified by addition of slight excess of acid. The impure NaCNO could not be made to give off any HCN gas and hydrolysed quickly in acid solution (equation 3).

I find that some working solutions are quite readily titrated direct with $KMnO_4$ as a precipitate forms much more rapidly. In one case I obtained distinct reddening with 28cc and the addition of 2cc more left a red colour in the filtrate after 16 hours. It is probable, therefore, that in practice the last solution leaving a vat could be titrated with N/10 $KMnO_4$ and by adding 10 or 20% to the figures thus arrived at, the amount of permanganate to be added, for a final wash before discharge to the mine, could be estimated. I may just add the observation that the addition of slight excess of $KMnO_4$ to 0·10% KCN at once and completely destroys its power of dissolving gold leaf.

ALDEHYDES AND KETONES.

Aldehydes have the structure R—CHO and ketones R—CO—R, and it is well known that both have the property of combining directly with HCN. In many cases this requires prolonged heating, but benzaldehyde will slowly combine with freshly liberated HCN, acetaldehyde will slowly react with KCN, and in the case of formaldehyde the reaction is almost instantaneous in the cold.

Formaldehyde is obtained commercially as a strong solution (about 40% by volume under the name of formalin, and is largely used as a safe and powerful disinfectant. This solution is heavier than water as the following table (from Allen's Organic Analysis) shows:—

Sp. Gr.	by weight.	by volume.
1·0025	1·0	1·0
1·0125	5·0	5·0
1·0250	10·0	10·25
1·0380	15·0	15·6
1·0530	20·0	21·1
1·0670	25·0	26·7
1·0830	30·0	32·5
1·1040	35·0	38·6
1·1250	40·0	45·0

All the following results refer to formaldehyde from commercial formalin of sp. gr. 1·09, about 32% by weight, and would require suitable correction if a stronger product were used. It is possible that other aldehydes or even ketones may be useful for our purpose (for instance, acetaldehyde, which is certainly very much slower, or benzaldehyde), but for preliminary experiments formaldehyde is much more suitable, and even in practice kindred reagents would have to show considerable less cost to render them worthy of attention. (Some forms of sugar have an aldehyde structure, but I understand that the cyanhydrin reaction (equation D) is slow and requires heat.)

The following is the reaction between cyanide of potassium and formaldehyde (Remsen, Organic Chemistry, p. 198):

$$2KCN + 2HCHO + H_2O = K_2O + 2HCH(OH)CN \text{ (cyanhydrin)} \quad (D)$$

$$2HCH(OH)CN + K_2O + 3H_2O = 2CH_2OH.COOK + 2NH_3 \quad (E)$$

(Potassium glycollate)

(In accordance with (D) no extra alkalinity is developed. Formalin estimated by weight at 32% was found to be 36% by titrating back excess of acid $AgNO_3$ with NH_4CNS).

No effect can be observed upon sulphocyanides or ferrocyanides but free ammonia will be destroyed in accordance with equation:

(From Sutton's Volumetric Analysis, p. 392.)

$$6HCHO + 4NH_3 = (CH_2)_6N_4 + 6H_2O \quad (5)$$

(Hexamethylene tetramine).

The above reaction is a very slow one and requires from 6 to 24 hours, so that it will probably play very little part in the consumption of formalin required.

In accordance with equations (D) and (E), no further alkalinity than that due to KCN is developed, and the following experiments indicate this :—

50cc 0·01% KCN required 5·3cc N/10 acid (phene)
7·6 : : (meth. orange)

If excess of formalin be added immediately before titration 7·5cc and 7·6cc of acid are required in the two cases: the HCN effect referred to by Dr. Moir and Mr. Gray being thus obliterated. If formalin be added to a working solution containing zinc a precipitate of zinc hydrate is formed containing no cyanogen, but silver cyanide is decomposed only in alkaline solution.

I have found it no easy matter to follow these equations with exactness by any direct titrations, because of the difficulty of finding an indicator for the end point. I have tried leucaniline and aniline hydrochloride, but the end point is not definite, and both require some little excess of cyanide to prevent the formaldehyde reaction. Leucaniline in acid solution is much slower in action when the reagents are balanced, but up to nearly twice the right amount of KCN a very slow formation of the violet colour is observed, in this case requiring 24 hours. The white precipitate with aniline seems to be formed at almost any proportion if you render alkaline and wait long enough. Unfortunately, the reaction with hydrocoeruligone is also very indefinite, but in this case a small excess of KCN is not decidedly indicated. With "phine" solution the reaction takes place except with very large excess of formalin, though it is much slower as soon as an excess is present at all, and the best direct result obtained was by spotting on phine paper which agreed closely with the exact method given below. This method is taken from Sutton's Volumetric Analysis, and depends upon the fact that an excess of KCN, after addition of formalin, is completely precipitated by acid $AgNO_3$ without reduction of silver. The excess of acid $AgNO_3$ is titrated with NH_4CNS by Volhard's method. A solution of formalin (estimated at 32% from sp. gr.) was made so that 1cc should equal 10cc of 0·1% KCN. This solution was found to show 1cc equal to 11·2 c.c. KCN. A working solution gave 1cc equal to 9·8 cc KCN 0·10%. Another gave 10·6cc. These working solutions evidently contained ammonia or other destroyer of formaldehyde in very small quantities.

If an attempt is made to use acid $AgNO_3$ to determine the end point of a formalin titration with KCN, it is found that an excess of about 2·6cc per 100 of 0·10% KCN is required to prevent the precipitate of AgCN, and in any case this method would be useless for working solutions containing chlorides, etc., and therefore the round-about method detailed above is at present the only accurate method.

I think the whole difficulty depends upon the fact that this reaction does not proceed to absolute completion, at any rate within two hours. From rough comparative experiments with the Prussian blue reaction I find that about 99·5% of the cyanide is destroyed immediately, and that the rest requires about 12 to 16 hours. In this way can be explained the fact that no HCN can be found to "phine" paper on adding acid to the formalin mixture balanced with KCN (0.10%) after 16 hours, no reaction with the Prussian blue test, very slow reaction (but complete) with the "phine" solution, and no solvent action on gold leaf. If the above reactions are tried within two hours it will be found that a slight amount of HCN gas is given off with acid, a very faint reaction with Prussian blue test, a good reaction with phine solution and a slow action on gold leaf (dissolves in over two hours). A slight isopurpuric reaction can also be obtained within two hours. Using 10% KCN a faint reaction to acid, with "phine" paper, can be observed even after 24 hours.

I have no means for proving the (E) equation in the cold but by using strong solutions a good blue colour can be obtained with $CuSO_4$, which may be due to ammonia, and a red solution is obtained with slightly alkaline $FeCl_3$ which may be due to the glycollic acid, but if a practical application is found possible no doubt it will be worth while going further into the matter. At the present stage I am uncertain whether this conversion of cyanide can be called a destruction, but in any case the evolution of HCN upon addition of mine water would apparently be insignificant after the 16 hour delay, taken up by application as a wash to the vats. The above equation (D) indicates that 1 lb. of KCN would require 1·44 lb. of the 32% formalin and on the same assumptions as before one ton of sand would require 0·075 lb. formalin; at 1s. per lb., in Johannesburg this would cost 0·9d. per ton. A small extra consumption may be required from other constituents of the solution, but this would in most cases be very small.

Alkaline Picrates.—Picric acid or tri-nitrophenol was formerly largely used for dyeing purposes, but its principal use at present is in the manufacture of lyddite and melinite for war and other explosives. The reaction with alkaline cyanides in an alkaline solution is fairly well known and is the base of a fairly sensitive test for either reagent. The reaction is not very quick in the cold but may be accelerated by the application of heat. At ordinary temperatures the reaction between sodium picrate in 1% solution and potassium cyanide in 0·1% solution is complete in about two hours, and a very deep brown solution of sodium isopurpurate is formed in the liquid. The following is usually given as the reaction :—

$$2H_2O + \underset{\text{(Picric acid)}}{C_6H_3N_3O_7} + 3KCN = \underset{\text{(Potassium Isopurpurate).}}{C_8H_4KN_5O_6} + NH_3 + K_2CO_3 \qquad (F)$$

(The colour developed prevents any accurate titration but 39 c.c. KCN 0·10% were found against 40·3 theoretical excess and alkali to lacmoid paper 1·5 c.c. (N/10) against 2·6 c.c. Experiments with 1 to 7 c.c. of N/20 picric acid, below, support above equation.)

Picric acid has no apparent action on sulphocyanides and ferro-cyanides.

The reaction (F) requires a neutral or alkaline solution ; even excess of acidity due to picric itself will entirely prevent it as the following experiments show : Seven lots, 10 c.c. each of 0.09% KCN were placed in test tubes and varying amounts from 1 to 7 c.c. of 1% picric acid solution were added. In 12 hours 1 c.c. and 2 c.c. gave equally dark solutions. Three c.c. picric acid showed some action and 4 c.c. a very faint effect. Absolutely no change was observable with 5, 6, and 7 c.c. though on the addition of alkali in excess the usual reaction proceeded forthwith. A shade less than 1 c.c. of picric acid solution is indicated by the equation. The characteristic silky green precipitate which is produced by a weak solution of cuprammonium sulphate could only be observed with 3 c.c. This test though exceedingly sensitive with pure picric acid is much obscured in the presence of iso-purpurate and is therefore not available for observing the exact end point of this reaction. Although it will be observed from equation F that 1/3 of the KCN is completely destroyed the fate of the remainder has to be considered. Though iso purpurates seem to be quite stable in alkaline solution and even with very weak acid solutions this is not the case with such strong acid as 20% sulphuric acid. The same, however, may be said of ferro and ferricyanides as the following series of experiments will show.

Test paper made with $CuSO_4$ and phine was suspended in large closed test tubes over the following mixtures :

(1). Excess alkaline picrate and 0·10% KCN.
(No colouration in 24 hours).
(2). To (1) excess 20% H_2SO_4 added, solution became lighter.
(Strong colouration in 5 minutes.
(3). 5% Ferrocyanide and excess 20% H_2SO_4.
(Fair colouration in 24 hours.
(4). 10% Ferricyanide and excess 20% H_2SO_4.
(Fair colouration in 24 hours).

Experiments 3 and 4 were repeated with excess of suitable iron salts to form Prussian blue, but this did not appear to effect the result either way. As this matter touches closely upon the work done by Dr. Moir and Mr. Gray I went further into the question and took for experiment a working solution giving 0·018% free and 0·057% total cyanide. This solution alone gave a faint reaction in closed test tube with "phine" paper in one hour. With excess H_2SO_4 strong reaction in 5 minutes. With slight excess $FeSO_4$ and acid strong reaction in 5 minutes. With large excess $FeSO_4$ alkali, then slight excess dilute acid strong reaction in 5 minutes. Of course, there is no comparison between the stability of the ferrocyanides and the isopurpurates as tested in this way as the former are much superior. As no attempt was made to carry out the exact method recommended by the authors this has only a very indirect bearing on their work. I shall later outline a method for reducing the ferrocyanides in our working solutions to a minimum, because their complete destruction does not appear feasible. It is a well-known fact that old solutions of potassium ferrocyanide exposed to light develop alkalinity and also give a light blue precipitate (observed in 3 above). But such solutions also give Dr. Moir's reaction with hydrocoerulignone though this is not so reliable in such a case. However, in such cases as these two, a practical test will decide if the reactions can be safely made use of in practice. I may say that this reaction (F) between alkaline picrates and cyanides is very delicate and will reveal the presence of one part of KCN in 100,000 with certainity, and a careful comparison will show the effect of 1 in 200,000 after allowing to stand for 24 hours, and observing the colour of 6 inches of solution.

It has been pointed out that a concentrated solution of picric acid if boiled for some time with caustic baryta yields a little HCN ; but no reaction with $FeSO_4$, NaHO then acid can be obtained with a cold saturated solution either of picric acid or its reduction product with zinc or sodium sulphite (picramic or picranic acid). If slight excess of picric acid be added to KCN in alkaline solution after 12 hours no Prussian blue reaction can be obtained, even if a little weak HCl (in excess) be added first of all. After one hour only, a strong Prussian blue reaction can be obtained by first adding strong HCl to the mixture, but after 16 hours this reaction is no longer possible, as upon the addition of the necessary alkali the isopurpurate is reformed immediately. (Cp. (2) above).

In accordance with equation (F) 1 lb. of KCN requires 1·03 lb. picric acid (quoted at 10d. lb., London). With the same assumptions as before one ton of sand would require 0·0515 lbs. about 0·77d. here. Upon continued investigation I find that a solution of 0·10% KCN, with excess of alkaline picrate will dissolve gold leaf in half an hour, either when freshly made, when two hours, or even when 24 hours old. This solution will give HCN gas when acidified but will show no HCN (at 24 hours old) to Prussian blue test. The obvious deduction is that isopurpurates are gold solvents but the point is not worth investigation in view of the expense and other difficulties attending their use. The picrates will not dissolve gold leaf in 1·0% solution. It seems fairly certain that in practice strongly alkaline solutions would have to be used to successfully "convert" cyanides, and probably a practical experiment might be left over till the other methods proposed had been found unavailing.

Practical Methods Suggested.—I have little wish to enter upon the mechanical side of this question, but I assume that, if current residues are to be utilized for sand-filling, they will be sluiced out of the tanks and the surplus water, after removal by cones with Caldecott diaphragms, run into a sump. As the sand has to be finally placed some distance below the ground, I should suggest that the boreholes proposed or any other method might easily be arranged to avoid pumping of sand pulp.

It is, however, with the chemical side of this question that I have concerned myself in order to arrive at the most economical methods practicable. In the first place it is clear that for this purpose we want as little cyanide as possible left in the sand, probably as little sulphocyanide and possibly as little ferrocyanide. Fortunately, in in most cases, these results can be simultaneously achieved by the following method outlined.

In the first place to minimise as much as possible the production of ferro and sulphocyanides it will be necessary to crush the ore in alkaline solution. This is almost invariably done now, but if the alkalinity of the mill water were maintained in the neighbourhood of 0·025% as NaHO little extra lime would be required, a small improvement in amalgamation would result, and ferrous salts would be precipitated and oxidised immediately (this *Journal* Vol. III. p 3. Mr. A. F. Crosse). Enough alkalinity would be present to ensure against any evil effect from decomposition of pyrite in collected charges awaiting treatment. Further, the practice on some mines of using cyanide solution so strong as even 0·15% available cyanide would in this case become without excuse and at the highest 0·10% would be amply sufficient. Indeed I see no reason why anything higher than 0·05% should go on the vats seeing that total cyanide may be nearly twice this, and if the precipitation declined the cyanide necessary to improve it should be added where required—*at the head of the boxes.* In addition to all this I would suggest that the last two washes should consist of precipitated solution from the slime plant, which should carry at the most 0.008% available cyanide as that strength is sufficient to ensure perfect precipitation with solution warmed to at least 19° C if these boxes are dressed about once a week. The interchange of solution would provide all the cyanide necessary for the slime plant at least that is my experience. In every case the final wash from the ordinary treatment should be completed as far as practicable by application of the vacuum pump. By the above means we should ensure that the least possible amount of cyanicide consumers remained to be dealt with.

Application of Cyanicides.—I assume that the finished residue is associated, per ton of sand, with 200 lb of solution and that this will not contain more than 0·025% of total cyanide (as titrated in presence of excess of caustic alkali) nor any other substance in sufficient amount to affect the calculation. Of course it will be necessary to determine these facts in every case and the calculations will have to be adjusted in accordance with the result. I suggest that to the finished residue, whichever is found by practical trial of the following solutions to be the most suitable, should be applied :—For every 100 tons 20 tons of (*a*) permanganate equal to about 0·045% $KMnO_4$ or (*b*) formalin equal to about 0·027% formalin (at 32% formaldehyde); or (*c*) alkaline picrate equal to about 0·019% picric acid, made by dissolving picric acid in lime water so as to show excess of alkalinity to phene. The two solids must not be allowed in contact or explosions may ensue. A margin of 50% in excess is used, which may not be necessary in practice. These washes may be run to the discharge sump direct if it is not found profitable to send them through zinc boxes or apply other methods (such as de Wilde's) for recovering the gold *en route.* I would recommend a special pump and pipe service to prevent mistakes or mixing of solutions and, of course, a sump for discharge water would be provided and suitable means for hosing out the vats if the engineers recommend that method. At present I have no reasons for believing that the methods I have outlined would result in anything but a decrease of total reduction expenses and as far as unavoidable facts are concerned the direct expense of the cyanicides mentioned should not exceed (*a*) for permanganate 0·6d. to 0·9d per ton : or for (*b*) formalin 0.9d. to 1·3d. per ton : or (*c*) picric acid 0.77d. to 1·14d. in practice if permanganate be used four hours contact must be given and the solutions leaving the vats must be distinctly red. It is possible that some trouble in leaching may be caused and in that case a weaker solution would have to be used for discharging and the sand could not be mixed with old dump residues until 24 hours at least had elapsed. If formalin be used the solution leaching must show excess of formaldehyde and give practically no Prussian blue or isopurpuric reactions. If alkaline picrate be used four hours contact must be given and leaching solution, which will be a brownish red colour must show no reaction with the Prussian blue test.

As far as laboratory tests go it seems clear that the use of alkaline permanganate must be attended with complete success, but the method of application must be decided on a working scale. The other two reagents have the defect of not destroying sulphocyanide, but practical experience may decide that this is not necessary either because both sulphocyanide and ferrocyanide may be present in insignificant quantities, or because they may prove sufficiently stable under the working conditions. If the latter is the case the use of ferrous sulphate, provided the reaction is complete, as proposed by Dr. Moir and Mr. Gray may become possible and should be cheap enough if an excessive amount of lime is not found necessary. Of all the methods proposed the use of formalin would be by far the most convenient from the rapidity of the action, from the absence of any large amount of precipitate, and because of the general cleanness of the reaction.

Mr. Andrew F. Crosse (*Past-President*): I should like to ask Mr. White an important question: in your experiments did you treat the solution after passing the sand, or did you treat a mixture of sand and solution?

Mr. H. A. White: In some cases, yes.

Mr. A. F. Crosse: You have not mixed it with the sand itself?

Mr. H. A. White: No.

Mr. A. F. Crosse: Because it is a very different problem using an oxidising agent on a very dilute cyanide solution by itself, to using the same agent in presence of sand containing pyrites, and in practice this would be the case.

Mr. H. A. White: A very dilute solution would be used as a wash, and there would be practically no action on pyrite, though filter cloths may suffer a little. I have put this matter before the Mines Trials Committee for a practical trial as it is of very little use making such experiments in a laboratory.

Mr. C. D. Leslie (*Member*): It may have occurred to many of us that the valuable papers read this evening by Mr. Pam and Dr. Moir must bring together the chemist and the miner on a subject of common interest, and tend to make our mining members realise what our chemists can do for them. Since sand filling is in its infancy here, and also because of the important bearing its successful application would have on present and future Rand mining practice, it is to be hoped that in discussion members will add to the value of the papers that have been read, by bringing forward new points in support of the subject, and refrain somewhat from recording minor objections until our knowledge of the whole question of sand filling has reached a more advanced stage.

The President: We have so far had a very interesting meeting with the two papers which fit in so well with each other, and I am sure the remarks offered are of practical use both for the miner himself and his relations with the chemist. It is a great pleasure to us to have two leaders of our mining industry, Mr. Anderson and Mr. Leslie, with us to-night. I hope this will frequently be the case in the year before us.

The vote of thanks was carried unanimously.

FURTHER NOTES ON RAND MINING.

(*Read at February Meeting, 1910.*)

By Tom Johnson (Member).

REPLY TO DISCUSSION.

Mr. Tom Johnson (*Member*): In reply to Mr. Roake, I must say that I am not in love with the idea of boys alone building pigsties or packwalls; it was seeing the fruits of this kind of thing that caused me to write the paper. It is a fallacy to think it can be done well by inefficient boys weeded from the others, that is the way to make man traps. My way in this work, as in other work, is to combine the labour of the white and coloured. I always found it to pay to keep a particular white man or men on this work, supplying sufficient native labour to keep the white man walling steadily, or building the cribs. It is much cheaper to do this than to have the work done in the manner I have seen it done by putting inefficient labour on the job—matters of safety require efficient labour.

As to managers agreeing to a policy of more machines in a stope, I am bold enough to prophecy that they will have to. As to the objections Mr. Roake brings forward about the number of machines being limited, etc., if the rock comes from one face it is not needed for another; grade will be regulated in the same way as now; of course a better system of handling rock would be necessary, but this would be a gain

Mr. Roake seems to imply that by working home we should lose some reef, that we should not lose if we worked from home, but I am afraid I cannot see it, for it seems that the drive would have to be kept open, or if closed, re-opened in either case. With drives under the reef a boxhole could be put up at the nearest point to the block left, even if the stopes were caved in. If the suggestions in the paper are followed the drives will not break up and close owing to the packs, sand filling, etc.

Mr. Richardson points out a slip right away. In the paragraph dealing with "Driving under the Reef," I should have mentioned that I would drive my first drive on the reef for a trial.

As to getting values, I have seen a great deal of development to the stoping after it, and nearly always found an appreciable difference between the development and stoping values. Also, if a drive shows poor values it is still driven on in the hopes of getting into better rock, so I do not think that a great loss would result from not driving on the reef. Nevertheless, it must be understood that I am not laying down hard and fast rules that are to be blindly followed; there are bound to be cases requiring divergence from general lines. It must also be understood that even with a drive driven on lines it is not necessarily straight from end to end as seems to be assumed. There are miles of drives on the reef which have any amount of curves in them, but which could have been driven straight from end to end without getting off the reef. For a beginning, take a case of this sort: after the first trial drives have gone in the next ones could go under the reef, and if the cross-cuts and boxholes turned out satisfactory evidence, then the next drive could also go under the reef. If the cross-cuts, etc., did not give out good evidence then there is nothing to prevent one from turning on to the reef. Where there are faults or bends in the reef to be dealt with, the drives could be driven straight for a certain distance and then the direction altered to suit the new circumstances. We have to do it at present, but the developer determines the number and extent of our curves now, which I do not think satisfactory. There is no necessity to get a great distance from the reef to try and put the drive in straight from end to end. I can assure Mr. Richardson that Fig. I. is a true reflection of a few drives I have seen lately, and there is no thought of taking out the men.

As to placing reliance on our strong quartzite roof, we shall have to! It is no use blinding ourselves to the fact that because we have done fairly well with our small pillars and small areas, that it will always be so. What is going to happen bye-and-bye, say, along the stretch of ground from the Rose Deep on the east to the New Goch on the west, unless sand filling is commenced soon? It will not be long before the whole of it is standing on the pillars, which are sure to go at the main reef leader. Now one can see this coming, and the same thing will happen in the deeper deeps, but with worse effects. For awhile there are three ways of handling the problem: (1) leaving larger and more pillars, which will be very expensive; (2) to take out all reef and pack, sand filling for preference, as stated in the paper; or (3) leave a large number of very small pillars, 3 ft. or 4 ft. diameter, to carry the immediate hanging until the face is advanced, these stumps then being taken out before sand filling. There is a great reliance to be placed on the elasticity of our hard roof. Who has not seen props that have been set to carry the roof broken, yet the roof never came down? You may also go along drives where the supporting ribs have been crushed out, and the roof only comes so far. What has happened in these cases to stay the movement of the roof? We know the roof has lowered some distance, but what holds it in its new position after the supports have got crushed away? Also, what about the folds in the Rose Deep and the Simmer, and what about the changes in dip? I think there is plenty of evidence as to the elasticity of our roofs; it only remains for us to take advantage of it, and allow it to help us instead of doing us harm; it only needs a little pluck and the thing is proved.

Mr. Richardson speaks of the softer strata overlying the coal measures. Now, it is a fallacy to think that all coal measures are soft; I have worked with a 40 yards thick rock roof which was much harder than our quartzites, and the drifts were worked out over greater areas than any stope I have seen on the Rand, without supports of any kind in them. He also thinks that because the coal faces are advanced rapidly for the purpose of roof control in its action in breaking down the coal, that this is of doubtful application here. Now it is "right here" where my friend should have asked himself the questions: why do they advance the face so quickly, and what would happen if the face was advanced slower or was stopped, and he would have come to the conclusion that the roof would break at the face, not only taking the weight off the coal but giving someone a job to open up the face again. It seems to me that the remedy the coal miner uses to stop his roof from breaking will be the same for us.

If a pigsty is likely to get blasted it requires four posts to stop it from getting scattered by blows on the ends of the pieces. Pigsties are not preferable to packs, there is too much "give" in them. They are good for where it is required to get supports up quickly or for close to the face where blasting is going on, and when kept to their proper sphere are mostly temporary.

Packs are far and away better for supporting and steadying the roof. I am speaking of the roof proper, not the immediate hanging wall, as seems to be assumed.

I do not know if I have administered a rude shock to anyone by saying that 500 ft. per month can be done in a drive, but considering that 360 ft. have been done in a single end, and the large footages that have been done in incline shafts, I am of the opinion that what I say can be done and will be done by those who will lay themselves out to do it.

Mr. Weston is still of the opinion that 8 or 10 machines cannot do good work on one face; well, we must agree to disagree on this. I have seen six benches a shift broken off half a stope face with good results, so I fancy I have a small reason for thinking 8 or 10 benches could be got from a full face. There is one thing that will prevent this 8 to 10 machines per face, and that is, to get around to that other suggestion of mine of one machine, one man, full shift's drilling, and two 4-hole benches per machine; something will have to be done to advance faces quicker, or expenses will go up without valid reasons. I cannot see that I have a new excuse for 8 to 10 machines. If I remember rightly, I said in answering his criticism of my "Notes on Mining," 1908, that, "I believe the time will come when it will be necessary to use large numbers of machines on each face, as the faces will have to advance much quicker than now, or they will not be kept open." Will Mr. Weston kindly point out where the new excuse comes in?

Speaking of the accident at the Cinderella Deep, if the pillars had not been left, but a few packs put in, I do not think the accident would have happened. There was no such thing as an air blast: rock pressure was the cause, and there was concussion of the air, due to the fall. We must start with the assumption that the rocks are flexible, also that they will "flow" or crumble if subjected to a pressure great enough; this pressure, of course, varies with the nature of the rock. In the Cinderella the pillars were probably large and did not crush up, but flaked as they would have done if they had been twice as large, owing to the nipping pressure on the edge of the pillar. This no doubt occurred at the solid faces also, but to a less extent, so as the pillars would not give way, either the roof or the floor or both places had to go, so as to give the relief necessary for the roof to support itself. Just as in the cases mentioned before, where timbers put in to support roof get broken still the roof does not fall.

In speaking on prop setting, I used the term "*perpendicular* to dip." I note Mr. Weston uses "*vertical* to dip," Which is right? As I understand it, vertical lines are gravity lines and therefore radial lines and cannot be parallel; perpendicular lines on the other hand, may be in any direction, for instance, two lines with an angle of 90° between them are perpendicular to each other.

Mr. Weston speaks of preaching to deaf ears. I look at it very much in the same light, but there is hope ahead. Are we not beginning to understand that it does not pay to mine waste? Other things will come in time if the preaching is only kept up. Difficulty occurs continually in any effort to point out remedies for some conditions that exist at present in our mining work, and a large part of the community show great unwillingness to admit that a change from present ways is desirable. Innovations are condemned without consideration of their merits, simply because they are innovations, whilst changes in old methods are resisted without reason or judgment. Still we shall "arrive."

I must thank Mr. Gascoyne for his contribution. He is right in thinking I like plenty of criticism, as I hold the opinion that however good a paper may be, it is not complete without a fair amount of criticism. What I have said about the condition of roofs and pillars is, to my personal knowledge, happening at the present time, so the remarks have to stand. I am one of those who think that there are cases where it would be better to take out the shaft pillars and pack instead of leaving the pillars in. I once had to tackle a mine where the shaft pillars had been so robbed that they and the shaft bricking were all "on the crush." I had the pillars taken out, packs put in, and where the brickwork was crushing, a band of brick work was taken out and re-built with wooden bricks, which took the squeeze splendidly. What I should do to keep the shafts out of danger at a depth of 10,000 ft. would depend greatly on circumstances. The incline portion I should put in the strongest ground I could find under the reef, and leave no pillars in the stoping (although a shaft would be fairly safe in the hanging if the whole of the reef was taken out and the places sandfilled, for then the whole roof would sink together without breaking up). I should keep away from any soft bed for that would certainly flow. It may possibly come to using small parallel shafts. As to the vertical shafts, I should leave no pillars about, taking out all reef and line around the shaft with timber logs or packs, sandfilling from behind as soon as it started to squeeze, which at that depth would be early, especially about the Main Reef Leader footwall. Of course, we are not certain what trouble we shall meet if ever we get to 10,000 ft. depth. There is a deal of experience to be got before then, and one can only assume what conditions will be and how to deal with them, but there is sure to be trouble with any soft bed met in depth, and round shafts would be best in my opinion.

The sketches of the coal drives would actually serve for drives here, as I should like to drive them, but it is hard work to get anyone to allow innovations. They must have boxes on the drives where they are an abomination. I showed my sketches with the roof cut, as I should want the cars to follow the breast face; they would therefore have to go under the roof, the roof and pack following the breast some distance back, not shown plainly in sketch.

There is one system of mining which might be suitable for those who do not like the idea of stoping straight ahead, and that is a kind of panel system or alternate stopes. In this, stoping would start in each alternate winze, and half of each block is taken out, then the stopes are cleaned out and filled with sand. In the meantime, stoping would commence in the other winzes and the blocks of ground be finished, these stopes would then be cleaned out and filled with sand. The first blocks left forms piers or pillars whilst the others are extracted, then the sand-filled area supports the roof whilst the remainder of the blocks are extracted. As Mr. Gascoyne points out, people are apt to take too narrow a view of this matter of handling roof. They have their ideas fixed on the immediate roof. They ought to think a little of that part which is out of sight.

Mr. Coombe starts off with a wrong assumption and consequently his remarks lose some of their force. I never mentioned anything about a combined heading or stope. What I intended, and what I thought I made plain, was that the drive should be driven 25 ft to 30 ft. wide on the reef, the whole face being advanced at once. It seems to me that most of my critics miss the point that breast driving is no different than a shaft of the same width, but with the advantage of being able to get the rock from the face much quicker that in any other shaft, allowing mucking and drilling to be carried on at the same time.

To hear people talk of the difficulty of shifting rock would almost make one believe that we had got to the end of our tether in this direction I do not believe so as far greater quantities than those mentioned have been cleared out of incline sinkings. We all know that at the present moment things are being done in our mines that were not thought possible a few years ago, and we have not reached finality even in clearing drives of rock. I have before now had the terms ridiculous, absurd, etc., applied to my suggestions, but still I find those same suggestions adopted as I shall find some of these about driving. If Mr. Coombe had as much knowledge as the writer, of what is and can be done in packing drives, etc., he would alter his opinion about the ridiculousness of the idea. As he knows, I always hold that we should be able to do, what others, with no better conditions than ourselves can do.

Above I have mentioned the word "flow" with regard to rock movement. Whether this term is correct or not I think it explains best what I mean. In collieries we have what is called "creep," with which I know several of our members are familiar. It is a gradual lifting of the floor, and in the main roads of some mines the floor is removed time and again and it still keeps lifting. Where does it come from? The pressure of the roof transmitted through the pillars to the floor is the cause, and I say the floor "flows" to the open spaces and rises there. Something of the sort happened at the Cinderella Deep causing the drive floor to rise. The Cinderella Deep is not the first mine on the Rand in which the drive floors have lifted. I have seen it twice here before, but a more gradual lift.

In conclusion, I must thank those gentlemen who have come forward with contributions. I had hoped there would have been more, but our mining members are very apathetic. There is too much of the idea that what is not done on the Rand in mining is not necessary. There are many things to be learnt from other parts, and much to be learnt from one another, even here on these goldfields, to our own and the industry's gain.

THE TUBE-MILL CIRCUIT AND CLASSIFICATION.

(*Read at February Meeting, 1909.*)

By G. O. SMART (Member of Council).

DISCUSSION.

Mr. J. E. Thomas (*Member of Council*): Mr. Smart's valuable paper on Tube Mill Circuit and Classification is of great importance at this time when the tendency of modern practice is to throw more work on the tube mills than has hitherto been the case.

The Simmer Deep-Jupiter joint plant has been equipped with similar cones to those used by the author at the Simmer and Jack Proprietary Mines, Ltd., and are run in practically the same manner as described by him, the only differences being in details of fittings, etc. Instead of using the cut-off gate illustrated in the paper, the writer prefers a nozzle fitted with liners and attached to the flange at the bottom of the cone. The lower part of this attachment is connected to the upper by means of a bayonet joint similar to that used

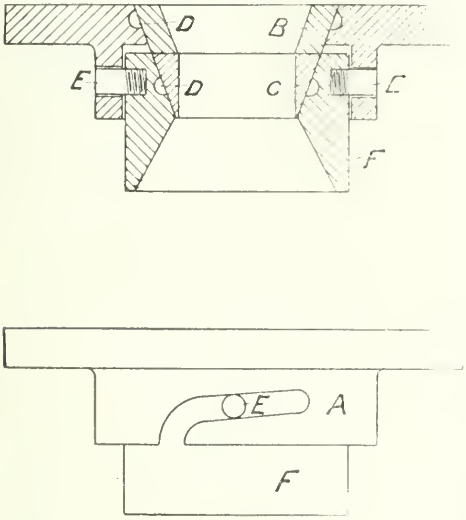


Fig. I.

A. Main Casting to Bolt to Cone. B. Upper Liner. C. Lower Liner. D. Recess for Packing. E. Pins or Bayonet Joint. F. Lower Casting.

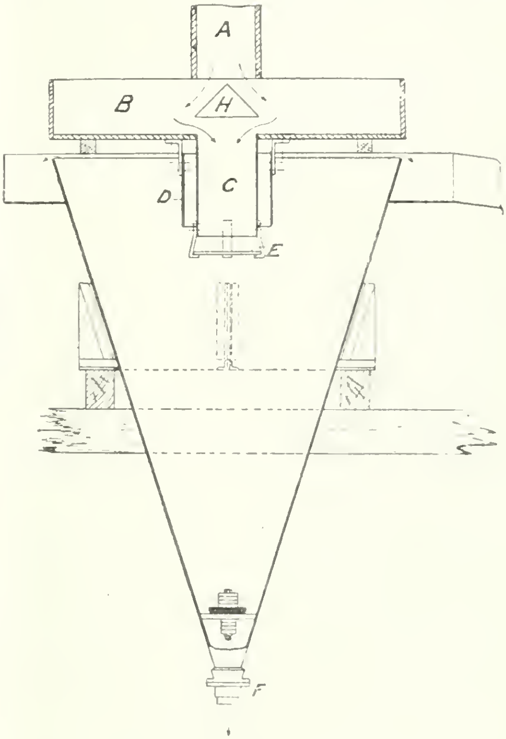
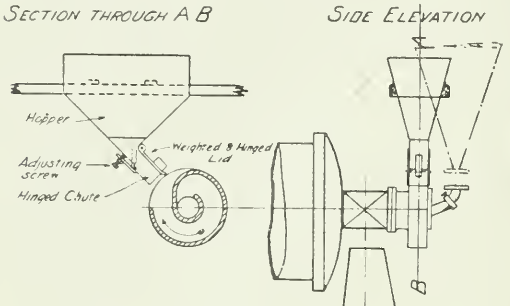


Fig. II.

A. Main Inflow Pipe or Launder. B. Launder carrying submerged Inflow Pipe and Air Sleeve. C. Submerged Inflow Pipe. D. Air Sleeve. E. Baffle. F. Bayonet-Jointed Nozzle. H. Distributing piece.

in electric fittings. The nozzle can be easily detached without stopping the underflow of pulp and fresh liners of the desired size slipped into place, the nozzle then being replaced without any difficulty. (Fig. I.) The size of liner to be used is determined by the foreman in charge of the plant, who can thus regulate affairs better than when the underflow is liable to be altered by the shiftsmen neglecting to see that the gate is set exactly as before, after occasion has arisen to open or shut it. Mr. Bayly's device also obviates irregularities arising from this cause. As pointed out by the author the distribution of the pulp, so that each cone does its fair share of work, is of the utmost importance. The device usually employed consists of a launder with iron gates to divert portion of the stream to each classifier. After a very short time indeed sand will find its way between the gate and the bottom of the launder, thus forming an unauthorised concentrator, and difficulty is experienced in getting the gate to shut tightly. An arrangement by which pulp is delivered to a central box, from which it may be distributed to each cone by means of separate pipes, will be found to give better results. The pulp should flow from the box into separate pockets, the inlets being at the same level, from which the pipes to the cones are led. The pockets allow of the pulp standing at various "heads" according to the amount of pipe friction, without interferring with the main distribution. Each cone of a set should be as nearly as possible under the same conditions as the others. Care should therefore be taken that the velocity of the inflowing pulp be even. This may be accomplished by running the pulp into a short launder over the cones and carrying the inflow pipe. The bottom of the launder should be only a few inches above the overflow level of the cone. A sleeve placed round the inflow pipe, about 16 in. to 18 in. larger in diameter and dipping into the pulp from 6 in. to 12 in., will collect most of the air bubbles formed at the inflow and allow the air to escape. These bubbles otherwise have a tendency to carry coarse particles over the periphery of the cone. (Fig. II.) The author attributes the excess of moisture over that required to fill the interstitial spaces in the overflow to filtering, but as this would be more rapid with coarse than fine material, it hardly accounts for the former carrying less moisture than the latter. In order to insure that as little coarse sand as possible overflows with the pulp going to the cyanide works, it might be advisable to instal return sand cones in much the same manner as those employed to prevent sand overflowing with the pulp to the slime plant. The maintenance of an adequate feed of the pebbles to each tube mill is a matter of very great importance, so that the pebble load may be kept up to its maximum, *i.e.*, a few inches above the centre of the mill when at rest. It has recently been demonstrated that the grinding efficiency falls off about 2 per cent. for every hour that no pebbles are fed in. In order to facilitate the feeding of pebbles, the writer has recently brought forward the device shown in Fig. III. for use in conjunction with Pryce's feeder. This consists of a hopper, with a chute at its lower extremity so arranged as to allow of pebbles, or pieces of ore, dropping into the inlet of the spiral feeder on its coming vertically below the chute. The latter is hinged so that in the event of a pebble jambing in the inlet of the spiral and coming round under the chute it will allow the pebble to pass without doing any damage. The chute is also provided with a cover working on a hinge. This serves to prevent pieces of ore, of the proper size to pass through the spiral, being carried over the periphery of the spiral feeder, and at the same time allows of pieces, which are too large to enter the spiral, being carried over and automatically thrown off on the side opposite the chute. The device is also suitable for use in conjunction with a conveyor belt from which the pebbles could be diverted in to the hopper by means of a plough The wear on the spiral feeder is considerably less than when fed by hand, and it could be still further reduced by placing a wearing ring round its periphery. The moisture given by the author, viz., 38 per cent. for the pulp entering the tube mill, is in the writer's opinion, somewhat too low, he having found that 40 per cent. or a little over, gives better results.



PEBBLE FEED HOPPER FOR TUBE MILL

REPLY TO DISCUSSION.

Mr. G. O. Smart (*Member of Council*): In replying to the discussion upon my paper, I would in the first place confirm after personal observation, the advantages claimed for Mr. Schmitt's pebble and pulp feeder for tube-mills, whereby the apex of the cone may be below the tube-mill axis where fall is limited, as is liable to be the case in additions to an existing plant. This device renders possible the installation of one or more tube-mills without extensive alterations to existing return launders and elevating plant, which would involve higher constructional and operating costs. In addition, this feeder spills very little of the inflow pulp, and thus enables the plant to be kept clean with less labour than is generally necessary. The curve of the spiral in Fig. III. appears rather sharp, and a longer and easier spiral should be better for a feeder of large diameter.

With regard to the underflow liner advocated by Mr. J. E. Thomas, it may be pointed out that in the arrangement described by the author a nozzle of the desired size is employed to regulate the underflow, the gate being merely used under special conditions, and especially to cut off the underflow entirely when not required, as when relining a tube-mill. Although not shown in Fig. II. in the paper, the sleeve round the inflow pipe mentioned by Mr. Thomas is certainly a good device and has been employed by the author for a considerable period. Regarding the lower percentage of moisture in a coarse underflow, the fact that such coarse product occupies fewer cubic feet to the ton than a finer product, shows the interstitial spaces to be less in the former case, and hence the percentage of moisture in such coarse pulp is lower. A recent series of tests, carefully carried out by the author, showed 39% of moisture to yield the best average results with a large tonnage of sand feed, although with a small tonnage of sand feed the best results were obtained with 27% of moisture, which is about the lowest percentage of moisture obtainable with diaphragm cones or any other dewaterer known to the author. Whilst the suggested safe-guard of return sand cones to intercept any coarse sand accidentally overflowing the tube-mill classifiers might be employed, yet it is doubtful whether the extra capital and elevating cost involved by the addition suggested would be repaid by the slight advantage gained, especially since a final cyanide pulp of the desired fineness can be obtained from the single classifier.

I have seen the simple and ingenious pebble feeder described by Mr. Thomas, and understand that it is being further developed and improved, and it is to be hoped that before long he will describe the final form of this useful device, for the benefit of the members of our Society, and that he will also afford them the opportunity of seeing it in operation.

Mr. Bayly's form of underflow cut-off gate with means provided for changing the underflow nozzle without discharging the contents of the cone, answers the purpose satisfactorily, and this advantage has already resulted in its adoption, extending beyond the Knights Deep where it was originally installed.

Mr. Blundun raises a very interesting point by his advocacy of obtaining a finished product by a single passage of the sand through the tube-mill. Unfortunately this system involves the reduction of many fine sand grains beyond the economic limit at the expense of sending coarse particles of ore to the cyanide works. Experience has shown that the coarser the average particles up to 0·27 in.* diameter in a tube-mill, the more fine sand is produced, and the coarser the outflow, the coarser must be the average sand particle in the tube mill. As a trial will show, the operation of a tube-mill plant to produce a finished product in one passage through the tube-mills will result in a coarser final pulp than if a larger proportion of the tube-mill outflow is returned for further crushing after reclassification.

It is difficult to ascertain Mr. Robertson's final conclusions as to the relative merits of the old and new system of classification, but a recent change at the Knights Deep to the new system has resulted in much better work. Whilst the principle of double classification referred to with interesting details by Mr. Robertson may be convenient with diaphragm cones in a crushing plant erected before tube-mills were introduced, yet the extra elevation of pulp it involves is unnecessary when satisfactory results can be obtained by single classification with diaphragm cones. In the case cited where two out of six tube-mills are stopped, this decrease in crushing units results in an increased volume of coarse pulp entering the remaining diaphragm classifiers, which ensures a coarser outflow and likewise a coarser feed to the four tube-mills still running, and thus their crushing capacity is utilised to the fullest extent possible.

Mr. F. Alexander's contribution is welcome for the definite information given, and the proposals set forth, but he has overlooked various important considerations which materially affect both his reasoning and conclusions. In the first place, whilst the pyrite in sand residues is richer per ton than the silicious portion, yet Mr. Alexander's own grading analysis as arranged below shows that :—

(*a*) The coarse +60 grade contains the most gold and the least pyrite per ton.

(*b*) The fine −90 grade contains the least gold and most pyrite per ton.

(*c*) The ratio of gold to pyrite differs very widely in the different grades.

TOTAL SAND RESIDUE.

Grade.	Per cent. by Weight in Residue.	Assay Value dwt. per ton.	Per cent. of Pyrite.	Dwt. Gold for 1% Pyrite.
+60	10·3	0·84	0·7	1·20
+90	30·0	0·70	2·0	0·35
−90	59·7	0·66	4·84	0·136

Hence with Crown Reef ore there is obviously more scope for winning gold by finer crushing of the coarse (+60) particles containing very little (0·7%) pyrite, than by reducing still finer the already fine (−90) particles containing seven times as much pyrite. The grading analysis further shows that only 20% of the total gold in the residue could be separated by careful removal of all the pyrite possible by panning. The average assay value of the whole +90 pyrite in the residue is actually less than the average value of the −90 pyrite, and even if the whole of the gold contained in the former were extracted, this would only reduce the total residue by 3·5%, or little more than half a grain per ton. From this it is apparent that in the design and operation of a tube-mill plant too much relative importance may be attached to the question of +90 pyrite.

Similar results to the foregoing have been obtained by investigation on various mines here,* and the following are the results of examination of a Simmer and Jack sand residue :—

TOTAL SAND RESIDUE.

Grade.	Per cent. by Weight.	Assay Value dwt. per ton.	Percentage of Total Gold.
+60	22·0	0·56	36·2
+90	30·6	0·40	35·4
−90	47·4	0·20	28·4
	100·0	0·34	100·0

It will be observed that the assay value of the −90 grade in the Crown Mines sand residue is relatively high in comparison with the +90 grade. This is probably due to the high percentage (43·2%) of very fine particles (−200) in the residue, which retards effective dissolving and leaching out of the gold ; if the charge had contained less slime, the value of the finer sand particles should have been lower and the amount of gold still less in proportion to the pyrite present.

From the Simmer and Jack residue it will be seen that the coarser (+90) portion constituted 52·6% by weight of the whole but contained 71·6% of the total gold present, whereas the finer

---

* W. A. Caldecott, "The Development of Heavy Gravitation Stamps," this *Journal*, Vol. X., April, 1910, p. 361.

* See Proc. *I.M.M.*, Vol. XIV., Oct. 20, 1904, p. 53.

(−90) portion, 47·4% by weight, contained 28·4% of the total gold. It is, therefore, obvious that the coarser richer particles will repay finer crushing better than the finer poorer portion, even though the latter is much more pyritic. Clean pyrite would no doubt repay extremely fine crushing were it possible to separate this material from the tailing-pulp at a low cost. The large amount of money spent in the past on close concentration by means of vanners or Wilfley tables here, and the fact that they have invariably been discarded, shows that such close concentration has proved unprofitable. Whilst Mr. Alexander does not propose to employ vanners, his suggested arrangement of hydraulic classifiers is avowedly intended to effect a similar result in separation of fine pyrite. Universal experience has shown that no hydraulic classifier can separate pyrite as perfectly as a vanner, owing to the overflowing particles of the former being determined by their size as well as by their specific gravity. Hence the sand overflow of the first square classifier in the Crown Mines diagram will carry away fine pyrite without its having passed through a tube-mill.

When laying stress upon the difficulty of classifying a "raging torrent" of pulp from 300 stamps, Mr. Alexander apparently lost sight of the fact that the settling area of the classifiers merely requires to be proportionately increased to obtain the same result as with the "placid stream" from 30 stamps. With single cone classifiers the number of tube-mills and cones increases in proportion to the coarseness of the battery screening, and hence when the limit of amalgamation in the stamp-mill is reached, the classifier settling area for securing separation of metallic gold becomes proportionately larger, and in the case of a 300 stamp-mill with coarse crushing would amount to several hundred square feet.

The diagram does not give dimensions of the square classifiers shown, but assuming that no particles above a certain desired size can overflow the first, it is difficult to see the reason why the second square classifier, whose operation is governed by the same laws, is not installed beside it, and the two employed to handle the mixed tailing and tube mill return pulp, since if the joint classifiers are proportional to the total volume of pulp, particles above the desired size will circulate through the tube-mill until crushed fine enough to overflow.

As already shown, Mr. Alexander's figures prove the coarsest and most silicious particles in the residue to be the richest; hence the reason is not apparent for fine crushing a portion of the tailing-pulp with a second tube mill whilst coarser richer particles are escaping from the crushing plant in the overflow of the first square classifier. In addition, the two-fold loss of fall in the operation of double classification and tube-milling shown in the diagram involves additional capital and elevating costs, and the number of classifiers and pumps increases the cost of attendance and maintenance, whilst the second tube-mill will not be supplied with the coarse particles whose presence so much assists in good tube-milling.

In single cone classification of a mixed pulp all the operating classifiers automatically adjust themselves to variations of the tonnage milled and volume of the tube-mill return pulp, but with two stage tube-milling, as illustrated in the diagram, the tonnage of sand fed to the first stage tube-mills must be regulated by the number of second stage tube-mills in operation at different times, in order to secure the desired fine overflow of the second stage square classifiers. Should one of the second stage tube-mills be out of commission, this may result in too small a feed to the first stage tube-mills to permit of good work.

The following is a useful formula for determining the tonnage of sand fed to a tube-mill, assuming that clear water from a steady head tank be used for diluting the pulp entering the tube-mill, and that by removal of the pebbles catcher or otherwise an accurate sample of the tube-mill outflow is obtained:—

$$S = \frac{W \times U \times O}{100\,(U - O)}$$

Where S = tons of sand feed per 24 hours.
W = tons of water added for dilution per 24 hours.
U = percentage of solids in cone underflow.
O = percentage of solids in tube-mill outflow.

As a slight error in the moisture determinations involved will seriously affect the calculated tonnage of solids, the greatest care in sampling and treatment of samples must be observed to obtain accurate results.

As regards the ratio of water to solids for amalgamation in the tube-mill circuit, this may be varied as desired with single cone classification, and my own practice is to employ a ratio of under 2 to 1 for the pulp passing over the shaking tables.

Where fine screening is used in the mill and a large tonnage of sand feed to the tube mills is desired, I have found advisable the use of two diaphragms in a 9 ft. × 6 ft. cone. These diaphragms are respectively 14 in. and 3½ in. in diameter, allowing an annular space 8½ in. wide between the edge of the upper diaphragm and the side of the cone, and a space 5 in. wide around the lower diaphragm. The upper diaphragm is 5 ft. 5 in. from the top of the cone and the lower diaphragm 1 ft. 3 in. above the apex of the cone.

ENDLESS ROPE HAULAGE.

(Read at December Meeting, 1909.)

By H. G. Kay (Associate).

REPLY TO DISCUSSION.

Mr. H G. Kay (*Associate*): In the discussion on the above paper which appears in the January *Journal*, Mr. Tom Johnson differs from my remarks on "Curve Wheels" by stating that "a deflection of 90° or more can be accomplished with little difficulty," this could only be possible in a case where the trucks are hooked wide and the rope is travelling slowly, and is not subject to a great strain at the curve wheel, such as would be the case on an up grade or steep incline. A 90° deflection wheel, together with a flatsheet would prove a very undesirable arrangement on a haulage where any considerable number of trucks are to be handled. As the angle of deflection at a curve wheel is increased so you increase the amount of wear and tear on the rope, jockey and wheel, and also the wheel friction. These items all mean extra expense, and by experience I find that if an angle of over 30° is to be negotiated it is cheaper, especially on a heavy and fast haulage, to disengage the trucks as they approach the wheel, gravitate them around the curve and rehook. If Mr. Johnson means that an endless rope haulage with the usual type of truck and jockey can be used on a 45° incline I entirely disagree with him. Such a grade would require a specially constructed truck with a long wheel base, low centre of gravity and a cover to prevent spillage. In stating that a grade of 1 in 6 was about the limit, I had in mind the common type of truck as used on the Rand, having usually a small wheel base and high centre of gravity, and with the jockey attached to the top of the body they become very unstable on a steep incline.

In the February *Journal* Mr. Webb calls attention to the fact that with a rope haulage in which the jockey is attached to the truck carriage, a stoppage occurring on the full track also necessitates a stoppage on the empty track. This would be the case in any rope haulage where the same rope works both tracks, but the trouble can be overcome by using two separate ropes, each worked by an independent clutch on the engine shaft. With the side jockey attachment Mr. Webb is quite correct in saying that the flange friction on the wheels is an objection, this friction is a small item on a level track but on a steep incline it becomes a serious item, by using two jockeys, one attached at each end of the side of the truck carriage, a large amount of this friction is done away with, but such an arrangement is inconvenient in working a truck around a curve.

GRADING ANALYSES AND THEIR APPLICATION.

(Read at May Meeting, 1910.)

By H. Stadler (Associate).

DISCUSSION

Mr. R G. Bevington (*Past-President*): As a preface to a few remarks upon Mr. Stadler's paper, I should like to say that I consider that the author has placed before our Society one of the most valuable papers upon a method of estimating the work done "mechanically," in reducing particles of ore from one size to a smaller one, which has been laid before a technical society.

As a portion of the work from which the author has been able to gather information and to verify his various formulæ has been carried out upon a plant under my control I have taken a great interest in his investigations.

Passing to par. 26 of the author's paper, it appears to me that it is difficult to apply Prof. Kick's law in its entirety to the material upon which we here have to work. The nature of our ore is a conglomerate composed of substances of different degrees of hardness and having different and unequal "technological states." Were one dealing with substances such as pure quartz or some homogeneous substance such as plate glass for instance, then Kick's law might apply in its entirety. I think, however, that for practical purposes, and that is what we workers want, the author's formulæ and deductions form an excellent basis for us to go upon. Passing on now to par. 28, the possibility of expressing the value of pulps in one term is undoubtedly of very great value. The time occupied in thus condensing the value of a pulp is small and comparisons are easy, and one is thereby enabled to keep a watch over the work which is being done by the crushing or grinding plants, and I can only say that I have derived great assistance in my work from the use of the author's formulæ for reducing the percentages of gradings of pulp to one unit, so to say, for purposes of comparison.

I quite agree with Mr. White when in the course of his remarks in proposing a vote of thanks to the author, he conveys that sufficient credit is not given to the tube mill for the work which it actually does in further reducing the particles which the battery discriminates against. As I have before mentioned, our ore is a conglomerate, and the entire credit must not be given to the stamps for producing by their powerful action *all* of what we term – 90 grade, as I take it that Mr. White will agree with me, some portion of the conglomerate is already of – 90 grade before it comes to the stamps at all. I think the tube mill is called upon to perform a very hard work in that it has to deal with the reducing of the hardest portions of our ore, and of pyrites which, (I think, all who have had to deal with the reducing of this material from one size to another will agree) is a very difficult substance to reduce, from one grade to a finer one, that is the only point upon which I think the author's paper does not give due credit to the fine grinding machine, the tube mill.

Mr. A. J. R. Atkin (*Member*): This contribution to the discussion is really an amplification of the remarks by Mr. E. H. Johnson and Mr. H. A. White.

The researches by Mr. Stadler into the mechanical energy required to crush banket are apparently based on a solid foundation when reared upon the law that bears the name of Prof. Kick, which postulates bodies of equal technological state for its truth. That this foundation is really as sound as it appears, I would like the assurance of the author, after considering the following brief description of the material to which he applies Kick's law. It appears fallacious to apply any figures deduced from the behaviour of homogeneous bodies, to one whose texture displays the greatest diversity, carrying as it does minerals from the softest in Mohs scale to the very hardest and most resistant remains of continental denudation.

The more fully to emphasise the difference between the banket and a rock to which this law might more fully apply, I will show you a couple of lantern slides prepared from sections of the rocks cut for petrological examination.

The first is a slide from quartzite. It will be noticed that the rock is very compact and that the original grains are all recrystalised, the orientation of the cementing silica being in optical continuity with the silica on to which it grew. The rock is practically one mass of silica. To this silica aggregate, if to any rock can Kick's law be applied.

The next slide is from a section of banket, and shows some of the many minerals that go to make up this varied rock. Here chloritoid appears filling the space between the pebbles together with chlorite, pyrite and silica. Now chloritoid is a brittle mineral, with a basal cleavage little less perfect than the micas, the cleavage of chlorite is very prominent, and its hardness is only 1·2. The pyrite is a brittle mineral easily broken. It will be readily realised from an examination of this slide that the falling stamp will produce a very different product of, say, – 200 (·003 in. aperture screen) size from the easily fractured lath shaped chloritoid crystals, and soft chlorite, with the brittle pyrite, than from the quartz pebbles, and it would be of considerable interest to have an analysis of the – 200 material produced by stamps, compared with an analysis of the – 200 material produced by tube milling finely crushed material that had been broken by the impact of the stamps. There can be little doubt that there would be a marked difference between the two products, for while

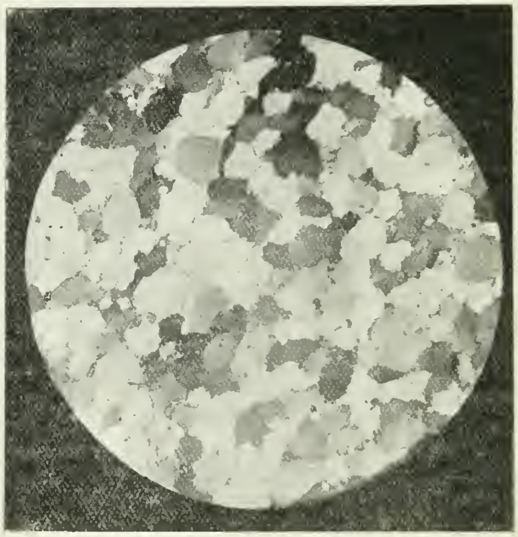


FIG. I. Orange Grove quartzite, consisting chiefly of quartz grains with occasional flakes of sericite between the quartz grains, × 20 diam. × Nicols.

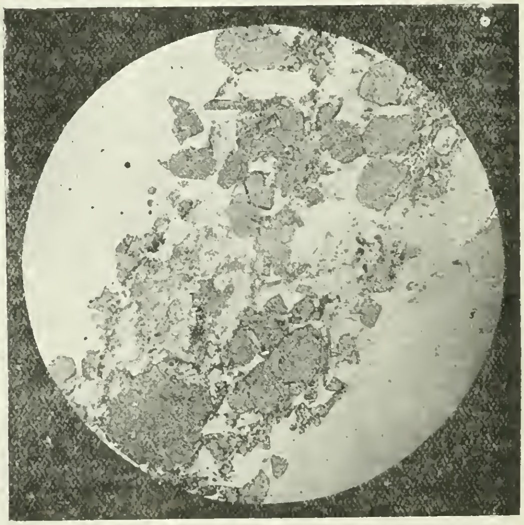


FIG. II. Banket, two pebbles (right and left side) are separated by the cement consisting of pyrite, quartz, chlorite, and chloritoid. Transmitted and reflected light, × 20 diam.

the matrix of the banket contains pyrite, chloritoid, chlorite, pyrrhotite, tourmaline, rutile, zircon, chromite, chalcopyrite, galena, carbon, blende, muscovite, sericite, gold telluride, iridosmine, apatite (R. B. Young, Geo. Society S.A., Trans. Vol. XII., 1909), and cobalt glance (A. F. Crosse, this *Journal*, Trans. Vol. V., 1904, p. 135), the pebbles consist of quartz, quartzite and allied silica, rocks all far more resistant than the minerals mentioned. Lest it be argued that the factor of error in the challenged investigation is constant for the rock from mine to mine, this is just what it is not. The calcite pebbles from the Meyer and Charlton—in reality pseudomorphs after quartz—consume very much less crushing power than the very hard pebbles from the Knights Deep, which again differ from the concretions often mistaken for pebbles from the Main Reef Leader of Langlaagte Block B. (R. B. Young, Trans. Geo. Society, S.A., Vol. XII., 1909, p 94). The more thorough the study of the mineralogical composition, and petrological characteristics of the banket, the more doubtful does the author's application of Kick's law appear. But before coming to any conclusion whatsoever, it is necessary to reason from experience, and to compare facts.

Mr. Chas. B. Saner (*Vice-President*) moved a vote of thanks to the retiring President for the able manner in which he had conducted the meetings of the Society during the past year. This was carried by acclamation, and the proceedings closed.

The Society's Annual Dinner.

The Annual Dinner of the Society was held at the Carlton Hotel, on Saturday, June 4th, the President, Mr. A. McArthur Johnston, M.A., in the chair. There were also present:—

Messrs. W. Abel, A. E. Adams, E. L. Adams, H. A. Adams, F. F. Alexander (Member of Council), F. Allen, R. Allan, D. G. Aspland, L. B. Bawden, W. Beaver, R. G. Bevington (Past President), E. Blume, F. L. Bosqui, C. B. Brodigan (Vice-President, Association of Mine Managers), R. Brooks, W. O. Brunton, A. F. Bruun, J. W. Buckley, Dr. W. A. Caldecott (Past President), C. W. R. Campbell (President, S.A. Institute of Electrical Engineers), J. G. Carter (Hon. Auditor), Palmer Carter, F. D. P Chaplin, F. T. Chapman, F. W. Cindel, L. Colquhoun, J. F. Cook, J. Cornet, J. Cowie (Secretary, Transvaal Chamber of Mines), W. M. Coulter, E. H. Croghan (Member of Council), A. F. Crosse (Past President), M. Cullen (Member of Council), W. Cullen (Past President), J. S. Curtis, G. A. Darling, H. F. Davis. W. H. Dawe, J. M. Dixon, Prof. J. H. Dobson (President, S.A. Association of Engineers), T. Donaldson, P. Duncan, C.M.G., J. F. Ferguson, M. Francke, A. Fraser, H. J. Fraser, J. A. Frerichs, F. C. Frey, J. A. P. Gibb, C. Glyn, E. Goffe (President, Transvaal Institute of Mechanical Engineers), K. L. Graham (Member of Council), Dr Grant H. Graumann (Mayor of Johannesburg), J. Gray, T. J. Greenwood, J. G. Hamilton, M.V.O. (President, Transvaal Chamber of Mines), Dr. J. McC. Henderson, W. G. Holford, E. Hollis, J. H. Hopkins, M. C. Inglis, H. J. Ireland, A. E. Irwin, W. H. Jane, E. H. Johnson (Past-President), J. H. Johnson, Tom Johnson, J. Johnston (President, Pharmaceutical Society of the Transvaal), T. S. Jolly. R N. Kotzé (Government Mining Engineer), E. J. Laschinger (Member of Council), Principal J. G. Lawn (S.A. School of Mines and Technology), H. Lea, J. Lea, Q. J. Leitch, C. D. Leslie, W. Ivor Lewis, R. Lindsay, W. D'A Lloyd, Dr. D. Macaulay, H. S. Macgregor, J. Matheson, S. Matthews. J. McCracken, Dr. J. McCrae, J. Meeser, H. Meyer, Dr. J. Moir (Vice President). H. H. Morrell, J. Munro, S. G. Murphy, Dr. Temple Mursell (President, Transvaal Medical Society), G. Nathan, S. Newton, W. Nicklin, F. B. Ogle, E. A. Osterloh, E. Pam, S. H. Pearce (Past-President), J. C. Phillips, Lionel Phillips, R. G Campbell Pitt (President, Rand Pioneers), O. P Powell, W. S. V. Price, J. F. Pyles, *Rand Daily Mail*, E. G. Rance, S. Richards, A Richardson (Member of Council), C. A. Robinson, W. Rockey, Fred. Rowland (Secretary), A. Salkinson, C. B. Saner (Vice-President), H. E. Saner. R. B. Saner. W Sharp C. B Simpson, *South African Mining Journal*, G. O. Smart (Member of Council), T. F. Smith, G M. Smythe, A. R. Stacpoole, H. Stadler, Prof. G. H. Stanley (Member of Council), *Star*, J. H. Stevens, R. Stokes. A. N. Stuart, W. R. Tait, W. Tayler, H. Taylor. R. M. Taylor, A. Thomas. H Musson Thomas. J. E. Thomas (Member of Council), H. T. E. Thorpe, W. E. Thorpe, M Torrente, *Transvaal Leader*, J. Walker, P. A. Walshe, G. W. Ward, F. Wartenweiller, F. W. Watson, T. H. Watson, J. W. Watts, H. H. Webb, E. M. Weston, W. Westwood, W. S. Whaley, A. Whitby, (Member of Council), H. A. White (Member of Council), E. J. Wiseman, Prof. J. A. Wilkinson (Member of Council), L. J. Wilmoth, Prof. R. B. Young (President, Geological Society of South Africa).

The usual Loyal Toasts having been duly honoured,

Mr. J. G. Hamilton, M.V.O. (President of the Chamber of Mines) in proposing the toast "The Chemical, Metallurgical and Mining Society of

South Africa," said, Mr. President and gentlemen, I have to express my acknowledgement of the honour you have done me this evening in making me one of your guests, and my admiration of your courage in entrusting to me the toast of the Society. I regret that it is a number of years since I was enabled to be present at one of these annual functions, which are amongst the greatest of our mining year. However, the loss has been mine and I am happy to be here this evening. To go back, the history of this Society is practically the history of these fields. It is I believe some 16 years since the Society was initiated. Since that time it has grown in numbers and importance and it has participated in every move in the mining world. I understand that the Society to-day is the strongest in the whole of the southern hemisphere, and that as a matter of fact it is one of the strongest of the kind in the whole world—from China to Peru—and the work of your members is well known throughout the world, and I believe I have to congratulate you on the fact that one of your prominent members and a former President, has recently been highly honoured by his *Alma Mater*, the Cape University. To realise the work which your Society is doing, it is necessary to glance through the *Journal* which you periodically publish. Otherwise it would be impossible to grasp the various subjects in which your members are interested. In proposing the toast of the Chemical, Metallurgical and Mining Society, I believe it is absolutely necessary to go back to your long ago predecessors in the title. I do not know whether I can go back to the time of the old astrologers, but anyway we can go back to the time of the Alchemists. I believe their chief function in life was the discovery of the philosophers stone. They imagined that when they discovered that stone they would have discovered a panacea for all ills, they would have got rich in their old age and would have become rejuvenated again, although I am not certain that their immediate posterity would have appreciated that. But their great work was to learn the secret of the transmutation of metals. Their idea was to transform the baser metals into gold, and when tired of making gold they might eventually make a little silver. Well, you gentlemen, are their lineal descendants in title. You have to move in a different sphere. You have followed in a more practical method and have assisted in the extraction of the metal from this ore in which the results have on the whole been more satisfactory. You have achieved much in the past. You have still much before you I believe to achieve. We are interested in the discovery of how far the baser metals—copper, tin and other metals—occur in this country. That will open a wide field for you gentlemen. Again, you gentlemen will be interested in the chemistry of agriculture and you can do a great deal in that direction. Mining in this country, as in other countries, leads up to other industries, and more particularly to the main industry of all countries, that of agriculture. The expansion of the gold mines here has to my mind rendered possible the success of agriculture in the Transvaal. It has to my mind even gone further. This industry has made the success which the Transvaal is to-day, it is this industry which has made possible the Union of South Africa which we have been recently celebrating. I do not intend to touch upon politics, my friends, Mr. Chaplain and Mr. Duncan, will no doubt deal with that subject. It is only a few days ago since those of us who have been in the House of Assembly, have been relieved of our Parliamentary duties, and have come back into private citizens again. Therefore, I have no intention to-night of touching upon politics in any way. As I have said, gentlemen, you have much before you. Scientists are insatiable. They are always sighing for fresh worlds to conquer, and we cannot wonder at it. If we look back at the strides made in the last century—nay to the last decade—we cannot wonder that scientists are sighing for more worlds to conquer. What would we have thought if any one had told us a short ten years ago that a man would have flown on a biplane from Dover to the French coast and back. We should have thought that such a man was drawing very largely on his imagination, and yet that is what has recently been done. In view of the success of the past, I think, gentlemen, we are justified in looking to you for innovations in the future. I think we can confidentally look to you still to show us that there are more things in Heaven and earth than we have yet dreamt of in our philosophy.

The President: In thanking the President of the Chamber of Mines for his kind remarks on our Society this evening, I would remind you, gentlemen, that there is an old saying that a prophet is not accepted in his own country, and another that listeners seldom hear good of themselves. You who are members of this Society are therefore in a unique position, since Mr. Hamilton has done his best to impress us that we are much thought of and that we are going good work.

Well, gentlemen, it is not for me to gainsay all he has said. Your Council has so often impressed it on me that we are the best Society in South Africa, that I am beginning to believe it myself and certainly when we remember that it is just about 16 years since we had our beginning, that at the first muster the members present numbered about a dozen, whilst they now approximate 1,200 we have reason to feel at any rate that we justify our existence. Not only do we excel in numbers, but the quality and character of our papers during the past year tend to maintain the high record set us in earlier days. My earliest remembrance of work in this Society recalls an occasional lively search for a paper, my latest tells that there has always been an abundance of papers, indeed during the past year there have generally been two original papers submitted to each meeting, when there has been an average attendance of about one hundred.

These remarks I make, gentlemen, not in a boasting spirit, but because I wish to impress upon those here to-night, who are not regular attenders of our meetings, that it were well for them to become so. I wish to impress also on the leaders of our mining industry that it is to the interest of the industry that they encourage us with their presence occasionally, whilst I may say that the Council is at present considering a proposition whereby mining papers may be encouraged and thereby make some at any rate of the meetings more personal to many of our mining men.

The work we have done in the past and what we intend to do in the future, for I speak from personal knowledge when I say there is no rusting going on, in the interests of the mining industry both by encouraging the meeting, with consequent interchange of ideas, of all those engaged therein and by the reading of papers and the dissemination by means of our *Journal* abstracts from the leading periodicals on subjects pertaining to the industry, is enormous and its magnitude may be roughly gauged by the fact that we have not been able to persuade a President to occupy the chair a second time.

Considering for one moment the industrial features of the past twelve months, we can but come to the conclusion that a steady, quiet evolution is going on. We have often been twitted for our conservatism and I acknowledge that there is some element of truth in the assertion, but when we consider the vast extent of our industry and the tonnage handled per day, I am convinced that even our most inveterate critics will, in their sane moments, acknowledge that a policy of *festina lente* is well advised.

Perhaps at no time during the existence of our Society has there been on the Rand an era of such great moment in metallurgical practice. In former days there have been many radical changes, but it was a common factor in our consideration that what was proved useful and advantageous to one mine was eagerly sought after by the others. In this way was built up what is known all over the world as Rand practice, a practice which has been severely criticised, more often unjustly than justly. At the present time, our leaders are divided. Great differences of opinion exist, and I think I am correct in saying that no more healthy sign could be evidenced of the interest involved than has been apparent in the discussions printed in our *Journal* during the last twelve months. We have followers of coarse crushing and tube milling, of fine crushing without tube milling, of plate amalgamation after the mill and after the tube mill, of amalgamation only after the tube mills, and of no amalgamation at all. In secondary treatment there are advocates who treat the sand and slime as one product, others who would treat the finest sand with the slime, whilst detractors of the decantation process for slime settling are claiming better results by means of filter presses, leaf filters, etc., Even the old established zinc shavings are being challenged.

I can thus safely say, gentlemen, that our industry is in a very interesting stage metallurgically, and I think that there are few members of our Society who are not well acquainted with each and all of these problems and indeed keenly interested therein. We, as individuals and as a Society, want to know all about improvements, we want solid facts and definite figures, and having been convinced that any change is for better extraction or for lower working costs, I am confident that metallurgists here will welcome such improvements not only in the proper spirit, but in the right manner by recommending their adoption. Mining men also are facing problems, which will tend greatly to the betterment of conditions underground. First of these is the matter of ventilation. I need scarcely recall to members the times of keen controversy experienced some years ago in our midst by this very question. It is pleasing to record that since then the ventilation of mines has received the attention of mining men which its importance demands, and there can be little doubt but that future reduction in working costs will benefit thereby since the output capacity of our miners will be increased. Of almost equal importance too is the sand filling of stopes. The two subjects are intimately connected, for I can recall how the worked-out ground in the upper levels was always best supplied with fresh air, so that only more or less vitiated air reached the working faces. The sand packed stopes will not only strengthen the hanging and the footwall, but they will tend to divert the pure air to the present workings. It may be some time yet, but we can, I trust, look forward to the day when close behind the working face will follow the solid packing of the sand.

A pleasing feature of the past year has been the increase in numbers of our student members. These, as most of you know, are apprentices to whom in the future we would look to carry on our work, and it behoves us to give them the right training, a theoretical as well as a practical. Many of these youths are graduates of our universities or are college students, who in former days joined the professional classes, but now are encouraged into the industrial ranks, with, let us hope, a brighter prospect for them and for the wealth of the country. There is no reason why with careful attention to work and zeal in following university or college extension work, they should not become highly trained technical men in their own departments. What we want from them is, after completion of their *alma mater* course, sufficient grit to enable them to become graduates in, what Cousin Jonathan calls, the university of hard knocks.

Let me add just one word of advice to all our student members and even to our younger members. Remember that your competitors are not only to be found here. Once you get into position of trust, you will find that rivals are drawn from all over the world and it is necessary for you to become thinking members, well read in everything pertaining to your work and strenuous to keep your end up. Keep well abreast of modern practice in all its phases—do not sit still and do your daily work only and then grumble should one be put ahead of you.

For the encouragement of these youths, your Council has been pleased to offer to the Transvaal University College a sum of £20 per annum to be given in book prizes to the evening students in chemistry and metallurgy. This extension of our usefulness will, I am certain, meet with your approval. The principle evolved a year ago of having Corresponding Members of Council has been further extended with beneficial results, and we can now claim to have representatives in all the gold mining countries of the world, who not only assist in procuring papers for us, but have extended and continue to extend our sphere of influence quite appreciably.

Of great benefit too has been the interchange of ideas and methods during the visits of several of our members to other mining centres, and by the visits to the Rand of other leading metallurgists from several of these gold fields. Our visitors we have received in as hospitable a manner as possible, indeed some of them have complained of our hospitality in so far as we have made them loth to leave South Africa. All of them have expressed general approval of our methods, whilst one of them, in a recent issue of a mining paper, refers to the milling practice on the Rand as being the best in the world.

Our connection with the Institute of Mining and Metallurgy has been cemented during the past year by the joint reading and discussion of papers of equal interest to the members of both Societies, by visits of Mr. McDermott, Mr. F. W. Harbord and Mr. Marriott, and by my own reception, as your President, from them in London. As you know, the I.M. and M. are initiating standards applicable to mining and metallurgical work which shall be world-wide in their application. Their adoption generally will simplify matters very much when comparison may be initiated between different centres We are endeavouring to follow in the matter of mining and metallurgical terms, and we trust soon to be able to invite all our members to criticise our efforts and assist us in formulating terms which will be acceptable, if not to all, at least to a majority of mining and metallurgical men.

Mr. F. D. P. Chaplin, who met with a very cordial reception, in proposing the next toast "South Africa," said :– Mr. President and gentlemen, in proposing the toast of the Society a few minutes ago, Mr. Hamilton made the remark that these dinners marked an epoch in the mining world of the Rand. I think also they mark year by year the progress of South Africa. Last year if I remember rightly we listened to an eloquent address from Sir Percy Fitzpatrick on the advantages that would accrue to South Africa from the accomplishment of Union. Well, since then, things have progressed and Union has been accomplished, and as your President has said just now, the condition of South Africa is like the condition of the industry, an extremely interesting one. We are face to face with a new order of things, and it is a matter of vital importance to all of us how things under the new order are to be carried out. We are all agreed that obviously we are all to do our best for the future of South Africa, but I take it as time goes on, it will appear that we shall not be all agreed as to the best means of bringing about the prosperity which we desire. It appears now that we are to have introduced what is called party government. I am not going to express any opinion as to whether any other form of government would have been possible, or whether if it had been possible it would have been desirable, but it appears that South Africa is now about to endorse the sayings of that great party leader in England, that without party, Parliamentary government is impossible. Well whether that is so or not, we are going to have party government in South Africa, and I take it we are going to make the best of it. I am quite sure that whatever differences of opinion may divide parties in South Africa during the years to come, members of both parties will combine on questions on which there can be no factious difference, and on questions on which the whole opinion should be concentrated. We are not going to quarrel amongst ourselves for purely party purposes, and whatever the Government of the day, whatever party it represents, puts forward a programme, and puts forward measures which appeal obviously to everyone, and are in the general interests of the country, then I feel sure that whatever party is in opposition, that party will not give way to pure factious disturbance, but will do its best to support the measures which are in the interest of all. Well, gentlemen, we are accustomed to look at the advantages of Union. I am not going to detail what I believe to be those advantages. I think they are patent to all, but I would like to dwell for a moment upon the obligations which Union must confer upon us. We have now as a South African nation to take our proper place in the Empire, and if we are going to take our proper place in the Empire, we are going to show that we are worthy of it. We are going to consult not only our own interests but also what in a wider sense is our own interest and that is the interests of the Empire at large. We are going to take upon ourselves a proper share in the defence of the Empire. We are going to make proper provision for the internal defence of the Empire. We are going, I hope before long, to see steps taken to bring about a system of land defences—a system which is not a hard and fast military system—but which is suited to the requirements of the country, and which will enable us when the time comes, if it ever does come, and we all hope it never will come, to play a proper part. Because gentlemen, the quarrels of the Empire are going to be the quarrels of South Africa, and we have got to be ready if and when the time shall come. But, gentlemen, these things cannot be done without effort. They cannot be done without heavy sacrifice and without money, and they cannot therefore be done without increasing the population. One of the things which I hope we are going to see in the near future, is a determined attempt to increase the white population of this country. There are surely openings for it. We are going, I hope, to see an increase of population on the land. We are going, I hope, to encourage immigration, the immigration of people of the right kind. We are going to facilitate their coming and to make provisions for them in this country by settling them upon the land, and by offering inducements, legitimate inducements to them to come and settle amongst us. And when they come we are going to make them at home and we are not going to treat them as outsiders. I hope gentlemen we are not going to be put off with the cry, which I have heard raised before now, that there are plenty of people in this country, and that everyone in this country is not already employed, and that we had better see that they are employed before we think of inviting anyone else. I think there are probably less people actually out of employment in this country than in any industrial community of the same size. I hope that will always be so. What we have got to do is to find openings for people. We can find these openings on the land by a proper encouragement of agriculture. We have got to spend money on it, and I believe no one in this industrial community which pays so large a share of the revenue will grudge for one moment reasonable sums expended upon agriculture and agricultural education, if they are sure that the money is wisely spent and spent without favour and strict impartiality. And gentlemen there is one thing which I think is worthy of attention at the present moment. We do not of course want to interfere, if we could, in Home politics, though of course we cannot disregard the fact that there is a very close relation between our politics and Home politics in the greatest sense of the word. Now what I mean is this. We have a great deal to gain in this country if the Home country, which supplies the biggest market in the world, would give some preference — even a slight preference — to Colonial products. I do not know whether any of you have read in the papers which come out from London only a few days ago a most interesting statement showing the effect of British trade on the preferential system with Canada. There is no getting away from it that the existence of the preference which Canada has given to British goods has made an enormous difference, and in this country we now give that preference. What we hope is—I am looking at it now from our point of view—that the time will come when the Home people will give us preference because if they do not after a time it must inevitably happen in this country as it appears to be happening in Canada, that people will not wait indefinitely, and they will say "We are not going on giving you preference if you do not give us preference in return,' and then they will make their own arrangements with the foreigner. I do not think any of us want to see that, because we are going to lose enormously by it if that comes to pass. We are going to lose the advantage we should get by being partners in that consolidation of the Empire, which I believe trade reciprocity will do more than anything else to bring about If we can do something in this country to bring that about, I think we shall be doing something to show that we have earned our proper place in the Empire Well, gentlemen, I have dealt with the land, and, of course, what I have said as regards the land applies with equal force to the industries of the country. Now, unfortunately, at the present time we have got only one prominent industry in the country. I hope, I wish for many reasons we had more. If for no other reason, it would be an advantage, because other industries would take up some of the criticism — not always fair — which is so abundantly showered upon our industry to-day. I think we can do a good deal, but it is no use thinking that everything is going to be plain sailing, and that our progress will come without effort. We know very well that the ore of our mines will not get any richer than it is, and we know quite well that if we are going to expand this industry we have got to work cheaply. We have got to take advantage of every form of mechanical and metallurgical skill. Then we are at once providing openings for a larger number of white people to earn a living in this country, because there is nothing to my mind more sure than this, that if we wish to see more white people employed here we have got to provide openings for them not only in unskilled work but in work that requires a certain amount of skill. It is where skill is wanted that the white man is going to find his opening. I believe far more in finding openings for white people in that way and by taking other measures—I see that I am treading on dangerous ground. Well, gentlemen, there is no body of men who can do more, who is doing more to help South Africa than this Chemical, Metallurgical and Mining Society of yours. You have heard from your President some of the things which the Society has done and is doing, and in my humble opinion your President has been far too modest. I believe the efforts of this Society, and the people connected with it have had an enormous effect in promoting the prosperity of the mining industry, and I believe as time goes on that they will help enormously in the starting of other industries. We are bound to have other industries in this country, and I hope that as they start we shall give them adequate encouragement. We ought to protect them sufficiently and give them a fair chance without overburdening the taxpayer of the country. We have, of course, in this country to reduce the cost of living. We are going, I hope, to see the railways administered on business lines We are going as a consequence to see the cost of bringing goods for example, say, building material reduced, and we are going to see it therefore made easier for people to establish their homes in South Africa. To that extent I hope we shall protect such industries as arise, and by so doing we shall do something to build up that white population which is so badly required in this country, and which, as I have said already, is absolutely necessary to us if we are going to take our place in the Empire to find the men and money which it is up to this South Africa to provide. Of course with the advent of an increased population, facilities for the living of that population will arise. We have got to see that all sections of this country have a fair chance, and that things are conducted in a manner which will give equality of opportunity, and we have got to see as I have said already that the new comer is made at home. I think the Government of the day, which begins the administration of South Africa, has an enormous responsibility laid upon it, and I believe everybody will be desirous of giving that Government its chance, and I believe as I said at the beginning of whatever party it belongs brings forward measures which are obviously in the interests of the country, all sections will unite in its support. I think we can look forward with hope and confidence to a prosperous South Africa. I therefore have much pleasure in giving you the toast of South Africa coupled with the name of my friend, Mr. Duncan, who was a member, as you know, of the Crown Colony Government which in spite of all criticism laid the foundation of the prosperity which is now before us.

Mr. Patrick Duncan, C.M.G., in response to the toast, said: Mr. President and gentlemen, I must begin by acknowledging the manner in which this toast has been proposed by my friend, Mr. Chaplin, and in which it has been received by you. After that I think I ought to apologise because I believe my appearance to-night is in the nature of a stop gap, that it had been originally intended had the fates been kind, that this toast should have been replied to by a member of the new Union administration. Well, gentlemen, I can only share your regrets that that has not been possible and that member could have seen his way to tell this gathering something of the events even of the last few days, he would have initiated you into a chapter of South African history which, if not, elevating, would at least have been entertaining. Well, I must acknowledge as I said, the way in which you have received this toast, and indeed it was only to be expected that such a toast as this should have been received by a Society such as this, because your Society recognises that while your work is primarily scientific and industrial, it is these things which have no small influence on the destinies of the country in which you work. I believe you all identify yourselves with these destinies, and it is in that spirit I wish to talk to you to-night. I remember seeing in a paper sometime ago in Natal, where a prominent politician replying to a toast said that the great thing about South Africa was that its future lay before it. Like many utterances of politicians that statement, though it might be called a platitude, was intended to have a real meaning. What I believe the gentleman who gave utterance to it meant was, that the future of South Africa looked to a greater extent than could be said of any of the older countries in the world, to be in the hands of the present generation, that it is for us to decide according to our opportunities. It was not for us to decide in what direction this new country should advance, and to that extent our responsibility is not a small one. It would have been equally true and equally significant if he had added that the past of South Africa lies behind it. If he had said so I believe he would have meant that the past of South Africa is one which marks it out in a special way among the other great nations which form the British Empire. We have to look back upon a record of internal conflict, first of all with the forces of nature and with the many plagues which for one reason or another have seen fit to land upon this part of the world, wild beasts and equally wild men, and last but not least, generations of internecine conflict between the two civilised races whose destiny it is to make this land a civilised nation. That I think is a history which is not shared to anything like the same extent by any of the other great colonies of the British Empire. It may give us reason for pride, but it also imposes upon us heavy obligations which are not imposed upon those other countries which have had a more reasonable course of development. But now I hope we have got beyond that stage of conflict between the two civilised races who have between them the task of civilising this country. At Vereeniging, eight years ago, a treaty was agreed upon, which put an end to armed conflict between these two races. A few days ago at Pretoria was consummated what I may call another treaty, a treaty which meant that the differences between the two races were to be laid aside, and from that time forward they were to work together as one nation. Now I believe that practically everybody has accepted that last treaty as a basis on which they are going to work out their future destiny. There will be relapses, but I believe in the main the spirit in which that last treaty was entered into is going to be observed in South Africa. Well, that gives us a basis on which we can work out our future destiny. Now, if we compare ourselves with our neighbours among whom we find ourselves in the British Empire, with those three great self-governing nations with whom we are now in partnership in the British Empire, I think we see we have some thing different in the task which lies before us and that which lies before them. We share with Canada Australia and New Zealand the problems of internal development, we share with them problems of defence, problems which are perhaps greater than anything, problems of bringing into closer relations the different parts of the British Empire. We shall take our share of that burden, but we have before us a task which they have not. That was alluded to in a speech in which Mr. Chaplin proposed this toast, namely the question of population. It is of little use to us in South Africa as a part of the British Empire if we work out all these other questions of internal development and material prosperity, if as a people we are going to be a small minority with an overwhelming number of an alien population who have a standard of civilisation rather different from ours and with whom we can never be assimulated. We must enter the British Empire on the same plane as Canada, Australia and New Zealand. If that is an impossible destiny for us, then our share in the future of the British Empire is going to be less than most of us hope it is going to be. There are conditions which are existing now which have got to be overcome, if that destiny is to be ours. For one thing we see great blocks of land in South Africa capable of bearing rich crops which are wanted in the markets of the world, and on which Europeans cannot live because of the various diseases existing to people and animals there. Again, we see in the industrial world conditions existing which do not make for the settlement of a permanent population which invite men to come here and leave their families behind them to join them again as soon as they can. Another thing we have is that vast alien population to which I have alluded, in the presence of whom we see that uneasiness—sometimes that fear—and whom we are afraid of treating justly, because we know how near they are to ousting and overwhelming us. When we have righted matters, perhaps we shall be able to do these men justice and give them the conditions they ought to have in this country. Now with regard to the first of these problems which I mentioned. It seems to me that science will do as much and more for us in the future than it has done in the past. On listening to your President's speech to-night, it was borne in upon me that for a scientific man there was no such thing as impossibility. There is no reason why we should despair as to what we can do in the future, or that those parts of our territory which are uninhabitable to Europeans, will continue to be. And the same thing will apply to the industrial conditions I spoke of. In regard to other questions such as the native population, I do not see that we should despair as to our statesmanship. It seems to me that there is a problem more difficult than any similar problem in any other part of the world, but there is no reason why we should sit down and say it is unsolvable. I believe if we had in this country, as in America, a basis of population, we should be able to look this question fairly and squarely in the face. Now I should like in mentioning these problems just to remind you that in approaching them we should do so with the idea in front of us that we are going to make this country a European nation. If we have that idea in front of us, and keep it in front of us, then it establishes the lines on which these problems are to be approached. Without such an ideal we are floundering in the dark. With such an ideal we shall attempt to solve them in the only way it is possible to attempt to solve them. Now it is up to us as representing the scientific and industrial population of this country to take the lead in the solution of these problems. I know as it has been said before this evening, that in this country those who come in from outside are apt to feel that they are tolerated rather than welcomed here. It has generally been the boast of any public body who has to make appointments to any office, that they have been very careful not to appoint anybody from oversea. It is generally regarded as a crime to ask people from oversea to come and fill an appointment here, and strange to say particularly in regard to education. I have seen various members of School Boards who are appealing to the electorate for a renewal of their support declare that they are dead against the policy of importing teachers from oversea. Now it seems to me they should mention that in a spirit of repentance rather than of self glorification. If we want to progress it should be in the direction of education and learning, and if there is one way to education which will benefit us in this country, it is in being brought into contact with ideas from elsewhere. We have to compete with the education of the world, and should have the best education in the world in order to meet that competition. Now I think that we ought to live down that state of things in this country, and we ought to do it in two ways. First by reflecting upon what we have already done in this country, and secondly by the spirit in which we approach the problems in this country. If you compare the country as it is now with what it was 30 or 40 years ago, surely you will be compelled to recognise that a great part of that immense development is due to men who have come in from elsewhere, and brought with them that science, education, and training which they have gained elsewhere. Not only in this vast industry on the Rand, where things mechanical have been brought to a pitch of perfection, but in the sphere of agriculture where new methods and innovations have been introduced and where diseases have been exterminated. There you will see the spirit which has come from outside and which would never otherwise have been developed in that period of time if all who were not natives of this country had been kept out as strangers If you will look on that record of progress I think you will see that the incomers to this country have something to be proud of. You will feel that the men who have come in from outside have done something for South Africa. You will find all over South Africa the graves of men who have laid down their lives for South Africa who have come in from outside, men who have broken the power of Zulus, who have broken the power of the Kafirs in the Eastern Province, who have made this country fit for civilised men to live in, and asked no reward for it. I think on looking back on that record you should also make up your minds that as far as you are concerned you will serve this country in the same spirit, not for the hope of reward but in that spirit of service to tht Union that will be for the good of the country, that other men who come after you may reap the fruits of what you have done. That is the true sense of patriotism, not the man who talks a lot about himself and does little, but the man who is prepared to serve his country obscurely, even if no one but himself knows it, without hope of reward or recognition, but on lines which he believes will be best. It seems to me that we have got to establish our claims to this country by serving South Africa in that spirit, and if it should be our lot to dig and plant without hope of seeing the harvest, yet if we know we are working on right lines, if we know we have a practical ideal which will make this country better than we found it, we shall have the consolation of knowing that those who come after us, even though they do not know it, will reap the benefit of what we have done, and we shall have done something to make South Africa one of the great new nations of the world.

Mr. Chas. B. Saner, A R S.M., M.I.M.M. (Vice-President), next proposed the toast "Our Guests" and "Kindred Societies," saying: Mr. President, Mr. Mayor, and gentlemen, it is my official privilege to-night to accord a hearty welcome on behalf of our Society to our guests and the representatives of kindred societies When I look round and see the many distinguished men sitting near our worthy President, men of science—I will not say men of letters as I do not know whether there are any here, men of law, masters of commerce and finance, I realise how fortunate we are in having such distinguished guests to-night. I feel that in having all the representatives of our kindred societies present to-night we, as a vigorous and straight talking society, should appreciate it. I will not dilate on the various short comings of the other societies but I may speak about the premier society here, the Town Council. It used to be called the gas society, not on account of the gas engines, but on account of the amount of gas they used in their deliberations. But thanks to our Mayor who pulled the finances together, who is one of the princes of finance, we may say that our Town Council is as good as that of any other city in South Africa. Then we come to the Mine Managers Association. I see Mr. Brodigan over there. That Association deliberates on Friday. They are but few in numbers, but most of them can give us the sack to-morrow. Then comes the South African Association of Engineers. Some few years ago Johannesburg arrogated to itself the compliment of having more brains to the square inch than any other city in the world, and I think the South African Association should have more brains per member than any other Society in South Africa. Whenever you go to their meetings you see *sage nods*, but you never find any criticism. We then come to that wonderful society who call themselves "Time" the Transvaal Institute of Mechanical Engineers, who spend a lot of money. Money is no object to them. The other societies are too numerous to mention and we can only hope that in the near future most of the societies excluding our own will amalgamate. The last society we have got here and which has just come into necessary being is the South African Institute of Electrical Engineers. They have taken on a very big title, and have a very big job before them. Electrical work is just making way on the Rand and it is going to loom very largely before us in the near future. I hope they realise that they have not simply got to light us here, but that they have got to light us 1,000 feet underground. We miners and engineers look to them to help us. If they give us the means, we will provide the gold.

Mr. H. Graumann (Mayor of Johannesburg), responding to the toast, said: Mr. President and gentlemen, I have in the first instance to thank Mr. Saner for the kind and flattering way in which he has proposed the toast of the guests and kindred societies. With regard to the latter I think I will leave them severely alone, as I find Mr. Campbell is here to respond on behalf of these institutions, and I hope and trust and have considerable confidence in the belief that he will be able to give Mr. Saner just as much as Mr. Saner was able to give him. With regard to the premier society, I propose to say nothing and I believe the reason is obvious. There is one other thing I would like to tell you, and that is this, that whilst I appreciate, and appreciate very highly indeed on behalf of the guests, the kind hospitality which has been extended to them to-night and for which I am truly grateful, I do not propose at this late hour, after having had the pleasure and privilege of listening to such eloquent speeches, either to worry or weary you. What I would say is this. I paid special attention to one of the observations that was made, I believe, by Mr. Duncan with reference to South Africa when he said that the future lay before it, and I might say in the same Irish manner, that the future of this Society lies before it, that a very great deal will depend upon the capacity and power of its members, as well as on the members of other kindred societies, to increase the operations of this country. I thank you from the bottom of my heart for the extremely kind way in which you have received and entertained your guests to night.

Mr. C. W. R. Campbell (President, S. A. Institute of Electrical Engineers) also responding, said: A few days ago, more by accident than otherwise, I was accompanying an eminent member of my own profession in a tour round the reduction works and surface plant generally of one of our largest mines. He was shown everything from the ore coming up the shaft to the little bar of gold produced, which represented approximately one day's production, and the thing which will always appeal to me about that was the enormous disproportion between the ore got up in the mass and the actual amount of the finished product. I think it is one of the most wonderful achievements of modern times. It deserves to rank with the wonders of modern science and it is the mining men, chemists and metallurgists of South Africa who have taught us how to achieve that economically and efficiently. Mr. Hamilton mentioned about scientists sighing for new worlds to conquer. During the last few weeks it has been my misfortune to travel a great deal up and down the reef in very windy weather, and it strikes me very forcibly that there is a world to conquer there, that our engineers, chemists and metallurgists should find some way of disposing of these dusty by-products, either by making some use of them or by putting them back where they came from.

Mr. Lionel Phillips, in proposing the health of the President, said: Mr. President and gentlemen, when I came here to-night I thought it would be my privilege for once at least to enjoy my dinner without having the exertion of a speech, I am fortunate however in being called upon at this late hour, for I am sure it is not your desire any more than mine that I should make a speech, but it gives me an opportunity of saying something which at this hour we are all pleased to have and that is the toast of the President. Before I do that, perhaps as everyone before has dwelt upon the necessity of increasing the population of this country it would not be fitting that I should sit down without saying something on the subject. According to modern theory it is believed to be possible to bring into contact inorganic elements and produce living bodies, and I therefore suggest that as you have in this place a great many inorganic elements, and Mr. Campbell complains of their flying about somewhat inconveniently, you may in the course of your studies perhaps find a means of bringing that powdered material together and producing the population which we need so much. I do not propose to encroach upon your time to-night and will ask you to rise and drink heartily to the health of your Chairman, who has presided so very ably to-night.

The President responding, said: Gentlemen I am sure I need not detain you any more than simply to thank you very much for the kind way in which you have honoured my health. My Presidency of your Society has given me a great deal of pleasure, and I am pleased to see how much my efforts have been appreciated as shown by the kind remarks regarding your Society this evening.

The singing of the National Anthem brought the proceedings to a close.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Separation of Vanadium, Molybdenum, Chromium and Nickel in Steels.—"The metal is dissolved in hydrochloric or nitric acid. The solution is heated to boiling, and a large excess of strongly alkaline solution of sodium hypobromite added; it is then boiled for some minutes and filtered through a plug of cotton. The precipitate of iron and nickel (with any cobalt and manganese) peroxides is dissolved in dilute hydrochloric acid and the filter well washed; the precipitation by hypobromite is repeated (to separate any traces of chromium, molybdenum, and vanadium retained by the original precipitate), the precipitate filtered off and thoroughly washed. The iron and nickel can then be separated from one another by means of ammonia. The chromium in the filtrate is reduced by alcohol and precipitated by ammonia, and the molybdenum and vanadium can then be separated by any of the usual methods, preferably that of Carnot (*Comptes rend.* June 27, 1887), as manganese vanadate." E Pozzi-Escot, *Comptes rend.*, 1909, *149*, 1131-1132.—*Journal of the Society of Chemical Industry*, Jan. 31, 1910, p. 92. (A. W.)

Solvent Effects of Ferric and Cupric Salt Solutions on Gold.—"Gold is soluble in hydrochloric acid solutions of iron alum and cupric chloride. If the concentration of the acid or salt be increased, solubility of the gold is increased. The rate of the solubility increases more with an increase in concentration in acid than with an increase in concentration of the salt. This is more noticeable in the case in which a copper salt is used. With a change of temperature from 38°—43° C. to 98°—100° C., the rate of solubility is increased seven times in the presence of an iron salt, and thirty-two times in a cupric chloride solution. Ferric salts act as a solvent in the presence of ferrous salts, but the solubility of gold is decreased if the concentration of the ferrous salt be increased. Hence in precipitating gold by ferrous sulphate, a rather large excess of the reagent should be used, and the mixture allowed to cool before filtering. Tables of solubilities are given. A solution containing 1 grm. of iron as iron alum, and 25 c.c. of hydrochloric acid in 125 c.c. of water dissolved 1 mgrm. of gold in 16 hours at 38—43° C. and 2·90 mgrm. in 185 hours." — W. J. McCaughey, *Journal of the American Chemical Society*, 1909, *31*, 1261-1270.—*Journal of the Society of Chemical Industry*, Jan. 31, 1910, p. 92. (A. W.)

Purification of Water Supplies by the Use of Hypochlorites.—"The water supply of Jersey City was treated during December, 1908 with bleaching powder in amount sufficient to yield 0·03 grain of available chlorine per gallon of water, and since that date with a less amount. Daily tests made during the month mentioned showed that the number of bacteria was reduced from an average of 559 per c.c. in the raw water to an average of 2·7 c.c. in the water after treatment, and that no excess of hypochlorite remained. It was found experimentally that the amount used was also sufficient to kill all members of the *B. coli* group present. Filtration was necessary to remove any natural opalescence in the water, but the bleaching powder treatment rendered the supply bacterially safe at a moderate cost. Instead of bleaching powder, sodium hypochlorite obtained by electrolysing a solution of common salt may be used, the disinfecting effect being the same."—W. P. Mason, *Proc. American Philosophical Society.*, *48*, No. 191. *Chemical News*, 1909, *100*, 321.—*Journal of the Society of Chemical Industry*, Jan. 1, 1910, p. 107. (A. W.)

Analysis of Phosphor Bronze.—"The analysis of phosphor bronze presents some little difficulty owing to the fact that on dissolving the alloy in nitric acid the phosphorus is found with the tin in the insoluble residue, or in the case of alloys rich in phosphorus partly in the tin oxide residue and partly in the copper solution. The tin may be separated from the phosphorus in the insoluble residue by fusing with potassium cyanide, digesting the fused mass with hot water, filtering off the metallic tin, destroying the cyanide in the filtrate, which contains the phosphorus as phosphate, by boiling with hydrochloric acid and estimating the phosphorus as phosphomolybdate. This method, although giving reliable results, is long and somewhat tedious, and the following method, which was found to be reasonably accurate and much quicker, was adopted in analysing most of the latter alloys:—

The phosphor bronze (0·5 to 2 grm. depending on percentage of phosphorus) is weighed out into a 10-oz. beaker, 20 c.c. of cold saturated solution of potassium chlorate. Boil until all the alloy is dissolved more potassium chlorate being added in the form of crystals if necessary, and all chlorine is driven off. The solution is then diluted with about an equal bulk of water, and pure granulated zinc added until all the copper and tin are precipitated and no further deposit is observed on a piece of clean zinc foil placed in the liquid. The deposited metals are then filtered off, washed, transferred to a beaker, dissolved in nitric acid, and the tin determined as SnO_2. The phosphorus is determined in the filtrate as phospho-molybdate in the usual way."—HUDSON and LAW. — *Institute of Metals*, Jan., 1910. (A. McA. J.)

ASSAY OF INDUSTRIAL GOLD ASSAYS.—"Buttons containing from 20% to 40% of silver have a greenish-yellow tint, but, when the silver amounts to 50%, the yellow tint is scarcely perceptible. With more than 50% of silver the buttons are silver-white.

The inquartation is best effected by cupellation with the smallest convenient quantity of lead. The practice of alloying the silver by means of the blow-pipe is to be deprecated.

The results obtained for gold by this indirect method are approximately accurate when due care is exercised, but they are invariably lower than those obtained for the gold determined in a separate assay piece by direct cupellation with parting silver.

The difference is, no doubt, partly accounted for by the fact that, in the latter case the presence of the parting silver facilitates the elimination of the copper, and also helps to reduce the loss of gold.

The actual differences between the two methods obtained with gold alloys of varying standards is given in the following table, compiled from results obtained by the author:—

TABLE XIII. — Comparisons of Results for Gold determined by (1) Single Cupellation, and by (2) Double Cupellation with Parting Silver Direct.

Approximate Standard of Alloys.	Single Cupellation. Gold Direct.	Double Cupellation. Gold Indirect.	Difference. Part per 1,000.
375·0	381·9	380·0	– 1·9
	374·0	372·0	– 2·0
	376·0	374·4	– 1·6
	376·4	375·9	– 0·5
	375·0	372·5	– 2·5
	377·1	376·3	– 0·8
500·0	467·1	467 0	– 0 1
	537·0	536·2	– 0·8
	499·3	497·7	– 1·6
	499·8	497·7	– 2·1
625·0	628·5	627·0	– 1·5
	625·7	623·3	– 2·4
750·0	752·6	751·0	– 1·6
	750·3	749·4	– 0·9
916·6	918 0	917·3	– 0·7
	917·1	916·7	– 0·4

From the experiments tabulated above, it will be evident that, in the hands of an inexperienced assayer, the results obtained for the determination of the gold and silver by the double cupellation method may be far from accurate, and, since the losses and gains previously detailed are dependent on so many conditions, it is always necessary, in all cases where extreme accuracy is desired, to use checks. This method of double cupellation, or 'Parting Assay,' in which the gold and silver are determined in the same assay piece, is largely used by trade assayers for the valuation of lemel bars, etc., and, in the hands of experienced assayers who have ascertained the best working conditions for the particular class of alloys with which they have to deal, the results are sufficiently accurate for commercial purposes.

It is not the usual practice of trade assayers to use checks except in special cases, but it may be remarked that the increasing demand for greater accuracy in trade assays is causing checks to be more frequently used than was formerly the case. A check, consisting of a piece of plate of known composition and of similar standard to the alloy being assayed, will meet the requirements in most cases. The results recorded above show to what extent the method is reliable when checks are not used."—E. A. SMITH.—*Institute of Metals*, Jan., 1910. (A. MCA. J.)

METALLURGY.

MEASUREMENT OF PULP AND TAILING.—"In wet-crushing mills, concentrators, and hydro-metallurgical works it is often desirable to measure the water and ore handled, either the amounts contained in tanks or the quantities passing in a given time. In some mills such measurements are systematically made, but in others the amounts are merely guessed, or they are measured once and ever after assumed to remain constant. Discrepancies between 'theoretical' and 'actual' recovery are due to errors in sampling and assaying the material before treatment, added to the corresponding errors affecting the material after treatment, and multiplied by errors in the estimate of tonnage treated. The last item is therefore fully as important as the others in calculating probable returns.

Commenting upon South African practice G. A. and H. S. Denny remark: 'The returns from any given mine from month to month, and the averages which are used as comparisons, actually refer to tonnages which may vary as much as 10 per cent.'[*]

The measurements most frequently required are: tonnage of sand collected in the tanks, tonnage of slime in suspension in vats or agitators, and flow of pulp or crushed ore—the last often involving measurements of water as well as of solid matter, or the ratio of water to solid, and of time.

Next to accepting the metric system in its entirety, it is most convenient for practical purposes to adopt as units the ton of 2,000 pounds, the cubic foot, and the fluid ton of 32 cubic feet—the last being the volume occupied by 2 000 lb. of water, on the assumption that a cubic foot of water weighs 62·5 lb. The fluid ton is a unit that has been tacitly assumed and used for years in connection with cyanide work, but I believe that it has not been thus specifically defined.[†]

As it frequently happens that certain measurements can be more conveniently made at mines in troy and metric units, a sufficient number of conversion factors are given in an annexed table.

Throughout this paper the word 'density' is applied to the density or specific gravity of the dry, solid material under consideration, while 'gravity' or 'specific gravity' is applied to that of the pulp or mixture of water and solid. This arbitrary distinction is made simply to avoid confusion, which sometime arises when the one term is applied indiscriminately.

---

[*] *Journal of South African Association of Engineers*, June 1906.

[†] Since this paragraph was written W. A. Caldecott (this *Journal*, Oct. 1908) has indirectly defined the fluid ton in the same way.

Conversion Table.—Units.—

1 cub. ft. ... ... =1,728 cub. in.
=28,340 cc. (gm. of water).
=62 5 lb. water=7·5 U.S. gal.
=6·25 imp. gal.
1 cub. in. ... ... =16·4 cc. =0 03617 lb. water.
1 litre .. ... =1,000 cc. =0·0353 cub. ft.
=2·2046 lb. water.
=0·2645 U.S. gal.
1 fluid ton ... ... =32 cub. ft. =2,000 lb. water.
=907 litres=0·907 cub. metre.
=240 U.S. gal. =200 Imp. gal.
1 Imp. gal ... ... =10 lb. water=0 005 fluid ton.
1 U.S. gal. ... ... =8·33 lb. water=0·1333 cub. ft.
=3775·5 cc. =$\frac{5}{6}$ Imp. gal.
=230·4 cub. in. approx.
1 ton ... ... =2,000 lb. =907 kg.
=29.166·6 oz. troy.
1 lb. .. ... =453·6 gm. =14·583 oz. troy.

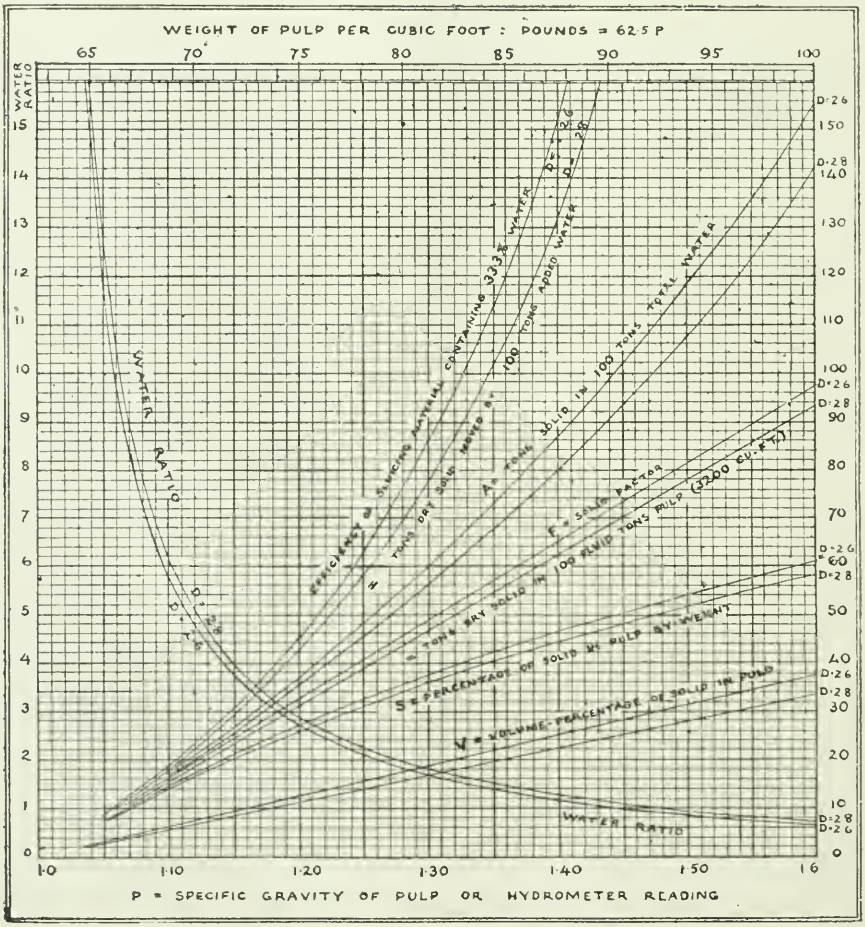
Ratios Involving Time.—

1 cub. ft. per min. =45 fluid tons per day.
1 cub. in. per min. =$\frac{5}{6}$ cub. ft or 0 8333 cub. ft. or 6 25 U.S. gal. per day.
1 cub. ft. per sec.... =2,700 fluid tons per day.
=40 miner's inches.
1 cub. in. per sec.... =50 cub. ft. per day.
1 U.S. gal. per min. =6 fluid tons per day.
1 Imp. gal. per min. =7·2 fluid tons per day.
1 lb. per sec. ... ... =43·2 tons per day.
1 ton per day... ... =1·3889 lb. per min.
=0·02315 lb. per sec.
1 fluid ton per day =0·1666 U.S. gal. per min.
=0·1389 Imp. gal. per min.
=0 0222 cub. ft. per min.
=0·0148 miner's inch.
1 miner's inch ... =1·50 cub. ft. water per min.
=67 5 fluid tons per day.
=0·025 cub. ft. water per sec.
1 litre per sec. ... =95·24 fluid tons per day.
1 gm. per sec. ... =190·5 lb. per day.
=0·09524 tons per day.

Miscellaneous—

1 lb. per cub. ft. ... =16 gm. per litre.
Specific gravity .. =grams weight per cubic centimetre.
=kilograms per litre.
=tons weight per fluid ton of 32 cub. ft.
=lb. weight per cub. ft. × 0·016.
=lb. weight per U.S. gal. × 0·12
=lb. weight per Imp. gal. × 0·1.

Tonnage of Sand in Vats.—The tonnage of sand in vats filled by settling under water is best ascertained by means of boxes of stout sheet iron (conveniently



Ratios of solid and liquid in pulp containing solids of density 2·6 and 2·8.

made of exactly one cubic foot capacity, but in any case accurately measured, having a number of small perforations in the bottom and provided with handles. Several of these are placed in the vat at various stages of the filling, and are allowed to remain throughout the treatment. While the vat is being discharged these are carefully removed and 'struck' level; the contents are then dried and weighed, giving the pounds of dry solid per cubic foot. Several charges should be thus tested and averaged to obtain a constant value for the ore or tailings treated. It is desirable to place some of these boxes near the centre and others near the periphery of the vat, so as to represent variations in horizontal as well as vertical distribution. The mean weight per cubic foot and the volume of sand in the charge give the total weight of sand. While rectangular boxes are often used, a cylindrical form is preferable, as being less liable to deformation.

The Leupold sand-sampler is a short sheet-iron cylinder open at the ends, one of which is provided with a cutting edge and the other with handles.* During the the discharge of a vat this is thrust into the sand, the lower end being closed by a sharpened steel plate introduced from the side. The sand thus removed is dried and weighed; then, the volume of the cylinder being known, the weight per cubic foot is determined. a round tin can, having the closed end punctured to allow the air to escape, can be used in the same way with fairly accurate results.

According to Caldecott † the tailings on the Rand have a density of 2·7, and a ton of 'collected' sand occupies 21·5 cub. ft.; of 'leached' sand, 24 cub. ft.; and transferred sand, 26 cub. ft.; These correspond respectively to 93, 83·3, and 76·8 dry pounds per cubic foot.

Volume of Vats.—In estimating the cubic content of a round vat several diameters should be measured (preferably three or four making equal angles with each other at about the middle depth), and for the greatest accuracy a similar set of measurements should be made near the bottom and another near the top, the arithmetic mean of all the diameters measured being used in the computation. In supposedly cylindrical wooden vats of over 25 ft. diameter, differences of 6 in. or more may be found, due to imperfect construction, to settling, or to unequal shrinkage of the staves, owing to their upper portions being intermittently dried while the lower ends remain wet.

If D be the internal diameter, and H the depth, in feet, of a cylindrical tank, the volume is $0·7854\ D^2H$ cub. ft., $0·024544\ D^2H$ fluid tons, $5·89\ D^2H$ United States gallons, or $4·908\ D^2H$ imperial gallons.

In measuring such a vat an error of one inch in the mean diameter makes a difference of 0·13 DH cub. ft. in the apparent volume—causing an error of about 40 cub. ft. in a vat 40 × 8 ft., or 65 cub. ft. in one 50 × 10 ft.

Datum Marks.—In vats usually filled to about the same depths it is convenient to establish a permanent mark, such as a nail or cleat, at a point near the average level of the surface of the sand or solution, and to determine exactly the volume or capacity when filled to that datum. A correction is then calculated for an inch variation in depth; this can be instantly applied when the deviation of level from the datum has been measured with a rule.

For instance suppose a round vat with 9 ft. staves and averaging 38 ft. 9 in. diameter is commonly filled to about 8 ft. deep with sand, averaging 83·3 lb. dry per cub. ft. The bottom area is 1,179·3 sq. ft., which gives 1179·3 × 83·3 ÷ 2,000 = 49·12 tons per sand per foot of depth, or nearly 400 tons for 8 ft. For exactly 400 tons the datum mark must be placed at a point 400 ÷ 49·12 or 8·14 ft. from the bottom. Also 49·12 ÷ 12 = 4·1 tons, or approximately 4 tons, which is the correction to be added or subtracted for every inch deviation from the mark established.

Interstitial Space.—The interstitial space in settled sands, or in slime filter-cake is sometimes important. It varies with the mode of settling, a given sand settled under water having more voids than the same sifted dry into a vat, while either is reduced by vibration during the process of filling. Maximum interstitial space is attained when material in a moderately moist condition is introduced by shovelling or by a suitable distributor, as is well known to those who have practised the Plattner process of chlorination in vats.

If a vat filled with tailings contains n per cent. total interstitial space, then the speed of the solution percolating through it will be $\frac{100}{n}$ times the 'leaching rate' observed at the surface. For instance, if the leaching rate, or fall of surface of the solution covering the charge of tailing, is two inches per hour, and the interstices form 40% of the volume, then the speed of the solution descending through the charge is $\frac{100}{40} \times 2$, or 5 in. per hour. At this rate a charge of sand 8 ft. or 69 in. deep will take approximately 19 hours to drain; also, if wash-water is run upon such a charge of sand while it is saturated with solution, the strength of the effluent solution will drop rapidly after about 19 hours, provided the material is uniformly distributed.

In settled sand the interstices may range from about 45 to 55%: uniformity of grain and freedom from fine material tending to give the higher value. Spheres of equal size, however, if packed as closely as possible, leave only 26% per cent. voids; if of two or more selected sizes, an even less percentage may be obtained; but such conditions do not occur in practice. Fine slime, collected in a filter press or by settling, may show as high as 75% interstitial space when determined in a wet condition. Such high results are probably due to films of moisture held by capillarity between the surfaces of the particles.

Interstitial Space.

W = gm. material in 100 cc. (including moisture).
  = lb. per cub. ft. × 1·6.
S = percentage of solid, if material is wet.
d = density of dry solid.
$k = \frac{100d}{d-1}$

In Dry Material:

(1) Percentage of voids or interstices

$$= 100 - \frac{W}{d}$$

In Wet Material:

Percentage of actual voids or air-space =

$$100 - W\left(1 - \frac{S}{100}\left(\frac{d-1}{d}\right)\right) \quad (2)$$

$$= 100 - W\left(1 - \frac{S}{k}\right) \quad (3)$$

Percentage of total interstices, occupied by air and water

---

[*] H. Leupold, this *Journal*, Nov. 1904. A core-sampler, for estimating the tonnage of slime in collecting vats, is described in the same paper.

[†] 'Capabilities of circular tanks,' this *Journal*, Oct. 1908.

$$(4) \qquad = 100 - \frac{W\ S}{100\,d}$$

Percentage of volume occupied by water

$$(5) \qquad = \frac{W \times \text{percentage moisture}}{100} = \frac{W\ (100 - S)}{100}$$

For instance, a tailing consisting of quartz and clay has a true density of 2·5. A cubic foot of the settled material contains 80·75 lb. dry solid. The percentage of voids is therefore :

$$\text{by (1) } 100 - \frac{80{\cdot}75 \times 1{\cdot}6}{2{\cdot}5} = 48{\cdot}32\%$$

The same material, settled and partially drained, weighs 95 lb. per cub. ft. in the moist condition, in which state it is found to contain 15% moisture or 85% solid matter. It is required to find the percentage of interstices occupied by air and by water.

This is done by substituting the given values in formulas (3) and (4), the value of k for density 2·5 being given as 166·7.

$$S = 85\%. \qquad W = 95 \times 1{\cdot}6.$$

$$\text{By (3), } 100 - (95 \times 1{\cdot}6)\ (1 - \frac{85}{166{\cdot}7}) 25{\cdot}52\% \text{ dry air space.}$$

$$\text{By (4), } 100 - \frac{(95 \times 1{\cdot}6) \times 85}{100 \times 2{\cdot}5} = 48{\cdot}32\% \text{ total interstices.}$$

The difference 48·32 – 25·52 = 22·80% space occupied by water.

The last result can be arrived at directly :

$$\text{By (5), } \frac{(95 \times 1{\cdot}6) \times 15\%}{100} = 22{\cdot}80\% \text{ space occupied by water.}$$

Slime Collected in Filter Presses.—The capacity of a filter press is most accurately determined by blowing the charge as nearly dry as possible before opening, then selecting a certain proportion of the frames at equal distances from end to end of the press, weighing the entire contents of each separately, and taking an individual moisture sample from each frame tested. The dry weight of slime in each is separately calculated and the average multiplied by the number of frames.

Another plan is to measure the total volume of water issuing from the press during filling, or to estimate it from an average of the flow, from several of the outlets. The water ratio (R) of the incoming slime-pulp must be known, and also the water ratio (M) of the cake formed, where

$$M = \frac{\text{percentage of water in cake}}{\text{percentage dry solid.}}$$

Then the following relation holds good :

$$\text{Tons solid retained in filter} = \frac{\text{Tons effluent water}}{R - M}$$

For instance, if a 3 : 1 sludge is pumped into a press and 28 tons of water filter out, while the cake formed contains 40% water, whence $M = \frac{40}{60} = 0{\cdot}666$; the slime filtered out weighs when dry $\frac{20}{3 - 0{\cdot}666} = 12$ tons.

When the contents of the press are sluiced out, or transferred to a vat or agitator, the weight of charge may be estimated in the first case by a measurement of the flow, or in the latter by measurement of the slime-pulp after thorough mixing by agitation.

Material collected by filters of the Moore type may be estimated by either of the last three methods, or by means of a 'pilot' or test frame of the same dimensions as those of the basket, which is treated in parallel with them and then weighed and sampled for moisture, each of the other frames being assumed to have collected an equal load."—W. J. Sharwood. *The Mining Magazine.*—Nov. '09., p. 226. (J. A. W.)

MINING.

Tributing.—"Tributing to-day is used in different senses in different localities; in the following notes I mean this term to apply to any party of practical miners who have agreed to work either the claims they have pegged or a mine which is fully equipped, and has been leased to them by the directors for a period of years on a royalty basis, with the option of purchase within that period at a stated price. This is in direct opposition to the system of tributing in Cornwall, where the individual miner or party take a pitch for a short time, but where the company continue to work the mine.

Although this subject may be summed up in one word, in reality it embraces every detail connected with the theoretical and practical profession of mining.

There are many so-called tributors, but of these few possess the qualities which lead to success; for in order to ensure good results one must have a thorough grasp of the working details of at least one or two departments of this intricate business, besides being up to date and able to adapt oneself to modern ideas and altered circumstances, and appreciate the advantages they carry. To be a hard worker is no recommendation without being capable of working in the right direction. For instance, the old-time prospector, who digs and delves and roughs it with the best, and spends much time in cutting trenches, but who is unacquainted with the first laws of geology, and can do nothing with a theodolite, nor supply precise data as to the dip of the reef and tonnage of ore in sight, or draw a plan of his work, is disappearing from the mining world as fast as the old sailing ships are leaving the sea. So the tributor, unless he is skilled—highly skilled—and a well-trained, practical man, can never hope to obtain a return for his capital and the labour he spends in this direction.

This is a business which cannot be lightly undertaken; it involves much calculation, besides a careful checking of all results. It is not sufficient that the tributor is competent; his assistants must also be men who are well up to their work. To crush a few tons of ore and carry the resulting gold to the bank sounds quite an easy performance, but when it is intended to carry this out on a profitable basis it will be found that many hitherto unthought-of details must come into the estimates before success can be attained.

The labour question naturally comes first. Under this heading Mr. R. Hamilton, in the address printed in *The Mining Journal*, May 1, writes: 'I have laid stress upon the advantages accruing to the workmen from the contract system, without dwelling upon the obvious advantages to the employer. Experience all the world over has shown that while there are many conscientious men who under any circumstances give their best services to their employer, it is impossible while human nature remains as it is to expect in the aggregate the same amount of work and intelligence from wages men as from contractors. I know it is urged against the contract system that it drives a man to put forth his utmost energies, or even to overtax his physical strength, but, after all, very few men die from overwork. In the whole course of my experience I have never come across a case where a man has worked himself to death, even on contract.'

While Mr. M. Francke, in his report to the Transvaal Government in September, 1907, 'advocates a more extended use of mechanical appliances—viz., rock drills, and also those used for the transport of ore in stopes.' He is also of opinion that 'the miner's wage be reduced to £15 monthly.'

Now in dealing with the contract system versus wages, it is well known (however fair one may wish to be towards the miner) that for various reasons he does not get through as much tonnage when working for a private individual as when he has a mining company to deal with, and does still more when working on contract.

The contract system encourages competition from the most highly skilled form of labour. The payment of wages often means that the worker, where the tributor is concerned, does not, owing to the impossibility of sufficient supervision, exert himself to the same extent as when engaged by a corporation. Better still would be a system of co-operation, for this scheme having been now thoroughly tested, has proved itself both theoretically and practically.

The next item of importance is the machinery to be used. Often the use of old and obsolete machinery spells ruin; it is a mistake to suppose that because a battery is on offer at an upset price it is likely to prove cheap and economical in the long run. The expense of long stoppages, which are likely to occur, must be considered, also the cost of repairs and renewals, which are always heavy.

It is in this department that the services of a good engineer can save so much in the purchase of suitable second-hand machinery. The acquisition of old boilers without a thorough test and expert advice is a mistake; should the tubes leak, the expense of repairs and unsatisfactory nature of same owing to the high wages a boilermaker earns, as well as possible trouble with the boiler inspector, should persuade one to purchase this portion of the machinery direct from the makers. Much, however, may be done in obtaining other mechanical requirements at the broker's yard, when often they are in quite good condition, and purchasable at a fraction of the price quoted when new.

Experience has shown that the services of experts are not to be despised; they are necessary for the success of an undertaking such as this. A small sum spent in obtaining the services of a consulting engineer, surveyor, sampler, and practical mechanical engineer may eventually save hundreds of pounds.

The stoping width is another item of the greatest importance. Strange though it may appear, it is sometimes cheaper to stope 50 in. than 42 in., even when it is not all reef matter. So much depends on local conditions.

In making an estimate, one must not forget to make due allowance for absorption by battery plates and cyanide plant. It is, of course, taken for granted that a thorough sampling and measurement of the reef has preceded everything. Even then it is a great risk to spend capital on new machinery unless the development is at least a year ahead of the mill. If this condition is complied with, and extraction tests for cyanidation made, nothing is left to chance.

Before entering upon the business aspect, it is interesting to compare the theory and practice of mining as shown by Dr. Voit in his article entitled 'The Genesis of the Rand Gold.' Dr. Voit there states: 'In the microscopic structure there is not the slightest difference to be found in the features of a rich banket or a poor banket.' I would like to say, however, that with the naked eye, when it is trained, bands of rich and poor reef can be quickly discerned. Their features are unmistakable. This, however, is a digression, for this method would not be sufficient; a thorough examination of the mine is desirable, which should be systematic, and all assay values properly checked.

In an article entitled 'A Comparative Survey of Rand Mining,' which appeared in the *Transvaal Leader*, the author speaks of 'the outlook which has followed upon a more business-like system of conducting what is, after all, essentially a business proposition, vaster in its scope, of course, and quite different in kind, but upon economic principles much the same as is the making of candles or the production of coal gas.' In fact, if one member of the party has not had a thorough business training, when it comes to the buying of mine stores and supplies, candles, dynamite, etc., it will most likely be found that inferior goods have been palmed off on the unsuspecting buyer for which he has the pleasure of paying an exorbitant price. The necessity for keeping a complete set of books and working out the daily costs soon becomes very apparent; also, the faculty of buying everything in the cheapest market, which is consistent with the quality supplied. Unless the mine is started on a good business footing, its future success is anything but assured. The rough-and-ready system applicable to the old-time prospector is not to be commended now. Often the development is so small that the future values are quite a speculation, for it is well said that a miner cannot see beyond the end of his pick. If so far the assays are encouraging, it is far better to put in a little development work than to start the battery and trust to Providence.

Another point is that costs per ton on a small mine can never compare with those on established mines. The machinery, for instance, never shows the same amount of efficiency as that in use by old-established corporations, where they have every appliance that mechanical skill can devise for executing repairs, and where much capital is invested in spare parts, renewals, and castings.

Sufficient notice is not always taken of the fact that cash payments mean a liberal discount. Cash in hand is required for emergencies; such things as dykes and poor patches may occur, and cam shafts may break, for which all due provision must be made. It is the details of expenditure which accumulate and handicap the tributor. As one cannot expect to reduce the expenses to the level of the companies working on a large scale, it is always wise to make a liberal allowance, and to estimate the costs of development on a liberal basis and the returns on conservative lines.

It would no doubt be of great benefit to the mining community generally if one could specify the amount of capital which would be useful or necessary for this particular branch of mining work, but really this depends so much upon the locality in which the property is situated, the condition of the machinery, the width and value of the reef, water supply, etc., and also whether an equipped mine is to be taken over, which the company, owing to lack of capital and inability for various reasons to raise more, have decided to lease to the would-be tributors, or whether claims are to be worked on which no machinery has been placed, and only a few prospecting shafts have been sunk. This is certainly the crux of the question; but again, information on this subject which is only approximate can, I fear, serve no useful purpose.

In estimating costs it must be borne in mind that dealers cannot give the same terms to the small purchaser as to the buyer for a big combine. While many tributors have reaped handsome profits, this is accounted for by the fact that they have seen the advantage of working on the lines herein mentioned; there are, unfortunately, many instances of failure, and many reasons for it. In one case known to the writer the work was begun without any systematic sampling. Two assays only were made, and on this basis a five-stamp battery was started, and a small KCN works. Besides natives, five white men were employed, the chief being an amalgamator with a slight knowledge of machinery, but who had never been down a mine. This was a foredoomed failure, and showed what so many tributors lack—viz., a want of thoroughness and attention to business detail, a want of faith in the mysteries of sampling and assaying, a lofty contempt for calculations of any kind, a burning desire to purchase and make use of old and obsolete machinery because it is cheap, and a general disregard of the heavy expenses incurred in the cost of repairs and long stoppages. Had a little foresight been shown, and careful business-like supervision been accompanied by some technical advice, the project would never have been embarked upon; the venture, in fact, like many others, was an instance of trusting to luck.

Plant and Development.—It is sometimes argued that tributors have no expenses under the heading of plant or development. This, however, I maintain is incorrect; the plant must be renewed and kept in running order, spare parts are frequently wanted, and improved machinery must be put in. Development must also be kept up if the agreement is to be carried on for any length of time.

Tributors have every working expense a large concern has, except that head office expenses, including claim licenses, are left out; but against this they usually pay a royalty. It is true they have a smaller capital to deal with on which to pay dividends; but, as a rule, you can take it that no sweeping reductions can be made. A company which just pays expenses and no more might pay a tributor, although it could never make a proportionate return to the shareholders. On the other hand, the tributor cannot buy so well; but as an offsett he works harder, and, having his own interests to consider, pays more attention to details and allows less waste of materials.

It is of the utmost importance that no unnecessary risks be taken, and that all things are as definite as possible. The policy known as picking the eyes out of a mine is a fatal one; it is advisable to begin as you mean to go on. One must always be prepared to meet with dykes and poor patches of reef, and make all due allowances. Nothing can be taken for granted, everything should be checked and worked out, for it is the unexpected that happens, at least as often in mining as in any other business. It is well to know the richness or poverty of the stopes, and have more than one stope working, so that the reef may be averaged. It is sometimes said that the best method of checking the mill is to place a shovel below the discharge point of each battery feeder once an hour, which gives a fair average sample of the ore being crushed.

The system so often practised in Rhodesia of working the mine as long as the development lasts and then seeking pastures new does not commend itself to one who is more ambitious, or is desirous of carrying on this work in a similar way to which it is conducted on the Rand—viz., by equipping, working, and developing a mine fairly, with an ultimate view of purchasing for flotation

It is well known that the chief objection to tributing, which applies equally to the lessor and lessee, is from the lessor's point of view that the worker will pick out all the rich patches, do nothing towards the upkeep of the plant, and abandon the mine at his convenience. The lessee also knows that it is a project of great uncertainty, which may swallow up his small capital or return him a fair income. It is against this system I wish to write. If the scheme were placed on a fair business footing, and it was stipulated that a certain amount of development be carried out; that the tributor be a practical man with some capital, and that he be held responsible for the up-keep of the property, both the surface machinery and the underground workings, many small capitalists who are practical men might be induced to invest and to work mines which, were it not for their efforts, would remain idle In that case the directors would not run the risk of making a total loss, but would have prospects of making some small return to their unfortunate shareholders.

This, of course, means that careful and systematic inquiry must be made on both sides before any agreement is signed. It may be the means of opening up a new industry, and would, one imagines, in many cases be preferable for shareholders to waiting, and paying claim licences for years, with no ultimate prospect of obtaining fresh capital or working the property.

Now one may well ask, how, then, does a tributor make his profit if the companies have tried and failed? The reply would be, 'because the enormous outlay required for the purchase of claims and machinery and expense of opening up the mine by shaft sinking, etc., which seldom amounts to less than £100,000, on which interest has to be paid, is avoided. Secondly, though his running expenses, as shown, are practically the same, he requires less profit to yield him a handsome return on this system of co-partnership. Few mines have less capital than £200,000; on this sum a clear return of £5,000 per annum would be useless for the payment of dividends, while this amount divided among, say, five white men, whose floating capital, perhaps, for contingencies is £1,000, would yield a handsome return for their labour.

If, however, claims which have only been prospected are pegged and worked, the reef must be correspondingly richer and wider, to allow for the increased capital outlay. Prospects similar to these are being carried out and worked on the Rand successfully to-day; but, owing to the lack of publicity, the industry as carried on in this direction is severely limited. From the company's point of view, this system means that the machinery is kept in repair and running order, as also is the mine. The expense of a caretaker and staff is saved, and there is always a possibility of a sale of their assets, or even a fresh discovery.

In considering, however, any proposition with a view to tributing, the five leading questions may be summed up as

(1) What are the working expenses likely to amount to?

(2) What amount of development has been done?

(3) What is the water supply?

(4) What do the reef values average?

(5) Is the gold free or refractory?

If these are satisfactorily answered, the question of investing capital in the venture has been settled. Could the existing tributors only be induced to send in a record of their working expenses they would stand to lose nothing and the mining world would gain much, as these figures would then form a basis from which would-be investors could form an estimate; for tributing is a business which has great potential possibilities, and in it lies a remedy for much of the idleness from which skilled labour is, for various reasons, now suffering."—*The Mining Journal*, June 5, 1909, p. 695. (C. B. S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

JOINT REPORT ON THE BITUMINOUS OR OIL SHALES OF NEW BRUNSWICK AND NOVA SCOTIA, also on the Oil Shale Industry of Scotland. Part I., Economics. Part II., Geology.—By R. W. ELLS, L.L.D. Canada, Department of Mines. Nos. 55 and 1107.

This report is an account of work undertaken by the Mines Department to ascertain the economic value of the known deposits of oil shales in the Counties of Albert and Westmorland, New Brunswick. A complete test on 50 tons of oil shale from the former county being carried out at the Pumpherston Works in Scotland. The results of this test showed that the oil shales were very satisfactory and will doubtless be worked on the large scale. In addition to this work, which is strictly of local value only, the report includes what is practically an up-to-date description of the oil shales of various countries their mining, retorting, and distillation. This will be found useful in many respects though a further treatment of many of the sections would have enormously increased its utility and might have proved an incentive to the research. Still deemed by the author to be necessary in bringing about the "improvement in the processes and wastes" which "still present themselves" (page 57). We should, however, be wanting in gratitude if we did not express our thanks to the authors for the glimpses given to us of a large modern oil shale works such as the celebrated one which is described, and more especially for the beautiful reproductions of the plant in actual use. (J.A.W.)

Science in Modern Life. A Survey of Scientific Development, Discovery, and Invention, and their relations to Human Progress and Industry. Prepared under the editorship of J. R. Ainsworth Davis. Vol. V. 4to., pp. viii. 207. *Gresham Publishing Co.*

Lewis, E. T. Quantitative Analysis Table for use on the bench on Four Cards; Varnished with Eyelet Holes and Rings. *Cambridge University Press.* Net 2s. 6d.

Fernali, H. H., Smith, C. D., and Grine, H. A. Incidental problems in Gas Producer Tests. (U.S. Geological Survey Bulletin.) Diagrams. 8vo. sd., p.p. 29. *W. Wesley.* Net 1s. 6d.

Kelvin, Lord. Mathematical and Physical Papers. Vol. IV. Hydro dynamics and General Dynamics. 8vo., p.p. 580. *Cambridge University Press.* 18s.

Randall, D. and Weeks, A. The Smokeless Combustion Coal in Boiler Plants. With a Chapter on Central Heating Plants. (U.S. Geological Survey Bulletin.) Engravings. 8vo., sd. *W. Wesley.* Net 2s.

Strong, R. M. Commercial Deductions from Comparisons of Gasoline and Alcohol Tests on Internal Combustion Engines. (U.S. Geological Survey Bulletin.) 8vo., sd., p.p. 38. *W. Wesley.* Net 1s. 6d.

Abstracts of Patent Applications.

(C.) 192/09. William Philip. Safety device for reversible winding and hauling plants. 20.4.09.

This application is for a device to prevent overwinding in the operations of winding on mines by means of mechanism actuating an alarm when the reversing gear is in the wrong position for starting up.

(C.) 127/10. Edwin Burt. Process of filtration. 24.3.10.

This application refers to an improved process for filtration by means of a rotatable shell lined with suitable filtering medium. The shell is partly filled with the pulp to be treated, then further pulp is forced in against the pressure of the imprisoned air. By this means the slime cake formed adheres closely to the filtered surface and can be easily washed or subjected to further treatment without re-pulping.

(C.) 361/09. Frank Ondra (1), Alfred Adair (2). Improvements relating to the treatment of zinc solutions for the recovery of metals or metal compounds therefrom. 6.8.09.

This application relates to the treatment of zinc solutions from cyanide works, for the recovery of the contained metals, consisting of—

1. Precipitation of copper and gold by the addition of copper and iron sulphates.
2. Removal of impurities such as iron manganese, by adding soda carbonate and chloride of lime and agitating with air.
3. Recovery of the zinc by precipitation with calcic hydrate, sodium sulphide or hypo sulphite.

(C.) 330/09. Hans Charles Behr. Improvements in pipes for conveying corrosive or abrasive matter. 15.7.09.

This application refers to the use of earthenware or similar linings for pipes conveying abrasive or corrosive liquids, and shows special methods of fixing and jointing same within the outer pipes.

(C.) 346/09. A. J. Arbuckle. Improvements relating to vats, tanks or vessels for separating solids from liquids. 24.7.09.

This patent relates to a device comprising a spiral blade, screw or worm and the means for rotating the same within a mass of settled material. It is applicable to vats or vessels employed for thickening ore pulp or separating ore in a granular or pulverulent condition from liquids.

The device is placed near the outlet of the vessel or tank (which is represented in the drawing as a conical tank) and can be rotated within the settled mass of solids in order to raise or lower it therein and thus increasing or diminishing the size of the passage or passages for the solids. Any desired adjustment can be made while the apparatus is in operation.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) means provisional specification, and (C) complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)

(P.) 191/10. Arthur Wilson Ferns. Improvements in acetylene gas generators and lamps. 27.4.10.

(P.) 192/10. James Chalmers Johnstone. Improvements in rollers for the underlay of conveyor belts. 27.4.10.

(P.) 193/10. Edward Harker Vintcent Melvell. Instrument for surveying and like purposes. 27.4.10.

(P.) 194/10. Benjamin Waites (1), George Ritchie Lambie (2), Ivor Howell Williams (3). Improvements in means for grinding or reducing ore or other material. 28.4.10.

(P.) 195/10. James Oliver Bartle Improvements in or relating to the axles and bearings of trucks and similar vehicles. 28.4.10.

(P.) 196/10. Tullio Bubola. Improvements in apparatus for electro-chemical treatment of ores. 28.4.10.

(C.) 197/10. Luossavaara Kurunavaara Aktiebolag (1), Henry Johan Hjalmar Nathorst (2), Gothard Lundstrom (3). Improved method and means for forming rock and like drills. 28 4.10.

(C.) 198/10. Donald Barns Morison. Improvements in or relating to steam engine condensing and vacuum producing apparatus. 28.4.10.

(P.) 199/10. Alexander McNamara. Improvements in or relating to rotary drilling or boring apparatus. 29.4.10.

(P.) 201/10. John Inshaw Rodway (1), James Esson (2). Improvements in means for automatically enflating the tyres of vehicle wheels while in motion. 29.4.10.

(P.) 202/10. Bertram Gray. Improvements in pipe linings 29.4.10.

(P.) 203/10. Thomas McCraw. Improvements in tube mill linings. 30.4.10.

(P.) 206/10. J A. G. Harrison. Harrison's sliding notch sluice gates. 3.5.10.

(P.) 208/10. John Gardner (1), George Henry Nolan (2). An improved pump valve and valve seating for use in mines and other works. 6.5.10.

(P.) 209/10. John Johnstone. Improvements in apparatus for receiving drills and issuing tokens and checks against delivery thereof. 6.5.10.

(P.) 210/10. Albert Holdsworth. Improvements in centrifugal pumps. 6 5.10.

(P.) 211/10. Elisha Tippell. Improvements in or relating to rock drilling machines for removing the detritus or cuttings from the holes formed therewith. 6.5.10.

(P.) 214/10. John van Nostrand Dorr. Improvements in decanting apparatus. 6.5.10.

(P.) 215/10. Johann Virch. Improvements in or relating to the construction of ceilings, roofs, walls, columns and the like of concrete. 6.5.10.

(P.) 216/10. William Charles Stephens. Improvements in rock drills. 6.5.10.

(P.) 252/10. James Edward Thomas. Improvements relating to feeding means for tube mills and like grinding apparatus. 1.6.10.

(P.) 253/10. William Arthur Caldecott. Improvements relating to candles. 1.6.10.

(P.) 255/10. Thomas Henry Chod (1), James Patrick (2) An improvement in the means of igniting fuses for blasting and analogous purposes. 1.6.10.

(P.) 256/10. William Berthelson. An improved pressure cock for taking off connections for rock drills and the like. 1.6.10.

(P.) 257/10. William Beaver. Improvements in means for separating the fine product from ores, particularly applicable for conveyor belts. 1.6.10.

(C.) 258/10. Ernest Graham Godfree. Electro mechanical selector. 2.6.10.

(C.) 259/10. Carl Emil Egner (1), Johan Gunnar Holmstroom (2). Improvements in cooling devices for telephone transmittors for strong current. 3.6.10.

(C.) 260/10. John Prue Karns. Improvements in drilling machines. 3.6.10.

(C.) 261/10. John Prue Karns. Improvements in cutter heads for drilling machines. 3.6.10.

(P.) 262/10. Archibald Charles Hughes. Improvements in rock-drilling machine bits, hand drills and the like. 4.5.10.

(P.) 263/10. Wilmont Elevin Clark. Improvements in filtering processes and apparatus. 4.6.10.

(P.) 264/10. Augustus Carron. Improvements in the treatment of wood, canvas, and similar materials to render the same fire resistant and water and vermin repellant. 4.6.10.

(P.) 265/10. Archibald McArthur Johnston. Combustible composition utilisable for fuse lighting. 7.6.10.

(P.) 266/10. William Thomas. Improvements in linings for tube mills and the like. 8.6.10.

(P.) 267/10. David Gilmour. Improvements relating to the linings of tube mills. 8.6.10.

(C.) 268/10. Maurice Brukner. Improved reinforced concrete railway sleepers. 8.6.10.

(P.) 269/10. Herbert Charles Fisher (1), Alfred George Newkey Burden (2). Improvements in and appertaining to the mortar boxes of stamp mills. 8.6.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Adams, H. A., *to* Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg.

Allen, H. E., *to* Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp.

Cameron, L. D., *l/o* W. Australia; Daxymusu Dev Co., Dunkwa, via Sekondi, West Africa.

Cronin, J., *l/o* Johannesburg; Abels Kop G. M. Co., Schweizer Reneke.

Glyn, C., *l/o* Germiston; City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.

Hahn, W. A. A., *l/o* German South West Africa; Survey Department, Rand Mines, Ltd., P. O. Box 1056, Johannesburg.

Johnson, A. D, *l/o* Bulawayo; Kimberley (Mashonaland) G. M. Co., Ltd., Kimberley Reefs, Rhodesia.

King, A. C., *to* 39, Great Brunswick St., Turffontein.

Lofts, H. F., *to c/o* W. Leyson, Esq., Matokos, Salisbury, Rhodesia.

McConnell, Thos., *to* Rosetta Mine, Barberton.

Melvill, G., *l/o* Fordsburg; Bantjes Consolidated Mines, Ltd., P. O. Box 2, Florida.

Meyer, C. E., *l/o* East Rand; Imperial Mine, Louws Creek, near Barberton.

Minards. W. I. K., *l/o* Knights; East Gwanda Mines, Ltd., West Nicholson, Rhodesia.

Pollitt, R. B., *l/o* Mexico; c/o Messrs. Gudgeons, Peacock & Prentice, Stowmarket, Suffolk, England.

Reece, F. B., *Journals to* c/o F. A Reece, Sunningdale, New Brighton, Cheshire, England.

Stokes, Ralph, *l/o* Johannesburg; Crown Mines, Ltd., P. O. Box 102, Fordsburg.

Strange, E. H., *l/o* Michigan; The Elms, 1517 Glenarm St., Denver, Colo., U S.A.

Thorne, T. L., *l/o* Roodepoort; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.

Walker, J. F., *l/o* East Rand; Consolidated Main Reef Mines, Ltd., P. O. Box 2, Maraisburg.

INDEX.

INDEX TO VOL. X., 1909-1910.

Compiled by the Secretary (Mr. Fred. Rowland, F.C.I.S.)

Those portions in Italics refer to Original Notes and Papers read before the Society.

PAGE

A

Abel, Heat Test of Guncotton and Nitroglycerine, Significance of the R. Robertson and B. J. Smart .. 373
Abrecht, K.—Rapid Analysis of Aluminium 147
Absorption of Carbon Dioxide ... 63, 176
Abstracts, Chemistry 21, 69, 115, 145, 184, 213, 258, 292, 320, 373 468
—— Metallurgical 23, 73, 117, 148, 186, 215, 259, 295, 322, 374, 406 469
—— Mining 30, 75, 118, 152, 190, 226, 263, 300, 332, 377, 411 472
—— Miscellaneous 36, 77, 155, 193, 227, 267, 301, 336 415
—— of Patent Applications 196, 231, 271, 306, 339, 379, 419 475
Accidents due to Electricity in British Coal Mines 226
Accounts, Statement of ... 422, 425
Accumulated Slime, the Treatment of, and the Use of Filter Presses for Clarifying Slime Solution and By-Products J. D. O'Hara 342
Discussion 403
Acetylene Lamps Underground W. F. Slaughter 152
Acid Washes, Experiments on the Assay of, resulting from the Cyanide "Clean-up" by the use of Bisulphate L. J. Wilmoth 136
Discussion 138, 211
do Reply to 290
Erratum 196
Acids and Alkalies, Standardization of E. Rupp 116
Adair, A.—on Some Experiments on Smelting Titaniferous Iron Ore 212
—— Usher Process 344
Address, Inaugural, President's (A. McA. Johnston) 3
Addresses, Changes of 40, 80, 120, 160, 196, 232, 272, 308, 340, 380, 420, 476
Advantages of Sand Filling 431
Agitating in W. Australia 223
Agitation and Aeration, Cyanide... ... 325
Agriculture, Application of Electricity to Dr. G. Kapp 303
Aimetti, S.—Notes on Classifiers 20
Air Cost of Machines 280
Air Lift Pumps for Underground Work, Notes on the Use of B. C. T. Solly 2
—— Mine, Moisture in 191
—— *Moist, Influence of, on Quicklime James Gray Discussion* 19, 62, 106, 144
Reply to Discussion 176
Albrecht, C. F.—Electricity in the Coal Mining Industry 334
Aldehydes and Ketones 445
Alexander, Fraser, The Treatment of Black Sands 174
Discussion 175

PAGE

Alexander, Fraser.—on Endless Rope Haulage 319
—— —— on the Tube Mill Circuit and Classification 398
Alkalies and Acids, Standardization of E. Rupp 116
Alkaline Picrates 446
Allen, Robt., Welcome to 162
Alloy, Nickel-Copper, New D. H. Browne 28
Alloys, Electro-Metallurgical, New Applications of A. D. Jouve 70
Alumina, Direct Volumetric Method for the Estimation of W. H. Seamon ... 293
Aluminium. R. B. Woodworth ... 336
—— Rapid Analysis of K. Abrecht ... 147
Alvarez, Piñerúa y.—Rapid Method for the Estimation of Vanadium in Ores and Technical Products 321
Amalgam, Silver, Rapid Economical Method of Making E. H. Croghan ... 42
Amalgamating Plates in the Stamp-Mill, The Position of 186
Amalgamation and Plates 262
—— Electro-Chemical E. E. Carey 26, 148
—— *Effect on, of different intervals of time between the dressings of Plates G. O. Smart Discussion* 141
Reply to Discussion 177
—— in the United States and Mexico .. 259
American Cyanide Practice S. F. Shaw ... 328
Analyses of Colonial Oats J. Lewis ... 301
—— *of Gases from Burning Nitro-Glycerine Explosives. Wm. Cullen and D. W. Greig* 90
Discussion 91, 210
do Reply to 241
—— *Grading, and their Application H. Stadler* 382
Discussion 390, 457
Analysis of Phosphor Bronze Hudson and Law 468
—— Thermal, and Intermetallic Compounds 25
Anderson, W. T.—on Sand Filling on the Witwatersrand 432
Andrews, A. J.—South African Coals and their Economics Discussion ... 92
Reply to Discussion 95
Annual Dinner 341, 459
—— Report of Council 422
Antidote, Cyanide, A Note on the Efficiency of the A. L. Edwards 82
Antifume Blasting Gelatine 90
Antimony, Chung Yu Wang (Review) ... 119
—— New Method for Determining M. F. Coolbaugh and J. O. Betterton ... 147
—— and Tin, Separation of G. Panajotow 116, 294
Application of Electricity to Agriculture Dr. G. Kapp 303
Applications of Concrete in the Metal Mining Industry H. W. Edwards ... 415
Argol 145

PAGE

Arsenic, Precipitation of, by Hydrogen Sulphide L. L. de Koninck ... 69
Arsenical Ores 189
Ashley, H. E.—Theory of the Settlement of Slime 149
Assay of Acid Washes resulting from the Cyanide "Clean-up" by the use of Bisulphate, Experiments on the L. J. Wilmoth 136
Discussion 138, 211
do Reply to 290
Erratum 196
—— *of Cyanide Solutions and Slime Residue Carrying Dissolved Gold A. Whitby* 134
Discussion 211, 288
do Reply to 289
—— of Dressed Tin Ores L. Parry ... 258
—— of Industrial Gold Assays E. A. Smith 469
—— *of Tin Ores Jas. Gray* 312
Discussion 402
—— —— —— Wet. J. J. Beringer ... 376
—— Values, Bullion and 310
Assaying, Behaviour of Tellurium in S.W. Smith 146
—— *Routine, on a Westralian Mine W. B. Blyth Reply to Discussion* ... 95
—— Solutions 96
Assays, Copper, Notes on the Iodide Method for R. T. D. Williams ... 71
—— Crucible A. A. Steel 145
—— Elutriated Slimes... 102
—— *Grading and Grinding Efficiencies A. Yates Correspondence* ... 67
Reply to Discussion 99
—— Ore Concentration 187
Associates, New 2, 41, 81, 121, 161, 197, 233, 273, 309, 381, 422
Atkin, A. J. R.—on Grading Analyses and their Application 458
Atomic Theory, Suggestions for a New Dr. J. Moir Reply to Discussion ... 96
Austin, L. S.—Metallurgy of the Common Metals (Review) 379
Ayers, G. F., and Cullen, Wm.—A Method for the Recovery of Zinc from Solutions of Sulphate 87
Discussion 143, 209
do Reply to 240

B

Balance Sheet, 1909-1910 425
Bailes, G. M.—Modern Mining Practice (Review) 338
Ball Mills 222
Ballarat System of Lead Mining 377
Bannister, C.O., & Stanley, W. N.—Thermal Properties of Cupels 145
Barberton District, Geology of 123
Barberton Goldfield, The A. Richardson ... 122
Discussion 179, 210
do Reply to 288
—— —— Early History of ... 122, 179
Barrett's Berlin Mine 127, 210
Battery, Efficiency Value of Ore Going to ... 100
—— Plates, Dressing of E. H. Martin 187
—— —— Scaling of J. E. Breakell .. 151
Bauerman, Hilary (Obituary) .. 234, 256
Bayly, C. F.—on the Tube Mill Circuit and Classification 320
Behaviour of Tellurium in Assaying S. W. Smith 146

Bell, R. N.—A Selective Electric Fuse Spitting Device 153
Belts, Driving 224
Beneficio de Metales de Plata y Oro for Cianuracion F. McCann (Review) 270
Benoni Consolidated Reduction Plant .. 408
Beringer, J. J.—Wet Assay of Tin Ores .. 376
Bermann, M.—Simple Identification Test for Iron and Steel 326
Bernewitz, W. M. von.—Practical Notes on Dry Crushing Mills 222
Bettel, W.—The Fixation of Atmospheric Nitrogen in Cyanide Manufacture ... 185
Betterton, J. O., & Coolbaugh, M. F.—New Method for Determining Antimony ... 147
Bevington, R. G.—on Congratulations to W. A. Caldecott 162
—— on Endless Rope Haulage 201
—— on Grading Analyses and their Application 457
—— on Tube Mill Circuit and Classification ... 288
Bismuth, Metallurgy of 189
Bisulphate, Experiments on the Assay of Acid Washes resulting from the Cyanide "Clean-up" by the use of L. J. Wilmoth 136
Discussion 138, 211
do Reply to 290
Erratum 196
—— *Solutions, Specific Gravity of Thos. Donaldson* 86
Black Sand Treatment, The Use of the Monteju for 344
—— *Sands, The Treatment of Fraser Alexander* 174
Discussion 175
Blankets as Concentrators 263
Blast Furnace, Ferro-Manganese in the O. Hohl 327
—— —— Smelting, Roasting for .. 266
Blomfield, A. L.—Smelting Gold and Silver Slime 406
Blundun, B. V.—on the Tube mill Circuit and Classification 401
Blyth, W. B.—Routine Assaying on a Westralian Mine Reply to Discussion ... 95
Bolts in Stamp Mills 261
Bonding New to Old Concrete E. P. Goodrich... 156
Books, New, and Reviews, 37, 78, 119, 157, 195, 229, 270, 304, 336, 379, 418 475
Borates, Simple Tests for 269
Borax, Effect of 145
—— as a Flux, The Use of 95
Bosse, F. J.—New Concrete 36
Bosqui, F. L., Welcome to 162
Braden, W.—Cost of Mining in Chile ... 378
Bradford, R. H.—Temperatures during Cupellation 330
—— Wager, The Late... ... 2, 21
Brandenburg, R.—Estimation of Free Lime in Cement 322
Breakell, J. E.—Scaling of Battery Planks ... 151
Briquetting 266
Bromo-Cyanide Treatment 108
Bronze, Phosphor, Analysis of Hudson and Law 468
Brown, A. Selwyn—Modern Cyanide Practice and Machinery 324
—— Gilmour E.—on the Development of Heavy Gravitation Stamps ... 331
—— F. C.—Tube Mill Liners 151

PAGE
Browne, D. H.—New Nickel-Copper Alloy .. 28
—— E.—on the Barberton Goldfield ... 210
Brunton, D. W.—Modern Progress in Mining and Metallurgy in the Western United States 263
Bullion and Assay Values 310
—— Copper, Gold in F. F. Hunt ... 70
—— *Gold Precipitates and, The Smelting of, with Oil Fuel A. Yates Discussion* 144
Reply to Discussion 393
Bürker, K.—Chemical Methods for Distinguishing Carbon Monoxide Haemoglobin from Oxyhaemoglobin ... 292

C

Calculations in Ore Concentration Dr. R. Gahl 187
Caldecott, Dr. W. A.—on the Barberton Goldfield 211
—— Capacity of Circular Vats per foot of Depth 407
—— Congratulations to ... 162, 276
—— *Continuous Collection of Sand for Cyaniding* 43
Discussion 54, 142 179
do Reply to 238
—— *The Development of Heavy Gravitation Stamps* 108
Discussion 144, 178, 215, 241, 331 353
do Reply to 358
—— Replying to Congratulations 162, 276
—— on the Treatment of Ore Slime, etc ... 176
Campbell, C. W. R.—At Annual Dinner ... 467
Cams, Designs of 245
Camshafts, Designs of 248
Canadian Mining Reports (Review) .. 158
Capacity of Circular Vats per foot of Depth W. A. Caldecott 407
—— of Sinking Pumps 226
Carbon and Cellulose, Notes on the Precipitating Effects of Substances containing various forms of, on Cyanide Solutions containing Gold and Silver A. J. Clark and W. J. Sharwood ... 234
Discussion 405
—— *in Lime, Experiments on the Precipitation of Gold from Cyanide Solution by E. H. Croghan* 391
Discussion 393
—— in Steel, Determination of .. 70
—— Monoxide Haemoglobin from Oxyhaemoglobin, Chemical Methods for Distinguishing K. Bürker 292
—— *Monoxide in Mine Gas, The Estimation of Dr. E. H. Weiskopf Reply to Discussion* 11
Carey, E. E.—Electro-Chemical Amalgamation 26, 148
Carrier, C. F.—Manufacture of Sodium Peroxide and Sodium Cyanide .. 115
Carter, Palmer—on Mining at the Redjang Lebong Mine 318
Case Hardening of Steel with Gases Olsen & Weiffenbach 416
Castner Process, The 115
Cause and Prevention of Rust 23
Cellulose, Absorption of Gold and Silver by .. 236
Cement, Estimation of Free Lime in R. Brandenburg 322
—— Process, Sinking a Shaft by the Dobbelstein 227
Centre Marks 262

PAGE
Cerium, Rapid Volumetric Method for Determining, in Presence of other Rare Earths F. J. Metzger 294
Changes in Dilute Solutions of Permanganate J. E. Purvis 292
—— of Addresses 40, 80, 120, 160, 196, 232, 272, 308, 340, 380, 420, 476
Chaplin, F. D. P.—At Annual Dinner ... 462
Charcoal, Making, in Pilgrims Rest District 17
Cheffirs, J.—Notes on Different Methods of Stoping 301
Chemical Methods for Distinguishing Carbon Monoxide-Haemoglobin from Oxyhaemoglobin K. Bürker ... 292
Chemistry Abstracts 21, 69, 115, 145, 184, 213, 258, 292, 320, 373, 468
—— of Cyanide Solutions J. E. Clennell (Review) 379
—— of Zinc Smelting V. Lepiarczyk ... 184
Chile, Cost of Mining in W. Braden ... 378
—— Mill, The M. R. Lamb 150
Chiltern System of Lead Mining 377
Chlorination 267
—— in the United States 259
Christy's Cell 22
Chrome Iron Ore 195
Chromite Ore from the Transvaal... ... 258
Chromium and Nickel in Steels, Separation of Vanadium, Molybdenum E. Pozzi-Escot 468
Clancy Process for Treatment of Ore ... 300
Clark, A. J.—Precipitation from Cyanide Solutions by Zinc Shavings and Dust Reply to Discussion 205
—— *and Sharwood, W. J.—Notes on the Precipitating Effects of Substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing Gold and Silver* ... 234
Discussion 405
Clark, D.—Gold Refining (Review) ... 338
Classification, The Tube Mill Circuit and G. O. Smart 282
Discussion 320, 397, 452
do Reply to 454
Classifiers, Cone 452
—— —— Diaphragms in 239
—— Grading Results from 287
—— *Notes on S. Aimetti* 20
—— Single Cone 283
—— Tube Mill 284
Clean-up and Precipitation in W. Australia ... 224
—— in Rhodesia, The 84
Clennell, J. E.—The Chemistry of Cyanide Solutions Resulting from the Treatment of Ores (Review) 379
Circular Vats, Capacity of, per Foot of Depth W. A. Caldecott 407
City Deep Mill, The 299
Coal and Coke, Sulphur in M. Holliger .. 69
—— Determination of Volatile Gases in ... 298
—— Dust Explosions 35
—— Mines, Electricity in R. Nelson ... 36
—— —— Temperature at Great Depths in .. 118
—— Mining, Electricity in G. R. Wood .. 190
—— —— Industry, Electricity in the F. C. Albrecht 334
—— Moisture in 298
—— Occluded Gases in... 298
—— Sampling 298
Coals, South African, and their Economics A. J. Andrews Discussion .. 92
Reply to Discussion 95

PAGE
Coaton, A. A.—on Stope Measurements ... 63
Cobalt, Determination of Nickel in Presence of J. A. Sanchez 258
Cobbe, H. N. G.—Welcome to 162
Coke and Coal, Sulphur in M. Holliger ... 69
— Smelting in S. Africa 163
Coloured Wrappers for Explosives W. Cullen 82
Collection of Sand for Cyaniding, Continuous W. A. Caldecott 43
Discussion 54, 142 179
do Reply to 238
Collett, E., and Eckhardt, M.—Estimation of Molybdenum in Molybdenum Glance 321
Collimation, To Adjust the Line of .. 228
Collieries, Haulage in 34
Collings, B. I. Small Mines of Rhodesia Reply to Discussion 140
Combustion, Heat of, and Explosion Temperature of Explosives O. Poppenberg and E. Stephan 321
—— Tubes, Quartz A. G. Levy ... 70
Commans, R. E.—on the Development of Heavy Gravitation Stamps .. 219
Committee on Fuel A. W. Warwick ... 298
Comparison of Costs With and Without Tube Mills 355
Computation of Crushing Efficiency of Fine Grinding Machines C. O. Schmitt 374
Comstock Mines, Ventilating System at the Prof. G. J. Young 411
Concentration 265
—— of Ores by Wet Methods 28
—— Ore, Calculations in Dr. R. Gahl ... 187
Concentrating Machinery, Water Required for 23
Concentrators, Blankets as 263
Concrete Beams, Test of Large Reinforced ... 157
—— Bonding New to Old E. P. Goodrich 156
—— Foundations for Stamp Mills .. 111
—— in the Metal Mining Industry, Applications of H. W. Edwards ... 415
—— for Shaft Tracks 415
—— New F. J. Bosse 36
—— The Use of, for Mine Support Prof. W. R. Crane 36, 118
Conicle Pebble Mill H. W. Hardinge ... 24
—— —— —— The Hardinge 27
Construction and Operation of the Stamp-Mill G. H. Fison 261
Continuous Collection of Sand for Cyaniding W. A. Caldecott 43
Discussion 54, 142, 179
do Reply to 238
Contributions and Correspondence 20, 65, 181, 212, 406
Conversion Tables 470
—— Table, Temperature Dr. L Waldo ... 417
Conveyors and Elevators in W. Australia ... 224
Coolbaugh, M. F., and Betterton, J. O.—New Method for Determining Antimony... 147
Coolidge, R. F.—White Precipitate found in Zinc Boxes 184
Coombe, M. H.—on Further Notes on Rand Mining 395
Copper Assays, Notes on the Iodide Method for R. T. D. Williams ... 71
—— Bullion, Gold in F. F. Hunt ... 70
—— Handbook for 1908 (Review) ... 157
—— Simple Tests for 268
—— Smelting, Reverberatory 266
Correspondence and Contributions 20, 65, 181, 212, 406
Corresponding Members of Council ... 426
—— —— —— for Rhodesia 234
Corrosion, Notes on, with Special Reference to the Corrosion of Steel Winding Ropes M. Thornton Murray 54
Discussion 59, 204
do Reply to 205
—— of Reinforced Steel E. R. Matthews 228
—— Protecting Steel from R. B. Woodworth 336
—— Test for Determining Durability or Resistance to 213
Cost of Mining in Chile W. Braden ... 378
—— Underground Work 155
Council, Annual Report of 422
—— Attendances of Members of 424
—— Corresponding Members of 426
—— —— —— for Rhodesia 234
—— Election of New 428
Crane, Dr. W. R.—Index of Mining Engineering Literature.. 229
—— Prof. W. R.—The Use of Concrete for Mine Support 36, 118
Croghan, E. H. Experiments on the Precipitation of Gold from Cyanide Solution by Carbon in Lime 391
Discussion 393
—— on the Assay of Tin Ores 402
—— *Rapid Economical Method of making Silver Amalgam* 42
Cropper, C. H—on Researches upon the Telluride Gold Ores of Cripple Creek 20
Crosscutting and Drifting 153
Crosse, Andrew F.—on the Assay of Tin Ores 315
—— on Bullion and Assay Values ... 311
—— on Cyanide in Sands 309
—— on the Destruction of Cyanide ... 449
—— on Experiments on the Assay of Acid Washes 138
—— on Inaugural Meeting of S.A. Institute of Electrical Engineers 233
—— on Metallurgical and Mining Terms ... 198
—— on a Method for the Recovery of Zinc from Solutions of Sulphate ... 143
—— on Notes on Corrosion, &c. 205
—— on a Rotary Extractor for Precious Metals from Solutions 204
—— on Some Experiments on Smelting Titaniferous Iron Ore ... 172, 211
—— on Specific Gravity of Bisulphate Solutions 87
—— on Test for Prussic Acid 342
—— the Use of Standards in Reading Gold Pannings 117
—— *Treatment of Ore Slime with an addendum on the Treatment of Black Sands* 172
Discussion ... 175, 254
do Reply to 318
Crowe, T. B. (edited by).—Researches upon the Telluride Gold Ores of Cripple Creek
Discussion 19, 107
Correspondence 179
Reply to Discussion 207
Crucible Assays A. A. Steel 145
Crushing Efficiency of Fine Grinding Machines The Computation of C. O. Schmitt 374
Crystallization of Iron and Steel, The A. McA. Johnston 3
Cullen, W., Coloured Wrappers for Explosives 82
—— *The Gases resulting from the Use of High Explosives Reply to Discussion* .. 11
Addendum 43

PAGE
Cullen, W., and Ayers, G. F.—A Method for the Recovery of Zinc from Solutions of Sulphate 87
Discussion 143, 209
do Reply to ... 240
—— *and Greig, D. W. Analyses of Gases from Burning Nitro-Glycerine Explosives* 90
Discussion 91, 210
do Reply to 241
Cupellation Losses Dr. T. K. Rose .. 146
—— Temperatures during R. H. Bradford 330
—— *Tests for H. A. White* 234
Cupels, Thermal Properties of C. O. Bannister and W. N Stanley ... 145
Cupric and Ferric Salt Solutions on Gold, Solvent Effects of W. J. McCaughey 468
Cyanicides, Application of 448
Cyanidation... 267
—— of Gold, Recent Progress in the Mr. Nugel 258
Cyanide Agitation and Aeration 325
—— Alkali, Estimation of Sulphide in T. Ewan 293
—— *Antidote, A Note on the Efficacy of the A. L. Edwards* 82
—— *"Clean-up," Experiments on the Assay of Acid Washes resulting from the, by the use of Bisulphate L. J. Wilmoth* 136
Discussion 138, 211
do Reply to 290
Erratum 196
—— *The Destruction of Dr. J. Moir and Jas. Gray* 433
Discussion 442
—— Double, of Potassium and Zinc ... 258
—— Effect of Varying Iron and Alkali on .. 436
—— Extraction of Gold from Tailings containing Pyrites 235
—— Manufacture, The Fixation of Atmospheric Nitrogen in W. Bettel ... 185
—— *Plant, Small, as Erected and Worked in Rhodesia, A Sketch of the F. J. Thomas* 82
Discussion 207
do Reply to 394
—— Plants, Small, Filtering Gold Slime on a 25
—— Poisoning 36
—— Potassium, Electro-Chemistry of Solution of Gold in Ekeley & Tatum .. 21
—— Practice, American S. F. Shaw ... 328
—— —— and Machinery, Modern A. Selwyn-Brown 324
—— Process Imlay 325
—— Sodium, Manufacture of Sodium Peroxide and C. F. Carrier 115
—— *Solution, Experiments on the Precipitation of Gold from, by Carbon in Lime E. H. Croghan* 391
Discussion 393
—— —— of Gold in 228
—— *Solutions containing Gold and Silver, Notes on the Precipitating Effects of Substances containing various forms of Carbon and Cellulose on A. J. Clark and W. J. Sharwood* ... 234
Discussion 405
—— —— *Precipitation from, by Zinc Shavings and Dust A. J. Clark Reply to Discussion* 205

PAGE
Cyanide Solutions and Slime Residue, The Assay of, carrying Dissolved Gold A. Whitby 134
Discussion 211, 288
do Reply to 289
—— in Sands 309
—— Tanks, Galvanised Iron 83
—— Treatment of Slime L. M. Green and L. Mennell 322
Cyanidation of Manganese Silver Ores of Mexico E. M. Hamilton 65
Cyaniding, Continuous Collection of Sand for W. A. Caldecott 43
Discussion 54, 142 179
do Reply to 238
—— in America M. R. Lamb 296
—— Graphitic Ores 23
—— in the United States and Mexico ... 259
—— Ways of Preventing Losses in ... 296

D

Datum Marks 471
Davies, John—(Obituary) 69
Decennial Index 310
Decomposition of Sulphates in an Air Current W. S. Landis 322
Deep Mining 30
Denny, H. S.—on the Development of Heavy Gravitation Stamps 217
De-rusting of Iron in Ferro-Concrete — Rohland 23
Destruction of Cyanide, The Dr. J. Moir and Jas. Gray... 433
Discussion 442
Determination of Nickel Prettner ... 72
—— —— in Presence of Cobalt J. A. Sanchez 258
—— of Vanadium H. F. Watts ... 294
Development of Heavy Gravitation Stamps W. A. Caldecott 108
Discussion 144, 178, 215, 241, 331, 353
do Reply to 358
—— Work .. : 332
Dewar, Sir James—Light Reactions at Low Temperatures 374
Diamond Drilling Problem, A Solution of a C. M. Haight 192
Diaphragms in Cone Classifiers 239
Die Diamantführenden Gesteine Süd Afrikas Dr. Ing. P. A. Wagner (Review) ... 305
Dies and Shoes 262
—— Wear of 114
Dicyandiamidine, Use of, for the Estimation of Nickel, and its Separation from Cobalt and other Metals H. Grossman and W. Heilbron 321
Dilution System of Slime Treatment ... 322
Dimethylglyoxime, Estimation of Nickel in Nickel-Steel by means of A. Iwanicki 293
Direct Filling in Rhodesia 84
—— Replacement System 264
—— Volumetric Method for the Estimation of Alumina W. H. Seamon ... 293
Discharge Area, Screens and 262
—— Height of 262
Dissolving Gold-Bearing Zinc 86
Distribution of Gold Produced on the Rand Dr. T. K. Rose Reply to Discussion 139
Dobbelstein.—Sinking a Shaft by the Cement Process 227
Dolomite, Analyses of 170

PAGE
Donaldson, Thos.—on a Method for the Recovery of Zinc from Solutions of Sulphate 143
— — *Specific Gravity of Bisulphate Solutions* 86
Dorr Thickeners 323
Dowling, W. R.—on Congratulations to W. A. Caldecott 162
— on the Continuous Collection of Sand for Cyaniding 179
— on Development of Heavy Gravitation Stamps 170
— on Inaugural Address 10
— on the Influence of Moist Air on Quicklime 63, 144
— on President's Return 274
— on Some Experiments on Smelting Titaniferous Iron Ore 170
— on Welcoming Guests and Members ... 162
Dredging, Gold 265
Dressing of Battery Plates E. H. Martin 187
— of Minerals, The H. Louis (Review) 37, 79
Drill Machines, Rock, Work done by ... 155
— Sharpening Furnaces Underground ... 281
Drilling, Diamond, Problem, Solution of a C. M. Haight 192
Drills, Rock, Dust Collector for M. Remaux 191
Driving Belts 224
— Breast 280
— Record at the Modderfontein "B" G. M. Co. 227
— Small Drifts and Loading Blast Holes G. C. McFarlane 152
Drop and Speed in Mills 262
Drucker, A. E.—Gold Mining in Korea ... 225
Dry Crushing Mills, Practical Notes on W. M. von Bernewitz 222
Duncan, Patrick—at Annual Dinner ... 464
Dust Collector for Rock Drills M. Remaux 191
Dyason, E. C.—Welcome to 276

E

Eckhardt, M., and Collett, E.—Estimation of Molybdenum in Molybdenum Glance 321
Edwards, A. L.—A Note on the Efficiency of the Cyanide Antidote 82
— H. W.—Applications of Concrete in the Metal Mining Industry 415
Effect on Amalgamation of different intervals of time between the Dressings of Plates G. O. Smart Discussion ... 141
Reply to Discussion 177
Efficiency Tonnages 101
Ekeley and Tatum—Electrochemistry of Solution of Gold in Potassium Cyanide ... 21
El Oro, Tube Milling at H. E. West ... 186
Election of Scrutineers... 381
Electric Furnaces in Germany 188
— Fuse Spitting Device, a Selective R. N. Bell 153
— Haulage and Winding Plants M. B. Mountain 34
— Lamps, Miners' W. H. Mills ... 33
— Signalling in Mines C. E. Grayson ... 333
Electrical Smelting of Iron Ore in Sweden ... 299
Electric Transmission 265
Electrically Driven Winding Engine H. H. Weymouth 75
— — — Engines E. K. Scott ... 300
Electricity, Application of, to Agriculture Dr. G. Kapp 303
— in British Coal Mines, Accidents due to, 226

PAGE
Electricity in Coal Mines R. Nelson ... 36
— — Mining G. R. Wood ... 190
— in the Coal Mining Industry F. C. Albrecht 334
Electro-Chemical Amalgamation E. E. Carey 26, 148
— Chemistry of Solution of Gold in Potassium Cyanide Ekeley and Tatum 21
— Magnetic Ore Separation C. G. Gunther (Review) 119
— Metallurgical Alloys, New Applications of A. D. Jouve 70
Electrolytic Precipitation M. R. Lamb .. 29
— Refining of Gold Dr. T. K. Rose . . 148
Elevators and Conveyors in W. Australia ... 224
Ellis, A. Russell.—Formation of Graphite by the Interaction of Magnesium Powder and Carbonates 71
Elutriated Slimes Assays 102
Endless Rope Haulage H. G. Kay ... 198
Discussion ... 254, 291, 319, 404
do Reply to 457
Errata.—Precipitation from Cyanide Solutions by Zinc Shavings and Dust .. 68
— Grading Assays and Grinding Efficiences 105
Escot, E. Pozzi.—Separation of Vanadium, Molybdenum, Chromium and Nickel in Steels 468
Estimated Cost of Shaft Sinking 412
Estimation of Carbon Monoxide in Mine Gas Dr. E. H. Weiskopf Reply to Discussion 11
— of Free Lime in Cement R. Brandenburg 322
— of Molybdenum in Molybdenum Glance E. Collett and M. Eckhardt ... 321
— of Nickel in Nickel-Steel by means of Dimethylglyoxime A. Iwanicki ... 293
— of Sulphide in Alkali Cyanide T. Ewan 293
Evolution of the Gravity Stamp Mill A. del Mar 117
Ewan, T.—Estimation of Sulphide in Alkali Cyanide 293
Experiments on the Assay of Acid Washes Resulting from the Cyanide "Clean-up" by the use of Bisulphate L. J. Wilmoth 136
Discussion 138, 211
do Reply to 290
Erratum 196
— — *Precipitation of Gold from Cyanide Solution by Carbon in Lime E. H. Croghan* 391
Discussion 393
Explosions, Coal Dust 35
Explosives, Coloured Wrappers for W. Cullen 82
— Explosion Temperature of, Heat of Combustion and O. Poppenberg and E. Stephan 321
— *High, The Gases resulting from the use of W. Cullen Reply to Discussion* 11
Addendum 43
— *Nitro-Glycerine, Analyses of Gases from Burning Wm. Cullen and D. W. Greig* 90
Discussion 91, 210
do Reply to 241
— Testing Station 92

PAGE
Extractor, A Rotary, for Precious Metals from Solutions W. D'A. Lloyd and E. T. Rand 201
Discussion 255
do Reply to 373

F

Feed Water, Mill 263
Ferric and Cupric Salt Solutions on Gold, Solvent Effects of W. J. McCaughey 468
Ferro-Concrete, De-rusting of Iron in Rohland 23
—— Manganese in the Blast Furnace O. Hohl 327
Ferrous Iron in Magnetite R. B. Gage ... 69
Filling, Sand-, on the Witwatersrand E. Pam 429
Discussion 432
Filter Presses, Details of, in America ... 330
—— —— *the Use of, for Clarifying Slime Solution and By-Products, the Treatment of Accumulated Slime and J. D. O'Hara* 342
Discussion 403
—— Pressing in W. Australia. ... 75, 223
—— Slime, The Hunt Continuous ... 295
Filtering Gold Slime E. Jensen .. 25
Fine Grinding 325
—— —— Machines, The Computation of Crushing Efficiency of C. O. Schmitt 374
Fines in Ore 100
Fires, Mine 378
First Aid to the Injured and Sick (Review) ... 418
Fison, G. H.—Construction and Operation of the Stamp Mill 261
Fixation of Atmospheric Nitrogen in Cyanide Manufacture W. Bettel ... 185
"Flatback" System of Stoping, The ... 301
Flushing, Mine 118
Fluxes for Silver Residues 24
Fluxing Charges, Directions for 145
Formaldehyde 445
Formation of Graphite by the Interaction of Magnesium Powder and Carbonates H. R. Ellis 71
Formula for Determining Tonnoge of Sand Fed to Tube Mill 456
Foundations of Batteries 243
Framing of Stamp Mills 261
Friend, J. Newton.—The Theory of Valency (Review) 78
Frue Vanners, Water required for ... 23
Fuel, Committee on A. W. Warwick . 298
—— The Direct and Indirect Saving of 93, 94
Fuels, Low Grade, The Gas Engine and ... 156
Fume Recovery 267
Furnace, The Tilting 297
Furnaces, Electric, in Germany 188
Further Notes on Rand Mining Tom Johnson 276
Discussion, ... 319, 394
do Reply to 449
Fuse Lighter, A Simple, Safe 413
—— Spitting Device, A Selective Electric R. N. Bell 153
Fusibility of Mixtures of Gold and Tellurium H. Pelabon 145
Fusions for True Silica J. Herman ... 320

PAGE

G

Gage, R. B.—Ferrous Iron in Magnetite ... 69
Gahl, Dr. R.—Calculations in Ore Concentration 187
Galvanised and other Zinc-coated Metals, The Testing of Dr. W. H. Walker ... 213
Galvanometer, Saladin Double-mirror ... 25
Gas Engine and Low Grade Fuels ... 156
—— *Mine, The Estimation of Carbon Monoxide in Dr. E. H. Weiskopf Reply to Discussion* 11
Gascoyne, R.—on Further Notes on Rand Mining 394
Gases, Analyses of, from Burning Nitro-Glycerine Explosives Wm. Cullen and D. W. Greig 90
Discussion 91, 210
do Reply to 241
—— *Resulting from the Use of High Explosives W. Cullen Reply to Discussion* 11
Addendum 43
Geological and Archæological Notes on Orangia J. P. Johnson (Review) 418
Geology of Cape Colony A. W. Rogers and A. L. du Toit (Review) 338
Glass, Non-cracking 227
Gold Assays, Assay of Industrial E. A. Smith 469
—— Contents in Oxidised Ore at Frankfurt Mine 16
—— in Copper Bullion F. F. Hunt ... 70
—— Cyanidation of, Recent Progress in the Mr. Nugel 258
—— in Cyanide, Solution of 228
—— *Dissolved, The Assay of Cyanide Solutions and Slime Residue carrying A. Whitby* 134
Discussion 211, 288
do Reply to 289
—— Dredging 265
—— Electrolytic Refining of Dr. T. K. Rose 148
—— Extraction in Korea 226
—— Fine, Saving by Adhesion J. D. Luttrell 28
— .. Method of Estimating Weight of, in a Quartz Sample 27
—— *Mines, a Suggestion relating to the Valuing of R. A. Wade* ... 122
Correspondence 183
—— Mining in Korea A. E. Drucker ... 225
—— —— New Method of 378
— *Ores, Telluride, of Cripple Creek, Researches upon the T. B. Crowe*
Discussion 19, 107
Correspondence 179
Reply to Discussion 207
—— Pannings, The Use of Standards in Reading A. F. Crosse ... 117
—— *Precipitates and Bullion, The Smelting of, with Oil Fuel A. Yates Discussion* 144
Reply to Discussion 393
—— *Precipitation of, Cyanide Solution by Carbon in Lime, Experiments on the E. H. Croghan* 391
Discussion 393
—— in Potassium Cyanide, Electro-Chemistry of Solution of Ekeley & Tatum ... 21

PAGE

Gold Produced on the Rand, The Distribution of the *Dr. T. K. Rose* *Reply to Discussion* 139
—— Production, Increase in World's ... 156
—— in Reefs L. de Launay 30
—— Refining D. Clark (Review) ... 338
—— and Silver, Absorption of, by Cellulose... 236
—— —— *Notes on the Precipitating Effects of Substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing* *A. J. Clark and W. J. Sharwood* 234
Discussion 405
—— —— *Slime, Smelting* *A. L. Blomfield* 406
—— Simple Tests for 267
—— Slime at Barrett's G. M. Co., Analysis of 210
—— —— Filtering E. Jensen... ... 25
—— Solvent Effects of Ferric and Cupric Salt Solutions on W. J. McCaughey .. 468
—— and Tellurium, Fusibility of Mixtures of H. Pelabon 145
Goldfield, The Barberton *A. Richardson* ... 122
Discussion 179, 210
do *Reply to* 288
Goodrich, E. P.—Bonding new to old Concrete 156
Gracht, W. A. J. M. v. d.—Limits of Deep Mining 414
Grading Analyses, Application of ... 386
—— —— *and their Application* *H. Stadler* 382
Discussion 390, 457
—— —— Methods of Conducting ... 386
—— *Assays and Grinding Efficiencies* *A. Yates* *Correspondence* 67
Reply to Discussion 99
Graham, K. L.—on the Influence of Moist Air on Quicklime 106
Graphite, Formation of, by the Interaction of Magnesium Powder and Carbonates H. R. Ellis 71
—— Recent Discoveries of, in British African Colonies 147
Graphitic Ores, Cyaniding 23
Graumann, H. (Mayor)—at Annual Dinner ... 467
Gravitation Stamps, Heavy, The Development of *W. A. Caldecott* 108
Discussion 144, 178, 215, 241, 331, 353
do *Reply to* 358
Gray, Jas.—The Assay of Tin Ores ... 312
Discussion 402
—— *The Influence of Moist Air on Quicklime*
Discussion 19, 62, 106, 144
do *Reply to* 176
—— *and Moir, Dr. J.—The Destruction of Cyanide* 433
Discussion 442
—— on Vote of Thanks to Scrutineers .. 429
Grayson, C. E.—Electric Signalling in Mines 333
Great Boulder (W. A.) Mine 23
Green, L. M.—Cyanide Treatment of Slime ... 322
Greig, D. W.—on Some Experiments on Smelting Titaniferous Iron Ore ... 253
—— *and Cullen, W.—Analyses of Gases from Burning Nitro-Glycerine Explosives*... 90
Discussion 91, 210
do *Reply to* 241
Grinding Efficiencies, Grading Assays and *A. Yates* *Correspondence* ... 67
do *Reply to* 99
—— Machines, Fine, The Computation of Crushing Efficiency of C. O. Schmitt 374
—— Pans 222

PAGE

Grossman, H., and Heilbron, W.—Use of Dicyandiamidine for the Estimation of Nickel and its Separation from Cobalt, &c. 321
Guess, H. A.—Ore Dressing in the United States and Mexico 259
Guides in Mills 262
Guncotton and Nitroglycerine, Significance of the Abel Heat Test of R. Robertson and B. J. Smart 373
Gunther, C. G.—Electro-Magnetic Ore Separation (Review) 119

H

Haematite 194
Haight, C. M.—A Solution of a Diamond Drilling Problem 192
Hall, R. N.—Prehistoric Rhodesia ... 157
Hamilton, J. G.—At Annual Dinner ... 459
—— *E. M.—Cyanidation of Manganese Silver Ores of Mexico* 65
—— R.—Western Australian Metallurgy ... 74
—— W. L.—on Some Experiments on Smelting Titaniferous Iron Ore .. 403
Handbook for Field Geologists C. W. Hayes (Review) 157
—— on Tacheometrical Surveying C. Xydis (Review) 157
Harbord, F. W.—on Some Experiments on Smelting Titaniferous Iron Ore ... 170
—— Welcome to 162
Hardinge Conical Mill 27, 82
—— H. W.—Conicle Pebble Mill ... 24
Harker, A.—Natural History of the Igneous Rocks (Review) 270
Haulage in Collieries 34
—— *Endless Rope* *H. G. Kay*... .. 198
Discussion ... 254, 291, 319, 404
do *Reply to* 457
—— and Winding Plants, Electric M. B. Mountain 34
Hayes, C. W.—Handbook for Field Geologists (Review) 157
Heat of Combustion and Explosion Temperature of Explosives O. Poppenberg and E. Stephan 321
—— of Deep Mines, The 191
Heavy Gravitation Stamps, The Development of *W. A. Caldecott* 108
Discussion 144, 178, 215, 241, 331, 353
do *Reply to* 358
Height of Discharge 262
Heilbron, W., and Grossman, H.—Use of Dicyandiamidine for the Estimation of Nickel and its Separation from Nickel, etc 321
Henderson, E. T.—Laboratory Screens ... 73
—— J. W.—A Tunnelling Record in California 192
Herman, J.—Fusions for True Silica ... 320
Hibbert, E.—Volumetric Method for the Estimation of Titanium, and of Titanium and Iron in the Presence of Each Other 293
Hildebrand, J. H.—Purification of Mercury ... 224
Hobart, J. F.—Millwrighting (Review) ... 119
Hohl, O.—Ferro-manganese in Blast Furnace 327
Hoisting, Mine 263
Holes, Blast, Loading and Driving Small Drifts G. C. McFarlane ... 152
Holliger, M.—Sulphur in Coal and Coke .. 69
Holloway, G. T.—Accident to 234
—— Obituary Notice of Prof. H. Bauerman... 256

PAGE

Hooghwinkel, G. H. J.—on the Development of Heavy Gravitation Stamps .. 220
Hoover, H. C.—Principles of Mining (Review) 119
How much Wood in a Cord? 302
Hudson and Law.—Analysis of Phosphor Bronze 468
Hunt, F. F.—Gold in Copper Bullion ... 70
—— Continuous Slime Filter 295
Hunter, R.—Some Notes on Testing Wire Ropes 193
—— S.—Methods of Deep Lead Mining ... 377
Hydraulic Classifiers, Water Required for ... 23
—— Stope Filling at the Robinson Mine ... 300
Hydrogen Sulphide, Precipitation of Arsenic by L. L. de Koninck 69
Hypochlorates, Purification of Water Supplies by the Use of W. P. Mason ... 468

I

Iddings, J. P.—Igneous Rocks (Review) ... 157
Igneous Rocks J. P. Iddings (Review) ... 157
Imlay Cyanide Process.. 325
Improved Tape Reel for Engineers and Surveyors, An 155
Inaugural Address, President's (A. McA. Johnston) 3
Increase in World's Gold Production ... 156
Index of Mining Engineering Literature Dr W. R. Crane 229
Influence of Moist Air on Quicklime Jas. Gray Discussion 19, 62, 106, 144
Reply to Discussion 176
Institution of Mining and Metallurgy ... 426
—— —— —— Standard Laboratory Screens 383
—— —— —— Standardization of Bullion and Assay Values 310
Instruments, Survey, Points on Adjusting L. A. Palmer 227
Inter-metallic Compounds, Thermal Analysis and... 25
Interstitial Space 471
Iodide Method for Copper Assays, Notes on the R. T. D. Williams ... 71
Iron, Cast, Permanent Expansion of, under Moderate Temperature 406
— — De-rusting of, in Ferro-Concrete Rohland 23
— — Ferrous, in Magnetite R. B. Gage ... 69
— — Ore, Electrical Smelting of, in Sweden... 299
— — — — *Smelting Titaniferous, Some Experiments on*
Part I.—Crucible Experiments ... 162
Part II.—Furnace Experiments... 345
Prof. G. H. Stanley Discussion 211, 253, 290, 351, 403
— — Ores, Transvaal, and Steel Production R. N. Kotzé 194
— — and Steel, Composition of ... 4, 5, 6
— — — *The Crystallization of A. McA. Johnston* 3
—— — Simple Identification Test for M. Bermann 326
Iwanicki, A.—Estimation of Nickel in Nickel-Steel by means of Dimethylglyoxime 293

J

James, A.—on the Development of Heavy Gravitation Stamps 220
Jensen, E.—Filtering Gold Slime 25
Jigs, Water required for 23
Jockey, Haulage 198

PAGE

Johnson, C. M.—Rapid Methods for the Chemical Analysis by Special Steels, Steelmaking Alloys, and Graphite (Review) 229
—— E. H.—on Grading Analyses and their Application 390
—— on the Treatment of Ore Slimes, etc. 175
—— J. H.—on the Precipitation of Gold from Cyanide Solution, etc 393
—— on the Treatment of Accumulated Slime, etc 403
—— J. P.—Geological and Archaeological Notes on Orangia (Review) ... 418
—— Ore Deposits of South Africa (Review) 338
—— Tom.—on Endless Rope Haulage 254, 319, 404
—— on S. A. Coals and their Economics ... 92
—— on Stope Measurements 105
—— *Further Notes on Rand Mining* ... 276
Discussion 319, 394
do Reply to 449
—— *White Labour in Mining Reply to Discussion* 14
Johnston, A. McA.—At Annual Dinner 461, 468
—— on Annual Dinner 341
—— on Annual Report 427
—— on the Assay of Tin Ores 315
—— on the Barberton Goldfield 134
—— on Bullion and Assay Values 310, 311, 312
—— on the Continuous Collection of Sand for Cyaniding 142
—— on Cyanide in Sands 310
—— on the Destruction of Cyanide ... 449
—— on Decennial Index 310
—— on the Development of Heavy Gravitation Stamps 82, 144
—— on the Effect on Amalgamation of Different Intervals of Time between the Dressings of Plates 141
—— on Election of New Council 428
—— on the Election of Scrutineers for Annual Ballot 381
—— on Experiments on the Assay of Acid Washes 138, 139
—— on the Gases Resulting from High Explosives 14
—— *Inaugural Address* 3
—— on the Influence of Moist Air on Quicklime 62
—— on the Late Mr. Wager Bradford ... 2
—— Leave of Absence 122
—— on Notes on Corrosion 59
—— on the Precipitation of Gold from Cyanide Solutions, etc. 393
—— on Presentation to Lord Selborne ... 341
—— Reply to Welcome on His Return . 274
—— on Seventeenth Anniversary ... 342
—— on the Small Cyanide Plant as Erected and Worked in Rhodesia 85
—— on Specific Gravity of Bisulphate Solutions 87
—— on Total Membership 422
—— on the Treatment of Accumulated Slime and the Use of Filter Presses for Clarifying Slime Solution and By-Products 345
—— on Welcome to Mr. E. C. Dyason ... 276
—— —— Mr. W. McDermott 274
—— —— Mr. H. G. Nicholls 276
Jolly, H. R.—on the Assay of Cyanide Solutions and Slime Residue carrying Dissolved Gold 288

PAGE
Journal of the Institute of Metals (Review) 418
Jouve, A. D.—New Applications of Electro-Metallurgical Alloys 70

K

Kalgoorlie Ore Bodies 23
Kapp, Dr. G.—Application of Electricity to Agriculture 303
Kay, H. G.—Endless Rope Haulage ... 198
Discussion ... 254, 291, 319, 404
do Reply to 457
Kendall G. M. Co. 184
Kilowatt Hour, Meaning of the 156
King, A.—on a Rotary Extractor for Precious Metals from Solutions 255
Kingposts, Various Types of 250
Koninck, L. L. de—Precipitation of Arsenic by Hydrogen Sulphide 69
Korea, Gold Mining in A. E. Drucker ... 225
Kotzé, R. N.—Transvaal Iron Ores and Steel Production 194

L

Laboratory Screens E. T. Henderson ... 73
Labour in Korea 225
—— *White, in Mining Tom Johnson Reply to Discussion* 14
Lamb, M. R.—the Chile Mill 150
—— Cyaniding in America 296
—— Electrolytic Precipitation ... 29
Lamp Tests, Miner's 34
Lamps, Acetylene. Underground W. F. Slaughter 152
—— Miners' Electric W. H. Mills ... 33
Landis, W. S.—Decomposition of Sulphates in an Air Current 322
Large Units in Mining L. Reyersbach ... 77
Laschinger, E. J.—on Inaugural Address ... 10
—— on Sand Filling on the Witwatersrand... 433
Launay, L. de.—Gold in Reefs 30
Law and Hudson.—Analysis of Phosphor Bronze 468
Lead Mining, Methods of Deep S. Hunter 377
—— Smelting 266
Leaks, Stopping, under High Pressure W. O. Wood 333
Lepiarczyk, V.—The Chemistry of Zinc Smelting 184
—— Zinc Smelting 259
Leslie, C. D.—on the Destruction of Cyanide 449
Level, To Adjust the Horizontal Cross-Hair of 228
Levy, A. G.—Quartz Combustion Tubes ... 70
Lewis, J.—Analyses of Colonial Oats .. 301
Leyson, L. T., and Moody, G. T.—Solubility of Lime in Water 292
Light Reactions at Low Temperatures Sir James Dewar 374
Lighting and Signalling, Mine 264
Lime Contracts 62
—— Estimation of Free, in Cement R. Brandenburg 322
—— Results of Experiments on 63
—— in Water, Solubility of G. T Moody and L. T. Leyson 292
Limes, Comparisons of Results obtained from Blue and White 107
Limits of Deep Mining W. A. J. M. v. d. Gracht 414
Limonite 195
Lindsay, R.—on the Influence of Moist Air on Quicklime 106
Liners, Chilled White Iron 186

PAGE
Liners, Mortar Box 262
—— Ribbed 151
—— Silex 151
—— Tube Mill, Brown's 297
—— —— F. C. Brown 151
Linton, Robt.—Zinc Dust Precipitation at Cerro Prieto Reply to Discussion ... 60
Litharge, Dissolving Action of 146
—— Freezing Point of 146
Lloyd, W. D'A. and Rand E. T.—A Rotary Extractor for Precious Metals from Solutions 201
Discussion 255
do Reply to 373
Loading Blast Holes and Driving Small Drifts G. C. McFarlane 152
Lock. C. G. Warnford.—on Amalgamation ... 26
Losses, Cupellation Dr. T. K. Rose ... 146
Louis, H.—The Dressing of Minerals (Review) 37, 79
Luttrell, J. D.—Saving Fine Gold by Adhesion 28

M

Macaulay, Dr. D.—on Analyses of Gases from Burning Nitro-Glycerine Explosives 91
Machinery, Concentrating, Water required for 23
Machine Air Cost 280
Magnesite, Analysis of local 88
Magnetite 194
—— Ferrous Iron in R. B. Gage ... 69
Mance, J. C.—on Pilgrims Rest Gold Fields... 16
Manganese, Ferro-, in the Blast Furnace O. Hohl 327
—— *Silver Ores of Mexico, Cyanidation of E. M. Hamilton* 65
Manufacture of Sodium Peroxide and Sodium Cyanide C. F. Carrier ... 115
Mapping, Mine 263
Mar, A. del—Evolution of the Gravity Stamp Mill 117
Marriott, H. F.—on the Development of Heavy Gravitation Stamps 218, 221
Martin, E. H.—Dressing of Battery Plates ... 187
Mason, W. P.—Purification of Water Supplies by the use of Hypochlorates ... 468
Matthews, E. R.—Corrosion of Reinforced Steel 228
McCann, F.—Beneficio de Metales de Plata y Oro for Cianuracion (Review) ... 270
McCarthy, E. T.—on the Development of Heavy Gravitation Stamps ... 331
McCaughey, W. J.—Solvent Effects of Ferric and Cupric Salt Solutions on Gold ... 468
McDermott, W.—Address in response to President's Welcome 274
—— on the Development of Heavy Gravitation Stamps 219
McFarlane. G. C.—Loading Blast Holes and and Driving Small Drifts... ... 152
Meaning of the Kilowatt Hour 156
Measurement of Pulp and Tailing W. J. Sharwood 469
Measurements, Stope O. Tonnesen Discussion ... 18, 63, 105, 140
Correspondence ... 183, 212
Reply to Discussion ... 369, 405
Members, New 1, 41, 81, 121, 161, 197, 233, 273, 309, 341, 381, 421
Membership, Tables of.. 423
Mennell, L.—Cyanide Treatment of Slime ... 322
Mercury in Italy 299
—— Losses and Traps 263

PAGE
Mercury Purification of J. H. Hildebrand 224
Metal Ribbon 187
Metallic Tellurides C. A. Tibbals ... 147
Metallurgical Abstracts 23, 73, 117, 148, 186, 215, 259, 295, 322, 374, 406, 469
—— and Mining Terms 197
—— Practice, Rand E. J. Way ... 408
Metallurgy of Bismuth 189
—— of the Common Metals L. S. Austin (Review) 379
—— of the Common (non-Ferrous) Metals (Review) 158
—— and Mining in the Western United States, Modern Progress in D. W. Brunton 263
—— W. Australian R. Hamilton ... 74
Metals, Precious, A Rotary Extractor for, from Solutions W. D'A. Lloyd and E. T. Rand 201
Discussion 255
do Reply to 373
—— Precious, Treatment of Ores of ... 300
—— Zinc-coated. The Testing of Galvanised and Other Dr. W. H. Walker ... 213
Method of Estimating Weight of Gold in a Quartz Sample 27
—— *for the Recovery of Zinc from Solutions of Sulphate, A Wm. Cullen and G. F. Ayers* 87
Discussion 143. 209
do Reply to 240
Methods of Deep Lead Mining S. Hunter 377
—— of Resuscitation 303
Metzger, F. J.—Rapid Volumetric Method for Determining Cerium in Presence of other Rare Earths 294
Mexico, Ore Dressing in the United States and H. A. Guess 259
—— Tube Mill Practice in 226
Meyer, C. E.—on a Method for the Recovery of Zinc from Solutions of Sulphate ... 209
Mill, The Chile M. R. Lamb 150
—— The City Deep 299
—— Conicle Pebble H. W. Hardinge ... 24
—— —— —— The Hardinge 27
—— Feed Water 263
—— Parts for Various Weights of Stamps, Table showing Weight and Size of ... 245
—— Pulp Classification 20
—— Stamp, Construction and Operation of the G. H. Fison 261
—— —— Evolution of the Gravity A. del Mar 117
—— —— The Position of Amalgamating Plates in the 186
Milling in W. Australia 222
Mills, Details of Stamps in Modern America 329
—— Dry Crushing, Practical Notes on, W. M. von Bernewitz 222
—— W. H.—Miners' Electric Lamps ... 33
Millwrighting J. F. Hobart (Review) ... 119
Mine Air, Moisture in 191
—— Fires 378
—— Flushing 118
—— Hoisting 263
—— Lighting and Signalling 264
—— Locomotives 335
—— Mapping 263
—— Samples, Checks of 95
—— Sampling Device 192
—— Support, The Use of Concrete for Prof. W. R. Crane ... 36, 118
Mine Valuation R. Stokes 183
—— Ventilation 265
Mineral Tests, Simple. and How to Make Them G. Surr 267
Miners' Electric Lamps W. H. Mills ... 33
Miner's Lamp Tests 34
Mines, Deep. The Heat of 191
—— Electric Signalling in C. E. Grayson 333
—— Trials Committee's Standard Grades ... 383
Mining Abstracts 30, 75, 118, 152, 190, 226. 263, 300, 332, 377, 411, 472
—— Areas on the Rand, Valuation of W. Fischer Wilkinson 31
—— in Barberton District 129
—— in Chile, Cost of W. Braden .. 378
—— Deep 30
—— —— Lead, Methods of S. Hunter . 377
—— —— Limits of W. A. J. M. v. d. Gracht 414
—— Gold, New Method of 378
—— Industry, Metal. Applications of Concrete in the H. W. Edwards ... 415
—— Large Units in L. Reyersbach ... 77
—— and Metallurgical Terms 197
—— and Metallurgy in the Western United States, Modern Progress in D. W. Brunton 263
—— *Methods, The Pilgrims Rest Gold Fields and J. Moyle Phillips Discussion* 16
Reply to Discussion 139
—— and Milling in Korea, Cost of ... 225
—— Reports 302
—— *Rand, Further Notes on Tom Johnson* 276
Discussion 319, 394
do Reply to 449
—— *at the Redjang-Lebong Gold-Silver Mine, Sumatra H. Philp* 315
Discussion 318
—— Surface 263
—— Terms 334
—— *White Labour in Tom Johnson Reply to Discussion* 14
Miscellaneous Abstracts 36, 77, 155, 193, 227 267
301, 336. 415
Misfires 153
Moir, Dr J.—on Annual Report 427
—— on Analyses of Gases from Burning Nitro-Glycerine Explosives... 92. 210
—— on the Assay of Cyanide Solutions and Slime Residue carrying Dissolved Gold 211
—— on Election as President 428
—— on Endless Rope Haulage 201
—— on Experiments on the Assay of Acid Washes 211
—— on the Influence of Moist Air on Quicklime 19. 106
—— on Metallurgical and Mining Terms .. 197
—— on Notes on Corrosion 205
—— on Researches upon the Telluride Gold Ores of Cripple Creek .. 19, 183
—— on a Rotary Extractor for Precious Metals from Solutions 204
—— on Stope Measurements ... 18, 105, 183
—— *Suggestions for a New Atomic Theory Reply to Discussion* 96
—— *Test for Prussic Acid* ... 342, 381
—— *and Gray, Jas.—The Destruction of Cyanide* 433
Discussion 442
Modern Cyanide Practice and Machinery A. Selwyn Brown .. 324

PAGE
Modern Mining Practice G. M. Bailes (Review) 338
—— Practice in Mining R. A. S. Redmayne (Review) 336
—— Progress in Mining and Metallurgy in the Western United States D. W. Brunton 263
Moisture in Mine Air 191
Molybdenum, Chromium, and Nickel in Steels, Separation of Vanadium. E. Pozzi-Escot 468
—— Estimation of, in Molybdenum Glance E. Collett & M. Eckhardt ... 321
—— Simple Tests for 269
Monazite D. B. Sterrett 154
Monel Metal 28
Moodie's G. M. & Ex. Co. ... 125, 128
Moody, G. T., and Leyson, L. T.—Solubility of Lime in Water 292
Morrison High Speed Stamp ... 218, 219, 251
Mortar Boxes 262
Mount Morgan (Barberton) Mine... ... 125
Mountain, M. B.—Electric Haulage and Winding Plants 34
Murdoch, R.—on the Small Cyanide Plant as erected and worked in Rhodesia ... 207
Murray, M. Thornton.—Notes on Corrosion, with Special Reference to the Corrosion of Steel Winding Ropes 54
Discussion 59, 204
do Reply to 205
Murex Process of Concentration 28

N

Natural History of the Igneous Rocks A. Harker. (Review) 270
Nelson, R.—Electricity in Coal Mines ... 36
New Applications of Electro-Metallurgical Alloys. A. D. Jouve 70
—— Concrete. F. J. Bosse 36
—— Method for Determining Antimony. M. F. Coolbaugh and J. O. Betterton 147
—— Method of Gold Mining 378
—— Nickel Copper Alloy. D. H. Browne 28
Newton, S.—on returning thanks for Scrutineers 429
Nichols, H. G.—Treatment of Slime ... 409
—— Welcome to 276
Nickel Copper Alloy, New D. H. Browne 28
—— Determination of Prettner ... 72
—— —— in presence of Cobalt. J. A. Sanchez 258
—— Estimation and Separation of, Use of Dicyandiamidine for the H. Grossman and W. Heilbron ... 321
—— in Nickel Steel by means of Dimethylglyoxime. Estimation of A. Iwanicki 293
—— in Steels, Separation of Vanadium, Molybdenum, Chromium and E. Pozzi-Escot 468
Nissen Stamp, the 259
Nitrates, Simple Tests for 269
Nitre 145
Nitric Oxide in Explosive Gases 90
Nitrogen, Atmospheric, Fixation of, in Cyanide Manufacture. W. Bettel ... 185
Nitro-Glycerine Explosives, Analyses of Gases from Burning Wm. Cullen and D. W. Greig 90
Discussion 91, 210
do Reply to 241
Nitroglycerine and Guncotton, Significance of the Abel Heat Test of R. Robertson and B. J. Smart 373
Non-cracking Glass 228
Note on the Efficacy of the Cyanide Antidote A. L. Edwards 82
Notes on Classifiers S. Aimetti ... 20
—— *Corrosion, with special reference to the Corrosion of Steel Winding Ropes M. Thornton Murray* 54
Discussion 59, 204
do Reply to 205
—— Different Methods of Stoping J. Cheflirs 301
—— the Iodide Method for Copper Assays R. T. D. Williams 71
—— *Precipitation Mather Smith Reply to Discussion* 62
—— *the Precipitating Effects of Substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing Gold and Silver A. J. Clark and W. J. Sharwood* ... 234
Discussion 405
—— *the Use of Air Lift Pumps for Underground Work B. C. T. Solly* ... 2
Nugel.—Recent Progress in the Cyanidation of Gold 258

O

Oats, Analysis of Colonial J. Lewis ... 301
Obituary Notices in Annual Report:— ... 423
—— Bauerman, Hilary ... 234, 256
—— Bradford, Wager 2, 21
—— Davies, John 69
—— Smith, Walter Herbert 115
O'Hara, J. D.—The Treatment of Accumulated Slime, and the Use of Filter Presses for Clarifying Slime Solution and By-Products 342
Discussion 403
Oil Fuel, The Smelting of Gold Precipitates and Bullion with A. Yates Discussion 144
Reply to Discussion 393
Olsen & Weiffenbach.—Case Hardening of Steel with Gases 416
Order of Fire and Misfires 153
Ore Concentration, Calculations in Dr. R. Gahl 187
Ore Deposits of South Africa J. P. Johnson (Review) 338
—— Dressing R. H. Richards (Review)... 304
—— —— in the United States and Mexico H. A. Guess 259
—— *Slime, Treatment of A. F. Crosse* ... 172
Discussion 175, 254
do Reply to 318
—— Treated, Consumption of Materials per ton of 330
Ores, Concentration of, by Wet Methods ... 28
—— Dressed Tin, Assay of L. Parry .. 258
—— Graphitic, Cyaniding 23
—— *Manganese Silver, of Mexico, Cyanidation of E. M. Hamilton* 65
Oxidising Accelerators... 108
Oxy-Hydro Smelting Process 188
Oxyhaemoglobin, Chemical Methods for Distinguishing Carbon Monoxide-Haemoglobin from K. Bürker ... 292
Ozone Manufacture 72

PAGE

P

Pachuca Tank ... 325
Packwalls and Pigsties ... 277
Palmer, L. A.—Points on Adjusting Instruments 227
Pam, E. Sand Filling on the Witwatersrand 429
Discussion 432
Panajotow, G.—Separation of Antimony and Tin... ... ... 116, 294
Pannings, Gold, The Use of Standards in Reading A. F. Crosse... ... 117
Parry, C. F.— on Further Notes on Rand Mining 282
—— L —Assay of Dressed Tin Ores ... 258
Patent Applications, Abstracts of 196, 231, 271, 306, 339, 379, 419
—— —— Selected Transvaal 38, 80, 120, 160, 196, 229, 272, 307, 339, 380, 419, 475
Pearce, S. H.—on the Development of Heavy Gravitation Stamps 239
—— on Precipitation from Cyanide Solutions by Zinc Shavings and Dust ... 206
—— on Tube Mill Circuit and Classification 288
Peat Bogs and Peat Industry of Canada (Review) 158
Pelabon, H —Fusibility of Mixtures of Gold and Tellurium... 145
Permanent Expansion of Cast Iron under Moderate Temperature 406
Permanganate, Changes in Dilute Solutions of J. E. Purvis 292
Permanganates, Alkaline 443
Phillips, Lionel, at Annual Dinner ... 467
—— *J. Moyle.—The Pilgrims Rest Gold Fields and Mining Methods Discussion* 16
Reply to Discussion 139
Philp, H.—Mining at the Redjang-Lebong Gold-Silver Mine, Sumatra ... 315
Discussion 318
Phosphates, Simple Tests for 270
Phosphor Bronze, Analysis of Hudson and Law 468
Pillars in Stopes 279
Pilgrims Rest Gold Fields and Mining Methods J. Moyle Phillips Discussion .. 16
Reply to Discussion 139
Pioneering F. Shelford (Review) ... 119
Plant, Cost of, in Barberton District ... 131
Plates, Amalgamating, The Position of, in the Stamp Mill 186
—— and Amalgamation 262
—— Battery, Dressing of E. H. Martin... 187
—— —— Sealing of J. E. Breakell .. 151
—— *Dressings of, The Effect on Amalgamation of different intervals of time between the G. O. Smart Discussion* ... 141
Reply to Discussion 177
Platinum, Soldering 302
Pneumatic Stamps 221
Points on Adjusting Instruments. L. A. Palmer 227
Poisoning, Cyanide 36
Pooler, F. J.—on Some Experiments on Smelting Titaniferous Iron Ore ... 290
Poppenberg, O., and Stephan, E.—Heat of Combustion and Explosion Temperature of Explosives 321
Position of Amalgamating Plates in the Stamp Mill 186
Potassium Cyanide, Electro-Chemistry of Solution of Gold in Ekeley and Tatum 21
Practical Notes on Dry Crushing Mills W. M. von Bernewitz 222
Precious Metals, The Dr. T. K. Rose (Review) 78
Precipitant, Charcoal as a, Literature relating to 237
—— Decaying Wood as a 236
Precipitants, Experiments on the Relative Activity of Some 234
Precipitate, White, found in Zinc Boxes R. F. Coolidge 184
Precipitates and Bullion, Gold, The Smelting of, with Oil Fuel. A. Yates. Discussion 144
do Reply to 393
—— Refining of 61
Precipitating effects of Chips, Charcoal, etc., upon Gold and Silver bearing Solutions 235
—— *Effects of Substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing Gold and Silver, Notes on A. J. Clark and W. J. Sharwood* 234
Discussion 405
Precipitation of Arsenic by Hydrogen Sulphide L. L. de Koninck 69
—— and Clean-up in W. Australia ... 224
—— *from Cyanide Solutions by Zinc Shavings and Dust A. J. Clark* ...
Reply to Discussion 205
—— Electrolytic M. R. Lamb ... 29
—— *of Gold from Cyanide Solution by Carbon in Lime, Experiments on the E. H. Croghan* 391
Discussion 393
—— *Notes on Mather Smith Reply to Discussion* 62
—— Practice in America 297
—— *Zinc Dust, at Cerro Prieto R. Linton Reply to Discussion* 60
Preece Test for Galvanised Iron 213
Prehistoric Rhodesia R. N. Hall. (Review) 157
Presentation to Lord Selborne 341
President's (A. McA. Johnston) Inaugural Address 3
President, The (A. McA. Johnston), Remarks by 2, 3, 10, 14, 43, 59, 62, 82, 85, 87, 122, 134, 138, 139, 141, 142, 143, 144, 274, 276, 310, 311, 312, 315, 341, 342, 345, 381, 393, 422, 427, 428, 449, 461, 468
Prettner.—Determination of Nickel . 72
Price, Sir Thomas.— on Inaugural Address ... 10
Principles of Mining H. C. Hoover. (Review) 119
Proceedings at Ordinary General Meetings—
July 17, 1909 1
Aug. 21, 1909 41
Sept. 18, 1909 81
Oct. 16, 1909 121
Nov. 20, 1909 161
Dec. 20, 1909 197
Jan. 15, 1910 233
Feb. 26, 1910 273
Mar. 19, 1910 309
April 16, 1910 341
May 21, 1910 381
(Annual) June 18, 1910 421
Props in Mines 278
Prospecting in Barberton District ... 127

PAGE

Protecting Steel from Corrosion. R. B. Woodworth 336

Prussic Acid, Test for Dr. J. Moir 342, 381

Pulp Settling 223

— — and Tailing, Measurement of W. J. Sharwood 469

Pumping 264

Pumps, Air Lift, for Underground Work, Notes on the use of B. C. T. Solly 2

— — Capacity of Sinking 226

Purification of Mercury J. H. Hildebrand 224

— — of Water Supplies by the use of Hypochlorates W. P. Mason ... 468

Purvis, J. E.—Changes in Dilute Solutions of Permanganate 292

Q

Quartz, Combustion Tubes. A. G. Levy ... 70

— — Sample, Method of Estimating Weight of Gold in a 27

Quicklime, the Influence of Moist Air on James Gray Discussion 19, 62, 106, 144

Reply to Discussion 176

Quicksilver, Simple Tests for 268

R

Rand, The Distribution of the Gold Produced on the Dr. T. K. Rose Reply to Discussion 139

— — *E. T., and Lloyd, W. D'A.—A Rotary Extractor for Precious Metals from Solutions* 201

Discussion 255

do Reply to 373

— — Metallurgical Practice E. J. Way ... 408

— — *Mining, Further Notes on Tom Johnson* 276

Discussion 319, 394

do Reply to 449

— — Valuation of Mining Areas on the W. Fischer Wilkinson 31

Rapid Analysis of Aluminium K. Abrecht 147

— — *Economical Method of Making Silver Amalgam E. H. Croghan* ... 42

— — Method for the Estimation of Vanadium in Ores and Technical Products P. y. Alvarez 321

— — Methods for the Chemical Analysis of Special Steels, Steelmaking Alloys and Graphite C. M. Johnson (Review) 229

— — Volumetric Method for Determining Cerium in presence of other Rare Earths F. J Metzger 294

Recent Discoveries of Graphite in British African Colonies 147

— — Progress in Cyanidation of Gold Mr. Nugel 258

Record Driving at the Modderfontein "B" G. M. Co. 227

Recovery of Zinc from Solutions of Sulphate, A Method for the Wm. Cullen and G. F. Ayers 87

Discussion 143, 209

do Reply to ... 240

Redmayne, R. A. S.—Modern Practice in Mining (Review) 336

Redjang-Lebong Gold-Silver Mine, Sumatra, Mining at the H. Philp ... 315

Discussion 318

Reduction of Ore in Barberton District ... 130

Reefs, Gold in L. de Launay 30

PAGE

Refining of Gold, Electrolytic Dr. T. K. Rose 148

Refractory Ores, The Treatment of ... 188

Relation between Rock Temperature and Air Temperature in Mines 226

Remaux, M.—Dust Collector for Rock Drills 191

Report of Council, Annual 422

Researches upon the Telluride Gold Ores of Cripple Creek (Colo.) Edited by T. B. Crowe Discussion 19, 107

Correspondence 181

Reply to Discussion 207

Residues, Disposal of, in W. Australia ... 223

— — Silver, Fluxes for.. 24

Resuscitation, Methods of 303

Revenue Account 425

Reviews and New Books 37, 78, 119, 157, 195, 229, 270, 304, 336, 379, 418, 475

Revolving Screens, Water required for ... 23

Reyersbach, L.—Large Units in Mining ... 77

Rhodesia, A Sketch of the Small Cyanide Plant as erected and worked in F. J. Thomas 82

Discussion 207

do Reply to 394

— — *Small Mines of B. I. Collings Reply to Discussion* 140

Ribbon, Metal 187

Richards, R. H.—Text Book of Ore Dressing (Review) 304, 337

Richardson, A.—The Barberton Goldfield ... 122

Discussion . .. 179, 210

do Reply to 288

— — on Further Notes on Rand Mining 282, 396

Rill Sytem of Stoping 301

Roake, H. G.—on Further Notes on Rand Mining 319

Roasting for Blast Furnace Smelting ... 266

Robertson, G. A.—on the Tube Mill Circuit and Classification 400

— — R., and Smart, B. J.—Significance of the Abel Heat Test of Guncotton and Nitroglycerine 373

Robinson Mine, Hydraulic Stope Filling at the 300

Rock Drills, Dust Collector for M. Remaux 191

Rogers, A. W., and du Toit, A. L.—The Geology of Cape Colony (Review) ... 337

Rohland, — Derusting of Iron in Ferro-Concrete 23

Rolfe, F. P.—Shrinkage Stoping in W. Australia 30

Roofs of Working Places in Mines ... 276

Rope Cappings and Attachment to Drum G. H. Winstanley 227

— — *Haulage, Endless H. G. Kay* ... 198

Discussion ... 254, 291, 319. 404

do Reply to 457

Ropes, Steel Winding, Notes on Corrosion with Special Reference to the Corrosion of M. Thornton Murray 54

Discussion 59, 204

do Reply to 205

Ropes, Winding, Strains on B. Woodworth 30

— — Wire, Some Notes on Testing R. Hunter 193

Rose, Dr. T. K.—Cupellation Losses ... 146

— — *The Distribution of the Gold Produced on the Rand Reply to Discussion* ... 139

— — Electrolytic Refining of Gold ... 148

— — The Precious Metals (Review) ... 78

Rosetta Sheba (Barberton) Mine 125

PAGE

Rotary Extractor for Precious Metals from Solutions *W. D'A. Lloyd & E. T. Rand* 201
Discussion 255
do Reply to 373
Routine Assaying on a Westralian Mine *W. B. Blyth* *Reply to Discussion*... 95
Rowland, Fred. (Secretary), Remarks by 2, 41, 81, 82, 121, 122, 161, 197, 233, 273, 309, 319, 341, 381, 422, 428
Rupp, E.—Standardization of Acids and Alkalies 116
Rust, Cause and Prevention of 23

S

Saladin Double-Mirror Galvanometer ... 25
Salkinson, A.—on the Precipitation of Gold from Cyanide Solution, etc. ... 393
Sampler for Residues, Dip 404
Sampling 265
—— Device, Mine 192
Sanchez, J. A.—The Determination of Nickel in Presence of Cobalt 258
Sand Bin for Sand Filling 430
—— *for Cyaniding, Continuous Collection of,* *W. A. Caldecott* 43
Discussion 54, 142, 179
do Reply to 238
—— Filling, Advantages of 431
—— —— *on the Witwatersrand* *E. Pam*... 429
Discussion 432
—— Filter Table Costs 238
—— —— Tables 44
—— Gradings of, at Simmer Deep, Ltd. ... 239
—— Sampler, Leupold... 471
—— Tonnage of, Fed to Tube Mill, Formula for Determining 456
—— in Vats, Tonnage of 470
Sands, Black, The Treatment of *Fraser Alexander* 174
Discussion 175
Saner, C. B.—at Annual Dinner 466
—— on an Accident to Mr. G. T. Holloway... 234
—— on Corresponding Members of Council for Rhodesia 234
—— on the Death of Prof. Hilary Bauerman 234
—— on Election of New Council... ... 428
—— on Inaugural Meeting of S.A. Institute of Electrical Engineers 233
—— on a Method for the Recovery of Zinc from Solutions of Sulphate ... 89
—— on Pilgrims Rest Gold Fields .. 17
—— on Precipitating Effects of Substances Containing Carbon 238
—— on Sand Filling on the Witwatersrand 432, 433
—— on Some Experiments in Smelting Titaniferous Iron Ores 351
—— on Vote of Thanks to Retiring President 459
Sartorius, K.—on Sand Filling on the Witwatersrand 433
Saving Fine Gold by Adhesion J. D. Luttrell 28
Sealing of Battery Plates J. E. Breakell . 151
Schafer Method of Resuscitation 303
Scheelite, Test for 116
Schmitt, C. O.—The Computation of Crushing Efficiency of Fine Grinding Machines 374
—— on the Continuous Collection of Sand for Cyaniding 180
—— on the Development of Heavy Gravitation Stamps 243

PAGE

Schmitt, C. O.—on the Treatment of Ore Slime 254
—— on the Tube Mill Circuit and Classification 397
Scott, E. K.—Electrically-Driven Winding Engines 300
Screen Aperture and Tonnages 101
—— Tests 187
—— —— Results of 25
Screening. Wire, Relation Between Diameters of Aperture and Wire, Screen Area, etc.... 384
Screens and Discharge Area 262
—— Laboratory E. T. Henderson .. 73
—— Standard Grades and 383
—— Standardization of 275
Seamon, W. H.—Direct Volumetric Method for the Estimation of Alumina ... 293
Secretary (Fred. Rowland), Remarks by 2, 41, 81, 82, 121, 122, 161, 197, 233, 273, 309, 319, 341, 381, 422, 428
Selborne, Lord, Presentation to 341
Selective Electric Fuse Spitting Device R. N. Bell 153
Separation of Antimony and Tin G. Panajotow 116, 294
—— of Vanadium, Molybdenum, Chromium, and Nickel in Steels E. Pozzi-Escot 468
Settlement, Acceleration of 408
—— of Slime, Theory of the H. E. Ashley 149
Shaft, Sinking a, by the Cement Process Dobbelstein 227
—— —— Estimated Cost of 412
—— Tracks, Concrete for 415
Shafts, Crushing Incline 279
Shaking Chutes 281
Sharwood, Dr. W. J.—Measurement of Pulp and Tailing 469
—— on Researches upon the Telluride of Gold Ores of Cripple Creek (Colo.) ... 181
—— *W. J., and Clark, A. J.—Notes on the Precipitating Effects of Substances Containing Various Forms of Carbon and Cellulose on Cyanide Solutions Containing Gold and Silver* ... 234
Discussion 405
Shaw, S. F.—American Cyanide Practice .. 328
Sheba Mine 125
Shelford F.—Pioneering (Review) . 119
Shockley, W. H.—on the Development of Heavy Gravitation Stamps .. 221
Shoes and Dies 262
Shrinkage Stoping in W. Australia . 30
—— System of Stoping ... 301
Signalling and Lighting, Mine ... 264
—— in Mines, Electric C. E. Grayson 333
Significance of the Abel Heat Test of Guncotton and Nitroglycerine R. Robertson and B. J. Smart .. 373
Silica 145
—— Combustion Tubes 70
—— Fusions for True J. Herman ... 320
Silicified Timber 35
Silicons, Ferro- 70
Silver Residues, Fluxes for 24
Simmer Deep, Ltd., Crushing Tests at . . 375
Simple Identification Test for Iron and Steel M. Bermann 326
—— Mineral Tests and How to Make Them G. Surr 267
—— Safe Fuse Lighter 413
Simplon Tunnel, Temperatures in ... 118

PAGE

Sinking a Shaft by the Cement Process Dobbelstein 227
—— Shaft, Estimated Cost of 412
Sketch of the Small Cyanide Plant as Erected and Worked in Rhodesia F. J. Thomas 82
Discussion 207
do Reply to 394
Slaughter, W. F.—Acetylene Lamps Underground 152
Slime, Accumulated, the Treatment of, and the Use of Filter Presses for Clarifying Slime Solution and By-Products J. D. O'Hara 342
Discussion 403
—— Collected in Filter Presses 472
—— Cyanide Treatment of L. M. Green and L. Mennell 322
—— Dry, Tons of, per foot Depth of Pulp ... 407
—— Filter, The Hunt Continuous ... 295
—— Gold, Filtering of E. Jensen ... 25
—— Pulp, Treating 135
—— *Residue, The Assay of Cyanide Solutions and, carrying Dissolved Gold A. Whitby* 134
Discussion 211, 288
do Reply to 289
—— Settlement 24
—— —— of, Theory of the H. E. Ashley 149
—— *Smelting Gold and Silver A. L. Blomfield* 406
—— *Solution and By-Products, the Use of Filter Presses for Clarifying, The Treatment of Accumulated Slime and J. D. O'Hara* 342
Discussion 403
—— Treatment of H. G. Nichols ... 408
—— —— Dilution System of 322
—— —— Dorr Thickeners in 323
—— —— *of Ore A. F. Crosse* .. 172
Discussion 175, 254
do Reply to 318
Slimes Assays, Elutriated 102
Small Cyanide Plant, A Sketch of the, as erected and worked in Rhodesia F. J. Thomas 82
Discussion 207
do Reply to 394
—— *Mines of Rhodesia B. I. Collings Reply to Discussion* 140
Smart, G. O.—The Effect on Amalgamation of different intervals of time between the Dressings of Plates Discussion ... 141
Reply to Discussion 177
—— *The Tube Mill Circuit and Classification* 282
Discussion 320, 397, 452
do Reply to 454
—— on a Rotary Extractor for Precious Metals from Solutions 204
—— B. J., and Robertson R.—Significance of the Abel Heat Test of Guncotton and Nitroglycerine 373
Smelting, Blast Furnace, Roasting for ... 266
—— Copper 266
—— *of Gold Precipitates and Bullion with Oil Fuel A. Yates Discussion* ... 144
Reply to Discussion 393
—— *Gold and Silver Slime A. L. Blomfield* 406
—— Process, The Oxy-Hydro 188
—— in Rhodesia 84

PAGE

Smelting Titaniferous Iron Ore, Some Experiments on
Part I., Crucible Experiments 162
Part II., Furnace Experiments 345
Prof. G. H. Stanley
Discussion 211, 253, 290, 351, 403
—— Zinc V. Lepiarczyk 259
—— —— The Chemistry of V. Lepiarczyk 184
Smith, E. A.—Assay of Industrial Gold Assays 469
—— *Mather—Notes on Precipitation Reply to Discussion* 62
—— S. W.—Behaviour of Tellurium in Assaying 146
—— Walter Herbert (Obituary) 115
Snake Bites 416
Society's *Journal*, The, Vol. IX. 122
Soda 145
Sodium Peroxide and Sodium Cyanide, Manufacture of C. F. Carrier ... 115
Soldering Platinum 302
Solly, B. C. T.—Notes on the Use of Air Lift Pumps for Underground Work ... 2
—— on Stope Measurements 212
Solubility of Lime in Water G. T. Moody and L. T. Leyson 292
Solution of Cassiterite in Tin Ores, References to the Methods of Effecting ... 314
—— of a Diamond Drilling Problem C. M. Haight 192
—— of Gold in Cyanide 228
—— —— in Potassium Cyanide, Electro-Chemistry of Ekeley and Tatum 21
Solutions, Bisulphate, Specific Gravity of Thos. Donaldson 86
—— of Permanganate, Changes in Dilute J. E. Purvis 292
—— *A Rotary Extractor for Precious Metals from W. D'A. Lloyd and E. T. Rand* 201
Discussion 255
do Reply to 373
—— *of Sulphate, A Method for the Recovery of Zinc from Wm. Cullen and G. F. Ayers* 87
Discussion 143, 209
do Reply to 240
Solvent Effects of Ferric and Cupric Salt Solutions on Gold W. J. McCaughey 468
Some Experiments on Smelting Titaniferous Iron Ore
Part I., Crucible Experiments ... 162
Part II., Furnace Experiments . 342
Prof. G. H. Stanley
Discussion 211, 253, 290, 351, 403
—— Notes on Testing Wire Ropes R. Hunter 193
South African Coals and Their Economics A. J. Andrews Discussion ... 92
Reply to Discussion 95
Spark Test for Iron and Steel 326
Specific Gravity of Bisulphate Solutions Thos. Donaldson 86
Speed and Drop in Mills 262
Stadler, H.—Grading Analyses and Their Application 382
Discussion 390, 457
Stamp Duty 261
—— Heads, Long 299
—— Mill, Construction and Operation of the G. H. Fison 261
—— —— Gravity, Evolution of the A. del Mar 117

PAGE
Stamp Mill, Vibration in a 261
—— Mills, Bolts in 261
—— —— Framing of 261
Stamps, Heavy Gravitation, The Development of W. A. Caldecott 108
Discussion 144, 178, 215, 241, 331, 353
do Reply to 358
—— Heavy, Advantages of 115
—— —— Proportion between Weights of Parts of 332
—— Order of Drop of 178
—— and Pulverisers, Water required for ... 23
—— and Tube Mills, Cumulative Percentage Curves for 102
—— Weight of 110, 113
Standardization of Acids and Alkalies E. Rupp 116
Standards, The Use of, in Reading Gold Pannings A. F. Crosse ... 117
Stanley, Prof. G. H.—on Election as Vice-President 429
—— on Experiments on the Assay of Acid Washes 139
—— on the I.M.M. Laboratory Screens .. 276
—— on a Method for the Recovery of Zinc from Solutions of Sulphate 89, 143
—— *Some Experiments on Smelting Titaniferous Iron Ore*
Part I. Crucible Experiments .. 162
Part II. Furnace Experiments ... 345
Discussion ... 211, 253, 290, 351 403
—— W. N., and Bannister, C. O.—Thermal Properties of Cupels 145
Steel, A. A.—Crucible Assays 145
—— Carbon in, Determination of ... 70
—— Case Hardening of, with Gases Olsen and Weitlenbach 416
—— Chemical Analysis of 9
—— Cost of Cast 188
—— *Iron and, The Crystallization of A. McA. Johnston* 3
—— and Iron, Simple Identification Test for M. Bermann 326
—— Nickel, Estimation of Nickel in, by means of Dimethylglyoxime A. Iwanicki 293
—— Production, Transvaal Iron Ores and R. N. Kotzé 194
—— Protecting, from Corrosion R. B. Woodworth 336
—— Reinforced, Corrosion of E. R. Matthews 228
—— *Winding Ropes, Notes on Corrosion, with Special Reference to the Corrosion of M. Thornton Murray* 54
Discussion 59, 204
do Reply to 205
Steels, Separation of Vanadium, Molybdenum, Chromium and Nickel in E. Pozzi-Escot 168
Stephan, E., and Poppenberg, O. Heat of Combustion and Explosion Temperature of Explosives 321
Sterrett, D. B.—Monazite 154
Stevens, Thos. B.—on Reseaches upon the Telluride Gold Ores of Cripple Creek (Colo.) 107
Stokes, R.—Mine Valuation 183
Stope Measurements. O. Tonnesen Discussion ... 18, 63, 115, 140
Correspondence ... 183, 212
Reply to Discussion ... 369, 405
Stope Filling, Hydraulic, at the Robinson Mine 300
Stopes, Pillars in 279
—— ready for Sand Filling 431
Stoping, The "Flatback" System of ... 301
—— Notes on Different Methods of J. Cheffirs 301
—— Rill System of 301
—— The Shrinkage System of 301
—— Shrinkage, in W. Australia 30
Stopping Leaks under High Pressure W. O. Wood 333
Storms, W. H.—Timbering and Mining (Review) 195
Strains on Winding Ropes B. Woodworth 30
Students, Apprentices and Learners admitted as 424, 427
—— New 2, 42, 81, 121, 162, 197, 233, 273, 309, 341, 381, 422
Suggestion Relating to the Valuing of Gold Mines R. A. Wade 122
Correspondence 183
Suggestions for a New Atomic Theory Dr. J. Moir Reply to Discussion ... 96
Sulphate, A Method for the Recovery of Zinc from Solutions of Wm. Cullen and G. F. Ayers 87
Discussion 143, 209
do Reply to 240
Sulphates, Decomposition of, in an Air Current W. S. Landis ... 322
Sulphide, Estimation of, in Alkali Cyanide T. Ewan 293
Sulphur in Coal and Coke M. Holliger ... 69
Surface Mining 263
Surr, Gordon—Simple Mineral Tests and how to make them... 267
Survey Instruments, Points on Adjusting L. A. Palmer 227
Surveyors, An Improved Tape Reel for Engineers 155

T

Tailing and Pulp, Measurement of. W. J. Sharwood 469
Tamping 152
Tanks, Agitation, Details of, in America ... 330
—— Leaching, Details of in America . 330
—— Settling and Collecting, Details of, in America 329
Tape Reel for Engineers and Surveyors, An Improved 155
Tappets, Designs of 245
Fatum & Ekeley.—Electrochemistry of Solution of Gold in Potassium Cyanide ... 21
Telescope Level, To adjust the 228
Telluride Gold Ores of Cripple Creek, Researches upon the T. B. Crowe. Discussion 19, 107
Correspondence 179
Reply to Discussion 207
—— Gold Ores, Treatment of, at Kalgoorlie 20
Tellurides 304
—— Metallic. C. A. Tibbals 147
Tellurium, Behaviour of, in Assaying. S. W. Smith 146
—— and Gold, Fusibility of Mixtures of H. Pelabon 145
—— Simple Tests for 269
Temperature Conversion Table. Dr. L. Waldo 417
—— of the Earth's Crust 414

PAGE
Temperature Explosion, of Explosives, Heat of Combustion and O. Poppenberg and E. Stephan 321
—— Gradient 192
—— at Great Depths in Coal Mines ... 118
—— Moderate, Permanent Expansion of Cast Iron under 406
—— in Mines, Relation between Rock Temperature and Air 226
Temperatures during Cupellation R. H. Bradford 330
—— Low, Light Reactions at Sir James Dewar 374
—— Moisture of Air in various 191
—— in Simplon Tunnel 118
Test of Large Reinforced Concrete Beams ... 157
—— *for Prussic Acid. Dr. J. Moir* 342, 381
—— for Scheelite 116
Testing of Galvanised and other Zinc-coated Metals Dr. W. H. Walker ... 213
—— Wire Ropes, Some Notes on R. Hunter 193
Tests for Cupellation. H. A. White ... 234
Text Book of Ore Dressing. R. H. Richards. (Review) 337
Theory of the Settlement of Slime. H. E. Ashley 149
—— of Valancy, The J. N. Friend (Review) 78
Thermal Analysfs of Intermetallic Compounds 25
—— Properties of Cupels C. O. Bannister and W. N. Stanley . . 145
Thomas, A.—on the Smelting of Gold Precipitates and Bullion with Oil Fuel... 144
—— *F. J.—A Sketch of the Small Cyanide Plant as erected and worked in Rhodesia* 82
Discussion 207
do Reply to 394
—— Jas. E.—on Endless Rope Haulage .. 319
—— —— on the Precipitating Effects of Substances containing Carbon ... 238
—— —— on the Tube Mill Circuit and Classification 452
Three Sisters (Barberton) Mine 126
Tibbals, C. A. Metallic Tellurides .. 147
Tilting Furnace, The 297
Timber, Silicified 35
Timbering 264
—— and Mining. W. H. Storms. (Review) 195
Tin and Antimony, Separation of G. Panajotow 116
—— *Ores, Assay of Jas. Gray* .. 312
Discussion 402
—— —— of Dressed L. Parry ... 258
—— —— References to the Methods for Effecting Solution of Cassiterite in ... 314
—— —— Wet Assay of J. J. Beringer .. 376
—— Separation of Antimony from G. Panajotow 294
—— Simple Tests for 268
Titaniferous Iron Ore, Analysis of ... 164
—— —— —— *Smelting, Some Experiments on*
Part I., Crucible Experiments .. 162
Part II., Furnace Experiments ... 345
Prof. G. H. Stanley
Discussion ... 211, 253, 290, 351, 403
Titanium, Volumetric Method for the Estimation of, and of Titanium and Iron in the Presence of Each Other E. Hibbert 293
Tonnage of Sand in Vats 470

PAGE
Tonnesen, O.—Stope Measurements
Discussion 18, 63, 105, 140
Correspondence ... 183, 212
Reply to Discussion ... 369, 405
—— K.—on Stope Measurements ... 140
Tramming, Underground 264
Transit, To Adjust the Standards of the ... 227
Transvaal, Chromite Ore from the ... 258
—— Iron Ores and Steel Production R. N. Kotzé 194
Treatment of Accumulated Slime, and the Use of Filter Presses for Clarifying Slime Solution and By-Products J. D. O'Hara 342
Discussion 403
—— *of Black Sands, The Fraser Alexander* 174
Discussion 175
—— *of Ore Slime A. F. Crosse* ... 172
Discussion 175, 254
do Reply to 318
—— of Ores of Precious Metals 300
—— of Slime H. G. Nicholls 408
Trevor, T. G.—on the Barberton Goldfield ... 179
Tributing 472
Truck Tipper, Mine 199
Truran, W.—on the Development of Heavy Gravitation Stamps 220
Truscott, S. J.—on the Development of Heavy Gravitation Stamps 215
Truscottite 316
Tube Mill and Conicle Mill, Comparison of Results with 27
—— —— *Circuit and Classification G. O. Smart* 282
Discussion 320, 397, 452
do Reply to 454
—— —— Feeder 397, 454
—— —— Formula for Determining Tonnage of Sand Fed to 456
—— —— Liners F. C. Brown ... 151
—— —— —— Brown's 297
—— —— Practice in America 296
—— —— —— in Mexico 226
—— Milling, Classification of Mill Pulp for... 20
—— —— Costs 187
—— —— at El Oro H. E. West ... 186
—— Mills, Comparisons of Costs, With and Without 355
—— —— Details of Modern American ... 329
—— —— and Stamps, Cumulative Percentage Curves for 102
Tunnelling Record in California J. W. Henderson 192
Tungsten, Simple Tests for 268

U

Underground, Acetylene Lamps W. F. Slaughter 152
—— Tramming 264
—— Work, Cost of 155
—— —— *Notes on the Use of Air Lift Pumps for B. C. T. Solly* 2
United States and Mexico, Ore Dressing in the H. A. Guess 259
—— —— Western, Modern Progress in Mining and Metallurgy in the D. W. Brunton 263
Urquhart, W. G.—on the Development of Heavy Gravitation Stamps ... 178
Use of Concrete for Mine Support Prof. W. R. Crane 36, 118

PAGE

Use of Dicyandiamidine for the Estimation of Nickel, and its separation from Cobalt, Iron, Chromium, Zinc, Manganese and Magnesium H. Grossman & W. Heilbron ... 321
— of Standards in Reading Gold Pannings A. F. Crosse 117

V

Valuation, Mine R. Stokes 183
—— of Mining Areas on the Rand W. Fischer Wilkinson 31
Valuing of Gold Mines, A Suggestion relating to the R. A. Wade 122
Correspondence 183
Van Nostrand's Chemical Annual (Review)... 337
Vanadium, Determination of H. F. Watts 294
—— Molybdenum, Chromium and Nickel in Steels, Separation of E. Pozzi-Escot 468
—— in Ores and Technical Products, Rapid Method for the Estimation of P. y. Alvarez 321
—— Simple Tests for 269
Vanners, Details of Tables and, in America ... 329
Vats, Circular, Capacity of, per foot of Depth W. A. Caldecott 407
—— Volume of 471
Vaughan, J. A.—on Some Experiments on Smelting Titaniferous Iron Ores ... 352
Ventilating System at the Comstock Mines, Nevada. Prof. G. J. Young ... 411
Ventilation, Mine 264
Vibration in a Stamp Mill 261
Viscosity of Water 149
Volumetric Method for the Estimation of Titanium and of Titanium and Iron in the presence of each other E. Hibbert 293

W

Wade, R. A.—A Suggestion relating to the Valuing of Gold Mines 122
Correspondence 183
Wagner, Dr. Ing. P. A.—Die Diamantführenden Gesteine Südafrikas. (Review) 305
Waldo, Dr. L.—Temperature Conversion Table 417
Walker, Dr. W. H.—The Testing of Galvanised and other Zinc-coated Metals ... 213
Wang, Chung Yu.—Antimony. (Review) ... 119
Wartenweiler, F.—on Notes on the Precipitating Effects of substances containing various forms of Carbon and Cellulose on Cyanide Solutions containing Gold and Silver 405
Warwick, A. W.—Committee on Fuel ... 298
Water required for Concentrating Machinery 23
—— Supplies, Purification of, by the Use of Hypochlorates. W. P. Mason ... 468
Watson, F. W.—on Bullion and Assay Values 312
—— Erratum 196
—— on Experiments on the Assay of Acid Washes 139
Watts, H. F.—Determination of Vanadium .. 294
Way, E. J.—on the Development of Heavy Gravitation Stamps. 352
—— Rand Metallurgical Practice ... 408
Webb, G. A.—on Endless Rope Haulage ... 291
Weight of Gold in a Quartz Sample, Method of Estimating 27
Weiffenbach and Olsen.—Case Hardening of Steel with Gases 416

PAGE

Weiskopf, Dr. E. H.—The Estimation of Carbon Monoxide in Mine Gas Reply to Discussion 11
West, H. E.—Tube Milling at El Oro 186
Western Australian Metallurgy. R. Hamilton 74
Weston, E. M.—on Further Notes on Rand Mining 397
Westralian Mine, Routine Assaying on a W. B. Blyth. Reply to Discussion 95
Wet Assay of Tin Ores. J. J. Beringer ... 376
—— Methods, Concentration of Ores by ... 28
Weymouth, H. H.—Electrically Driven Winding Engine 75
Whitby, A.—The Assay of Cyanide Solutions and Slime Residue carrying dissolved Gold 134
Discussion .. . 211, 288
do Reply to 289
—— on Experiments on the Assay of Acid Washes 138
—— on Test for Prussic Acid 381
White Labour in Mining Tom Johnson Reply to Discussion 14
—— Precipitate found in Zinc Boxes. R. F. Coolidge 184
—— H. A.—on the Assay of Cyanide Solutions, etc. 136
—— on Bullion and Assay Values 311, 312
—— on the Continuous Collection of Sand for Cyaniding 142
—— on the Destruction of Cyanide 442, 449
—— on Experiments on the Assay of Acid Washes 139
—— on Grading Analyses and their Application 390
—— on the Influence of Moist Air on Quicklime 63
—— on Metallurgical and Mining Terms ... 197
—— on Test for Prussic Acid 342
—— *Tests for Cupellation* 234
Wilmoth, L. J.—Experiments on the Assay of Acid Washes resulting from the Cyanide "Clean-up" by the use of Bisulphate 136
Discussion 138, 211
do Reply to 290
Erratum 196
Wilfley Tables, Water required for ... 23
Wilkinson, Prof. J. A.—on Analyses of Gases from Burning Nitro-Glycerine Explosives 92
—— on Bullion and Assay Values 311, 312
—— on Congratulations to Dr. W. A. Caldecott 276
—— on a Method for the Recovery of Zinc from Solutions of Sulphate ... 89
—— on Notes on Corrosion ... 59, 204
—— on Sand Filling on the Witwatersrand ... 432
—— W. Fischer.—on the Development of Heavy Gravitation Stamps ... 221
—— —— Valuation of Mining Areas on the Rand 31
Williams, John R.—on Continuous Collection of Sand for Cyaniding 54
—— R. T. D.—Notes on the Iodide Method for Copper Assays 71
Winding Engine, Electrically Driven H. H. Weymouth 75
—— Engines, Electrically Driven. E. K. Scott 300
—— Plants, Electric Haulage and M. B. Mountain 34

PAGE

Winding Ropes, Strains on B. Woodworth 30
Winstanley, G. H.—Rope Cappings and Attachment to Drum 227
Wire Ropes, Some Notes on Testing R. Hunter 193
Witwatersrand, Sand-Filling on the E. Pam 429
Discussion 432
Wood, G. R.—Electricity in Coal Mining ... 190
—— W. O.—Stopping Leaks under High Pressure 333
—— in a Cord? How much 302
Woodworth, R. B.—Aluminium 336
—— Protecting Steel from Corrosion ... 336
Worcester (Barberton) Mine 126
Working Costs in Barberton District ... 132

X

Xydis, C.—Handbook on Tacheometrical Surveying (Review) 157

Y

Yates, Prof. J.—on Continuous Collection of Sand for Cyaniding 54
—— *A.—Grading Assays and Grinding Efficiencies Correspondence* ... 67
Reply to Discussion 99

PAGE

Young, Prof. G. J.—Ventilating System at the Comstock Mines, Nevada ... 411

Z

Zinc Boxes, White Precipitate found in R. F. Coolidge 184
—— Coated Metals, The Testing of Galvanised and other Dr. W. H. Walker 213
—— *A Method for the Recovery of, from Solutions of Sulphate Wm. Cullen and G. F. Ayers* 87
Discussion 143, 209
do Reply to 240
—— *Dust Precipitation at Cerro Prieto R. Linton Reply to Discussion* ... 60
—— Electrolytic Precipitation of... ... 89
—— Precipitation, Combination of Electrolytic and 30
—— *Shavings and Dust, Precipitation from Cyanide Solutions by A. J. Clark Reply to Discussion* 205
—— Smelting V. Lepiarczyk 184
—— —— Chemistry of V. Lepiarczyk ... 259

3

TN
1
S66
v.10

South African Institute of
Mining and Metallurgy
Journal

~~Physical &
Applied Sci.
Serials~~

Engineering

PLEASE DO NOT REMOVE
CARDS OR SLIPS FROM THIS POCKET

UNIVERSITY OF TORONTO LIBRARY

ENGIN STORAGE